R-04-73

Äspö HRL

Experiences of blasting of the TASQ tunnel

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November 2004

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Summary

A new tunnel was developed at the Äspö Hard Rock Laboratory (AHRL) during the spring and summer 2003. The tunnel was specially designed for a rock mechanics experiment, the Äspö Pillar Stability Experiment (APSE). In this pillar experiment there was a high demand to initiate high in-situ stresses and therefore the tunnel was designed with a large height/width ratio and with a circular floor.

There were high requirements on bore hole precision and of a minimized EDZ (Excavation damaged zone) in the pillar area. This included a maximum borehole deviation of 10 mm/m, a maximum overbreak due to the lookout angle of 0.3 m and an EDZ of 0.3 m. To make a charge control feasible cartridged explosives was prescribed. The initiation was made with Nonel. The last three rounds used electronic initiation to enable studies of possibility to further reduce the EDZ.

The collar of the tunnel was very close to installations and shaft and it was very important to avoid fly-rock and vibrations. Special types of stemming were used as well as steel plates and rubber mats.

The excavation works was divided in three different phases. The first phase of the tunnel was an ordinary 26 m² tunnel. After approximately 30 m a ramp separated the tunnel section into a top heading and a bench, total 33 m². After the last top heading round was excavated and the roof had been reinforced with fibre reinforced shotcrete the bench was taken out with horizontal holes as the third phase.

The drilling precision was very good and 95% of all half-pipes fulfilled the demands. The total amount of visible half-pipes in the APSE-tunnel was high and indicated a successful smooth blasting.

The EDZ was examined further by cutting slots in the wall and roof. Existing cracks appear very clearly when a dye penetrant is sprayed on the cleaned surface. A typical crack pattern consists of blast cracks, induced cracks (cracks from the distressing caused by blasting) and natural cracks. The maximum crack length in the tunnel wall was roughly 0.2 m. However, longer cracks existed but these cracks probably originate from water filled blast holes. The cracks from the bench were generally much shorter than what is normally achieved during excavation of a horse-shoe shaped tunnel. This may be due to the stress situation under the arched bottom profile, the hight requirement on drilling precision also in the floor as well as the low confinement in the arch shaped contour of the floor:

Some conclusions from the APSE tunnel:

- There is a demand for new drilling equipment with a better guidance control to increase the drilling accuracy.
- Electronic detonators have very good accuracy and a high potential to reduce cracks from blasting. However, they must be more easy to use.
- It is possible to minimize the damage zone in the floor by using top heading and bench. However, there is a demand for more development in order to minimize the damage zone in the floor without a separate bench.
- Water in bore holes increases the damage zone in terms of lengh and frequency of induced fractures. This could be avoided by drilling the holes pointing slightly upwards.

- The look-out angle and distribution of specific charge along each round causes a discontinuous EDZ along the tunnel. It is therefore indicated that the impact of the EDZ on hydraulic conductivity along the tunnel has very limited impact.
- During similar conditions is it believed that the extent of the EDZ is manageable through D&B design and QA control during excavation.

Sammanfattning

Under våren och sommaren 2003 drevs en ca 70 m lång tunnel i Äspö-laboratoriet på 450-m nivån. Till följd av höga krav på skonsam sprängning blev drivningen speciell. Tunneln, som kallas APSE-tunneln, gjordes för att kunna studera pelarstabilitet mellan vertikalt borrade hål, 1,8 m i diameter och 6 m djupa. Man önskade höga spänningskoncentrationer i området mellan hålen och därför utformades tunneln med hög tvärsektion och en cirkulär sula.

I området där pelarförsöken skulle göras ställdes extra höga krav på hela konturen inklusive sula. Kravet innebar en hålavvikelse på högst 10 mm/m och en stickning och skadezon som ej fick överstiga 0,3 m. Hålen i tunneln laddades med patronerade laddningar och initierades huvudsakligen med Nonel. För att undersöka om skadezonen kunde reduceras ytterliggare designades de tre sista tunnelsalvorna för initiering med elektroniska sprängkapslar.

Tunnelpåslaget blev förlagt nära befintligt schakt varför höga krav ställdes på en begränsning av vibrationsnivåerna. För att skydda installationer förladdades hålen i de första salvorna med lerpluggar och dessutom användes olika typer av kastskydd.

Drivningen delades upp i olika delar. Den första delen var vanlig tunneldrivning av en 26 m^2 ort. Efter ca 30 m delades tunneln in i dels en topport och dels en pall med en total area av 33 m^2. Då topporten var färdigdriven förstärktes taket med fiberbetong varefter pallen togs ut med horisontell borrning och sprängning.

Borrningsprecisionen var mycket bra och 95 % av halvpiporna i väggarna klarade den uppsatta målsättningen (piporna i taket kunde inte mätas in pga. taket var betongsprutat). Andelen synliga halvpipor i APSE-tunneln var också mycket stort vilket normalt tolkas som en lyckad skonsam sprängning.

För att ytterliggare undersöka skador i berget undersöktes sprickorna i kvarstående berg i ett antal sågade sektioner. Sprickbilden i sektionerna bestod av sprängsprickor, inducerade sprickor (har uppkommit till följd av spänningsomlagringar efter sprängningen) samt naturliga sprickor. Sprängsprickorna i väggen hade en maximal spricklängd på ca 0,2 m. Dock fanns också ett mindre antal betydligt längre sprängsprickor. Förklaringen till detta är troligen på att hålen varit vattenfyllda vid sprängningen.

Sprickutbredningen från pallsprängningarna var generellt mer begränsad, med en maximal spricklängd på ca 0,1 m i sulan. Detta är betydligt kortare än de sprickor som brukar uppkomma vid sprängning av sulhålen vid normal tunneldrivning. Orsaken till detta kan bero på att man vid pallsprängning har en annan spänningssituation, att hålen varit rakare samt att salvan eventuellt har lättare att slå ut i och med den bågformade bottenprofilen.

Vilka slutsatser kan man dra av detta projekt?

- Ett system med högre precision än dagens rikt- och positioneringssystem behöver utvecklas.
- Elektroniska sprängkapslar har en mycket god precision och hög potential för att reducera sprickbildning i kvarstående berg. Dock behöver kapslarna göras med användarvänliga.

- Det är möjligt att uppnå högt ställda krav på skadezoner även i sulan. I APSE-tunneln lyckades detta genom att dela upp drivningen i en topport och en liggarpall. Utveckling av ny teknik för skonsam sprängning i sulan, utan uppdelning av topport och liggarpall, bör utvecklas.
- Vatten i borrhål ger betydligt fler och längre sprickor vid sprängning än motsvarande laddning i torra hål. Hål som borras svagt uppåt reducerar den risken.
- Stickning och fördelning av sprängämnet längs salvorna medför att skadezonen och dess hydrauliksa kornektivitet blir lokal och ej kontinuerlig i tunnelns längdled.
- Under liknande platsförhållanden anses det troligt att omfattningen av sprängskadezonen är möjlig att kontrollera dels genom design av sprängningen, dels genom kontrollplaner vid utförandet.

Contents

1 Purpose of the tunnel

A new tunnel was developed at the Äspö Hard Rock Laboratory (AHRL) during spring and summer 2003. The tunnel is located at the 450-m level close to the shaft, see Figure 1-1. The tunnel was specially designed for a rock mechanics experiment, the Äspö Pillar Stability Experiment (APSE) /Andersson, 2003/. The tunnel is therefore referred to as the APSE tunnel, whereas the correct ID in SKB database is the TASQ tunnel.

There were four major considerations for siting the experiment:

1. The tunnel shall be subject to as high in-situ stresses as possible achievable at the AHRL. This brought the siting of a the experiment down to the deeper part of the facility.
2. To achieve the highest possible stresses for the planed experiment the tunnel should be orientated perpendicular to the major principal stress, and the boundary stress conditions should be understood. This implemented the need for a new tunnel, sufficiently long to avoid secondary stresses from existing tunnels or the face of a new tunnel.
3. Available rock mechanics data from the AHRL shall be able to use as much as possible. The most valuable information came by that time from the Prototype Repository Experiment and from a test of three different stress measuring methods in two orthogonal boreholes /Jansson and Stigsson, 2002/. This brought the siting of a new tunnel to the western part of the 450-m level.
4. The impact on monitoring in ongoing experiments shall be as small as possible, especially was concern to not change the boundary conditions for some long term monitoring. This pointed at having the tunnel as far as possible away from the so called True Block Scale area /Winberg et al. 2002/. This lead to the actual location.

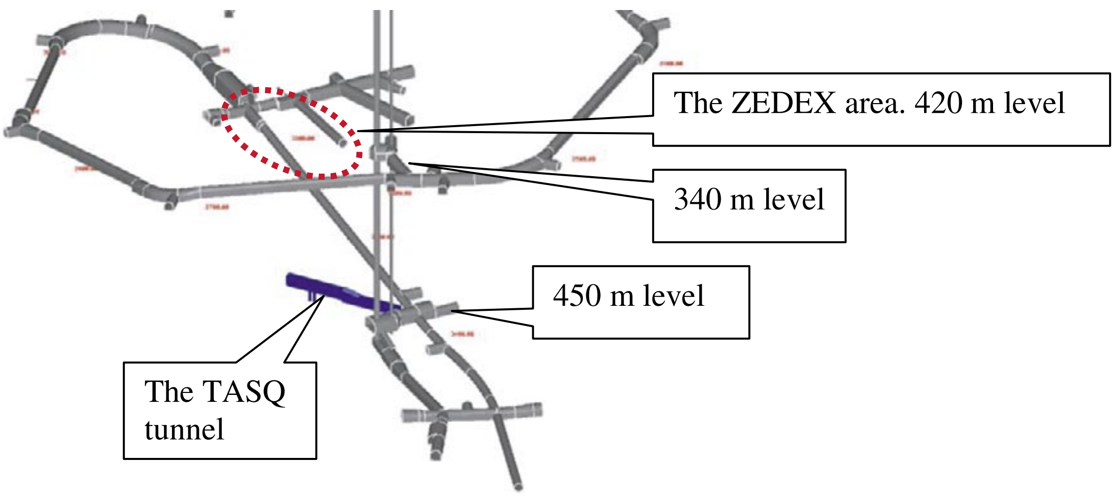


Figure 1-1. *Location of theTASQ tunnel.*

The significant need for a new tunnel at the AHRL for the planned rock mechanics experiment lead to a planning process for the construction works. Site investigations by drilling new cores /Andersson, 2003/ gave also additional detailed geological information. It was concluded that the tunnelling was going to experience a somewhat variable ground in terms of fracturing, lithological variation and water bearing fractures. None of the conditions were however unexpected for the conditions at the AHRL and had no significant impact on the possibilities for the construction works. It was however concluded that a tunnelling for approximately 70 m would provide some opportunities for studying the possibilities to control the development of an Excavation Damaged Zone (EDZ) caused by the Drill & Blast operations (D&B). The new tunnel was to be located only some 300 m away from the ZEDEX tunnel (Figure 1-1), where a study of the EDZ in a D&B respectively a TBM tunnel was carried out 1995 /Olsson and Reidarman, 1995/. The simple question was, if the current state-of-art for the D&B method, together with the actual requirement to have an extreme smooth blasting of the floor by taking the lower part of the cross section as a separate bench (see Chapter 3) could produce a less pronounced EDZ compared to the results 8 years earlier.

In addition, in studies primarily in quarries a large number of holes have been blasted and the cracks in the remaining rock have been examined /Olsson and Ouchterlony, 2003/. Coupling ratio, spacing, water in the holes, scatter in the initiation and the influence of different explosives on crack lengths were some of the examined factors used in a proposed new formula. The proposed formula shall be tested under various conditions. SKB provided such an opportunity by applying the new ideas for the inner three rounds of the top heading.

The purpose to also study the EDZ from the new tunnel lead to a discussion in the planning process for the excavations. Fruitful discussions with Skanska, the actual contractor, lead to practical measures to be covered by the Quality Plan for the contractor.

This report presents the actual D&B design, the actual outcome of the excavation works, including visible signs from the tunnel on possible damage caused by the excavations. Based on the records the report gives recommendations for locations on were to study the EDZ at a later stage. A parallel report /Nyberg et al. 2005/ presents the results of monitoring of vibrations during excavation of the tunnel.

2 Site conditions

2.1 Lithological composition of rocks

The dominating rock types in the Äspö area are the plutonic Äspö diorite and Ävrö granite. They belong to the postorogenic phase of the Transscandinavian Igneous Belt (TIB), their ages are c 1.8 Ga. The Äspö diorite is a medium-grained, grey to reddish grey rock. It is generally porphyritic, with K-feldspar megacrysts. Its composition does not correspond to a true diorite, but ranges from granite to granodiorite to quartz monzonite /Rhén et al. 1997; Wikman and Kornfält, 1995/. The more felsic Ävrö granite, which is a medium-grained, greyish red rock, is mainly classified as true granite. Transitions between these two rock types are gradual rather than sharp. Chemical relationships between them indicate that they can be considered as two varieties of the Småland granite /Wikman and Kornfält, 1995/. Subordinate rock types intruding the TIB-rocks are mafic rocks, pegmatites and fine-grained granites. To distinguish between diorite and granite through visual inspection is not easily done. Based on density logs from boreholes a density of > 2,700 kg/m^3, has been used to define the diorite /Rhén et al. 1997/.

The TASQ tunnel is dominated by different varieties of Äspö diorite. The "Äspö diorite" is a quarz-monzodiorite. The major rock volume consists of unaltered Äspö diorite, but relatively large volumes also consist of oxidized or sheared Äspö diorite. This is primarily associated with a ductile deformation zone that strikes in the tunnel orientation and dips approximately 45 degrees to SE. Other rock types present are mafic rocks, pegmatite and fine-grained granite. Rock contacts are generally diffuse, as they are successive transitions from one type of Äspö diorite to another. Contacts between dikes and host rock are sharp, but well healed. Regional metamorphism appears to be absent or of very low grade in the TASQ rock volume. Occasionally a diffuse foliation can be found in the Äspö diorite, which appears to be associated with the regional foliation pattern. Hydrothermal, low grade alteration appears to some extent in association with the old ductile deformation zone that strikes along the tunnel.

From the perspective of D&B operations the rock is not expected to have large heterogeneities in the location of the TASQ tunnel. The main difference in mineral composition between the Äspö diorite and the Småland granite is the content of quartz versus mafic minerals. The quartz and kalifeldspar content trend to decrease towards diorite as especially plagioclase and biotite increases, causing the somewhat higher density of the diorite, see Table 2-1. This may lead to that the diorite can be a little "tougher" for D&B, compared to the granite.

Table 2-1. Estimated mineralogical composition and plagioclase composition of rock types (QMD = quartz-monzodiorite).

Sample Number of samples	Äspö QMD 5	Altered ÄQMD 1	Ävrö granite 3	Fine-grained granite 2
Quartz	12	15	25	31
Plagioclase	40 (An 25–30%)	25 (An 0–5%)	28 (An 20–30%)	15 (An 20–25%)
K-feldspar	20	23	32	36
Biotite	18	0.5	7	3
Chlorite	0.3	14	1.0	0.5
Titanite	1.3	1.3	0.4	0.2
Amphibole	0.2	–	–	–
Epidote	4	10	2.5	3
Sericite	4	10	4	10.5
Opaques	0.6	0.4	0.3	1.0

2.2 Structures

Geological structures in TASQ are dominated by three main orientations, all of them related to the regional pattern. In Figure 2-1 the three main orientations are presented in stereographic projections. The figure also shows the orientation of the tunnel. The tunnel is aligned perpendicular to the major principal stress.

1. Set 1 in Figure 2-1 represents a steeply dipping set stiking in NNE-NE. Fracture filling material in this set is dominated by chlorite. This set is rarely associated with water seepage in the AHRL. An example is given in Figure 2-2.
2. Set 2 in Figure 2-1 is striking NW-SE and dipping sub vertically, i.e. perpendicular to the tunnel. It consists of relatively wide continuous brittle fractures and faults. This set is common in the whole Äspö HRL and is often associated with water seepage, both in the TASQ tunnel, and in the Äspö region in general. An example is given in Figure 2-3.
3. The third main structural set is a subhorizontal (set 3 in Figure 2-1). Fracture filling minerals in all these orientations are dominated by chlorite, epidote and calcite. This set is rarely associated with water seepage.

An oxidized, brittle-ductile shear zone strikes along the tunnel in 030–035 degrees, parallel to set no 1. It is dipping to the southeast and is present along the major part of the tunnel. In places, this set of structures generates critical surfaces of brittle reactivation with fracture filling of mainly epidote-chlorite. It is however mainly sealed.

The structures striking in the tunnel direction (set 1) may have some influence on the drilling precision. Especially the partly "softer" rock in the ductile shear zone may have had influence in some bore holes. But the area of the ductile zone is mall, compared to the tunnels cross section area, so this influence is believed to be insignificant.

Water leakage from the fracture set oriented in NW-SE/sub vertical was most pronounced in sections 0/060 and 0/064 of the tunnel. This area was subject grouting /Emmelin et al. 2004/. Generally the leakage causes damp surfaces or, at most, seepage. Occasionally, however leakage has been classified as flowing, in these cases exclusively generated in the above mentioned tunnel-perpendicular set. Two fractures of this set are intersected by deposition hole DQ0066G01. Calculations of total inflow in this deposition hole indicated 30 litres/minute.

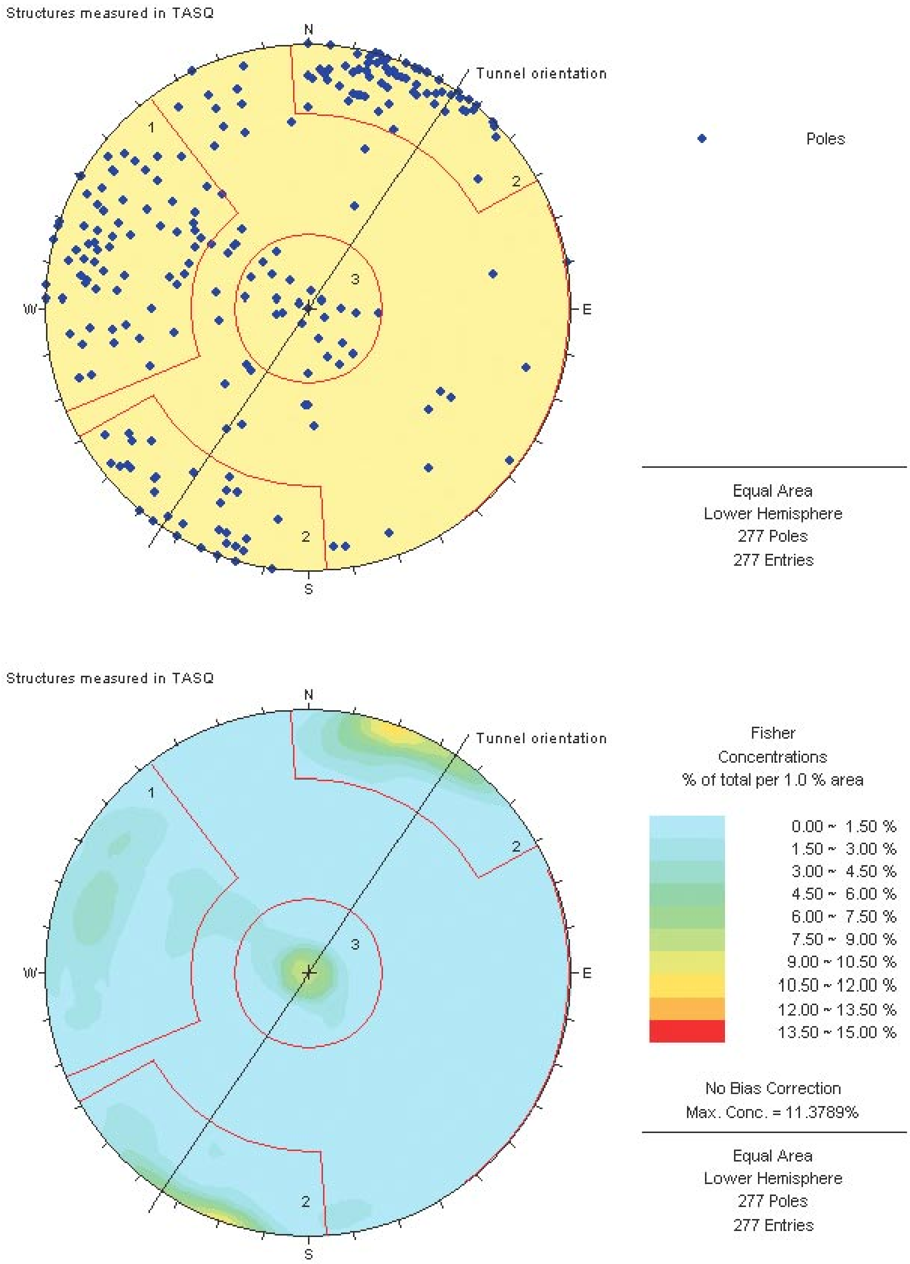


Figure 2-1. *Projection of poles to fracture planes to all structures measured in tunnel TASQ /Staub et al. 2004/. Group 1 includes the shear zone parallel structures. It also contains a set of brittle, steeply dipping structures striking N-NNE. Group 2 consists of a brittle, sub vertical set striking perpendicular to the tunnel. This group includes the majority of the faults in TASQ. Group 3 represents a set of sub horizontal brittle fractures.*

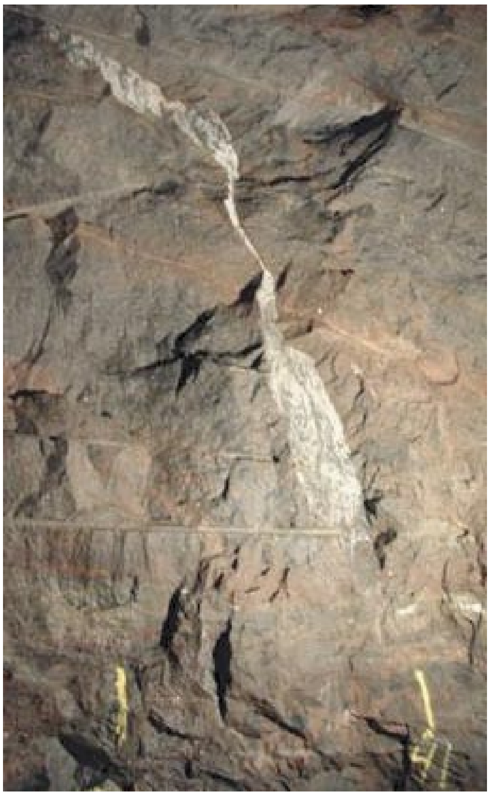


Figure 2-2. *The NE trending fracture set (set 1), one fracture in a small angle to the left wall and aboutment of the TASQ tunnel.*

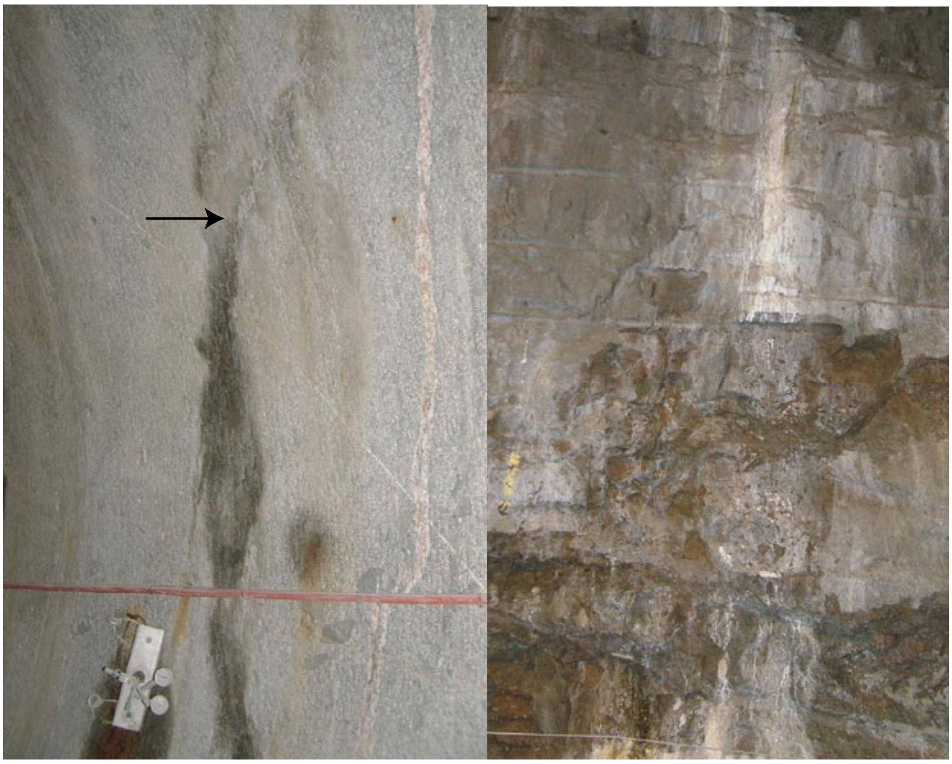


Figure 2-3. *The waterbearing fracture set 2, trending NW. Observation in the TBM tunnel just outside the TASQ tunnel (left). Notice how the fracture spays into two in the upper part of the photo. The right photo is from the TASQ tunnel. The fracture set is perpendicular to the tunnel, and (at the arrow) less visible. This section was grouted, the light strips are calcite precipitation from the cement.*

2.3 Mechanical characteristics of the rock

The state of stress in the lower part of the AHRL is described by /Jansson and Stigsson, 2002/. These data has been confirmed by /Staub et al. 2004/. The principal stresses in the rock mass around the TASQ tunnel are summarised in Table 2-2. There is believed to be an absolute error of roughly 2 MPa in the stress determination. Notice that the major principal stress is almost perpendicular to the TASQ tunnel, and close to parallel with fracture set 2 in Figure 2-1.

The mechanical properties are summarised by /Andersson, 2003/, Table 2-3.

Table 2-2. State of stress around the TASQ tunnel /Staub et al. 2004/.

Principal stress	Magnitude (MPa)	Trend/Plunge (degrees)
Sigma 1	27	310/07
Sigma 2	15	090/83
Sigma 3	10	208/00

Table 2-3. Mechanical properties of the rock.

Parameter	Mean value	Unit
Uniaxial compressive strength, low	130	MPa
Uniaxial compressive strength, high	210	MPa
Crack initiation stress	121	MPa
Crack damage stress	204	MPa
Young's modulus, intact rock	76	GPa
Young's modulus, rock mass	55	GPa
Poisson's ratio, intact rock	0.25	–
Poisson's ratio, rock mass	0.26	–
Friction angle, intact rock	49	Degrees
Friction angle, rock mass	41	Degrees
Cohesion, intact rock	31	MPa
Cohesion, rock mass	16.4	MPa
Tensile strength	14.3	MPa
Thermal conductivity	2.60	W/m, K
Volume heat capacity	2.10	MJ/m^3, K
Thermal linear expansion	7.0E–06	l/K
Density	2.731	g/cm^3
Initial temperature of the rock mass	15	°C

The lower uniaxial strength is associated to some portions of altered diorite that occur to a smaller extent.

2.4 Summary on conditions for excavation

The rock mass is very competent, with few heterogeneous sections that could influence the excavations. The density of the rock is however a little higher than for average crystalline rock in Sweden.

The structures in the rock mass form a relatively "blocky" rock mass. However, with the low fracture frequency the rock conditions are good, for example the Q-index rating is in the range of "good to very good rock", see Figure 2-4. The only fracture set of some significance for the excavations is the NW-SE trending steeply dipping fracture set. This set is partly open and water bearing (Figure 2-3). Because the tunnel is orthogonal to this set does the tunnel face trend to be parallel to the partly open fractures, causing a tendency for a drummy face. The actual state of stress contributes to this phenomenon.

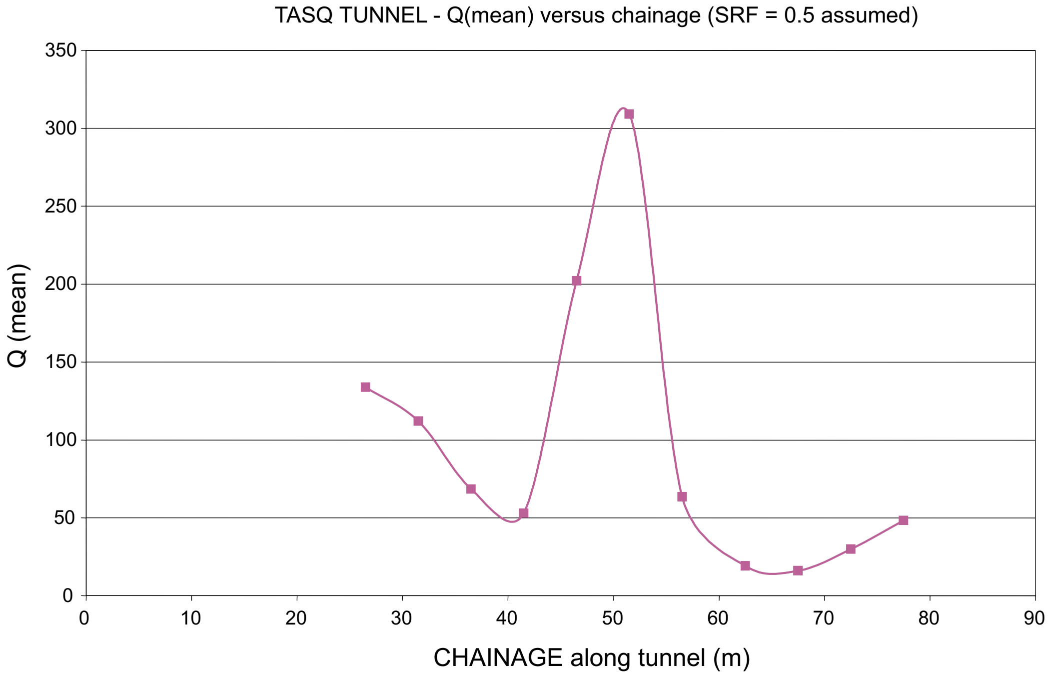


Figure 2-4. *Distribution of Q index along the TASQ tunnel. /Barton, 2004./*

3 Requirements on the excavation works

As described in Chapter 1 was the main purpose with the tunnel was to provide suitable conditions for a rock mechanics experiment. The planned Pillar Stabilty Experiment (APSE) was designed to form a pillar by drilling two large-size bore holes close to each other /Andersson, 2003/. To enable a sufficient stress concentration in the pillar it was found that two geometrical requirements for the tunnel had to be met:

1. For a tunnel perpendicular to a horizontal major principal stress will the stress concentrations in the roof and the floor increase with increasing height/width ratio for the tunnel cross section.
2. A circular floor concentrates the stresses to the centre of the floor, whereas a traditional horse-shoe shaped tunnel concentrates the stresses to the corners of walls – floor.

The geometrical requirements were important of these reasons. However, the current Swedish practice in tunnelling to allow a maximum over-break due to look-out angle etc of 0.3 m was considered to be practical, and of no significant impact for the planned experiment. Therefore were "standard" requirements on tolerances in contours were adapted to the technical specifications for the tunnelling.

There was only at one location were the requirements on bore hole precision and a minimum EDZ were high, and that was in the planned pillar position. The correct circular floor would concentrate the stresses to the centre of the tunnel floor, and a minimum depth of the EDZ would not reduce the deformation modulus of the rock mass to a large depth. To achieve a practical excavation the tunnel was designed with a top heading in a traditional horse-shoe shaped tunnel, and a bench with circular floor, see Figure 3-1.

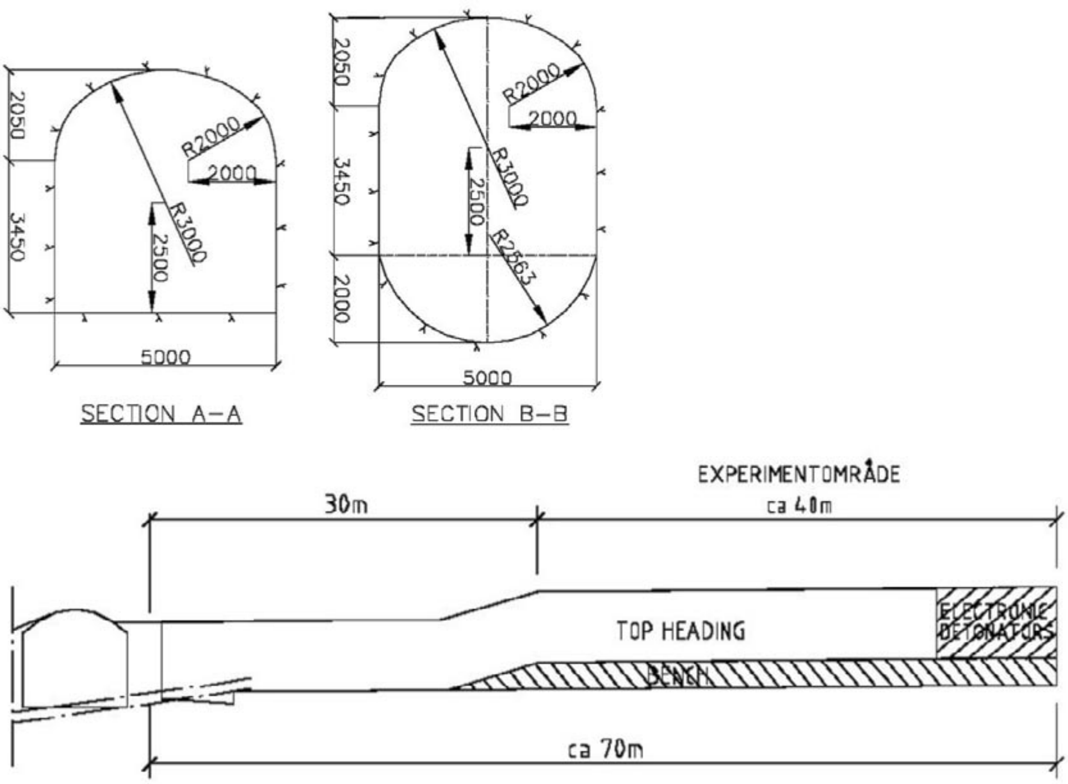


Figure 3-1. *Designed tunnel geometry.*

The relatively simple relations for excavation damage, based on empirical relations have been applied for more than a decade in Sweden, even although it is based on relatively few observations /Olsson and Ouchterlony, 2003/. But because these recommendations have been applied in many civil engineering projects in Sweden it was realistic to state in the requirements that "the maximum depth of the excavation induced damage shall not exceed 0.3 m in the contour". These requirements were stated for the roof and walls of the top heading, as well as for the circular floor. Even although it is not a standard procedure to determine the depth of excavation induced fractures in Swedish civil engineering projects it was known from the ZEDEX experiment /Olsson and Reidarman, 1995/ that a maximum depth of 0.3 m was achievable at the actual site. To make a charge control feasible was it prescribed to only use cartridged explosives.

To achieve the requirement for the APSE experiment to have control of boundary stresses was it necessarily to have the cross section with the curved floor extending least some 10–12 m to each side of the planed experiment position. To enable the Contractor to trim in for drilling the circular bench the excavation of the bench was started relatively early, Figure 3-1.

Other requirements for the D&B operations were related to vibrations, especially at the nearby shaft station and its furnishings.

There were strict restrictions in accessibility to the working phase. Due to other ongoing works further down the tunnel was blasting not permitted before 10 pm. And because of the daily activities were the mucking out had to be completed before 7 am. This caused the need for careful measures (Figure 3-2) at the collar of the TASQ tunnel. The first 16 m took almost a month to excavate.

Figure 3-2. *The restrictions on damage in the main tunnel, as well as the limited working hours caused the need for extensive safety measures during excavation of the first roughly 16 m of the tunnel. The photo shows installing a rubber math for fly rock protection. Notice shape and size of the mucked rock in the lower right part of the photo.*

4 Planning

4.1 Blast planning

The basis for all blast planning work could be found in different documents and for each part of the tunnel different pre-requisites have been governing the planning and execution of the blasting work.

The following documents, planned activities in the tunnel, and physical conditions in the close vicinity of the tunnel has effected blast planning:

- The Technical specification and the Blasting Risk Assessment, "Riskbedömning samt rekommendationer för sprängarbeten vid utbyggnad av nya tunnlar inom –450 m nivån, Äspö HRL, Metron Mätteknik AB", from the tender documentation.
- The three last top heading blast rounds including testing of the electronic detonators, which were not included in the Technical specifications.
- Installations close to the tunnel portal and how to protect these from fly-rock and air shock wave damage.

The Technical specifications outlined the pre-requisites regarding the main purpose of the tunnel – the Pillar Stability Experiment. This is described in more detail in section 3 of this report.

The blast risk assessment indicated a first approach of maximum charge weight allowed for the tunnel blasting. A PM from Skanska Teknik, "*Kommentarer till PM: Riskbedömning samt rekommendationer för sprängarbeten vid utbyggnad av nya tunnlar inom –450 m nivån, Äspö HRL, Metron Mätteknik AB, February 12*", made comments to the report and identified some further restrictions in how much explosives that could be used instantaneously. Due to some uncertainties in the earlier conducted test blast that was a part of the Blast risk assessment, the start up of the tunnel blasting should be carried out in a very cautious manner due to the furnishing in the vicinity of the nearby shaft station. The maximum allowed charge weight was indicated as lying between 0.1–1.0 kg. From the design point of view the start up of the tunnel was similar to a test blasting procedure.

Experiences from the tunnelling of the Äspö ramp was also embedded in the background conditions of the design. The specific charge and specific drilling had been higher than in normal hard rock tunnelling indicating a somewhat more solid rock mass.

4.2 Phases in the excavation

4.2.1 General design

The objective with the blast design was to find a general blast pattern that could be used along the entire tunnel and that could function with the range of charges that were expected to be used. Despite the slightly different conditions in the separate parts of the tunnel it was an advantage not to change practice too often. The tunnel is only 70 m long and there was no time to adjust to new practice of blasting over such a short distance.

The excavation work was divided in three different phases, see Figure 3-1. The first phase of the tunnel was an ordinary 26 m^2 tunnel. After approximately 30 m a ramp separated the tunnel section into a top heading and a bench, with a total area of 33 m^2. In the transition zone the floor changed shape from a flat to an arched floor.

The top heading was around 40 m long. Phase two comprised a three round test with electronic detonators in the contour holes

After the last top heading round was excavated and the roof had been reinforced with fibre reinforced shotcrete the bench was excavated out with horizontal holes as the third phase.

4.2.2 The test blast and first part of the tunnel, up to chainage 0/016

The proximity, 8–10 m, from the portal of the TASQ tunnel to the main ventilation system including a large ventilation valve in the ramp at 450 m level was the most governing factor for the first part of the tunnel. Also other support systems – power, communication, drainage etc passed very close in front of the tunnel portal. Consequently, all other restrictions became secondary. In this part of the tunnel the tunnel face was divided into two or more blasts. Round lengths started at 1.5 m at the portal and reached 4 m in the round before 0/016. The maximum charge weight was 0.4 kg at the portal but it was quickly raised to over 1 kg. At section 0/012 the charge weight passed 2 kg without exceeding the vibration limits.

4.2.3 From chainage 0/016

The rest of the tunnel was more or less designed according to the specifications related to cautious and careful blasting. Fly-rock and air shock waves were under control and the distance to the tunnel portal was now fairly safe. Round lengths were in general between 3.9 and 4.5 m with a few exceptions. No limitations regarding charge weight were experienced.

5 Methods for excavations

5.1 The top heading

Basically one drilling pattern was used along the whole tunnel length. The cut of the round was lowered one row at section 0/016. The main reason for this was to make the protecting steel plate to cover the whole cut.

The drill pattern also worked for different round lengths. So when the allowed charge weight was changed only the length of the round had to be changed. The hole sizes were 48 and 102 mm in diameter. Four 102 mm diameter open cut holes would ensure a safe opening of the round. Since the pattern was planned to contain charges of 22 mm up to a maximum 25 mm diameter in the column parts the pattern became fairly dense.

The cautions blasting called for perimeter holes with a 0.45 m spacing and a 0.5 m burden which is a somewhat tighter pattern than normal. The helper row holes had a spacing of 0.6 m and burden of 0.5 m. Stoping holes were typically drilled in a 0.6×0.6 m pattern.

The contour holes were fully charged all along the hole length. The number of plastic pipe charges shown in Appendix 1 may vary both due to the distribution in hole lengths despite the nominal round length and because the Dynotex pipe charges are difficult to cut. The risk of back break is low due to the weak charge concentration. The same situation also applies to the helper holes. In practice, this means that the uncharged length in contour and helper holes varies between 0 and 0.5 m which corresponds to the approximate lenght of the explosives column.

Figure 5-1 shows the drilling pattern for the first part of the tunnel, up to section 0/027 and for the top heading up to the end point at 0/081.

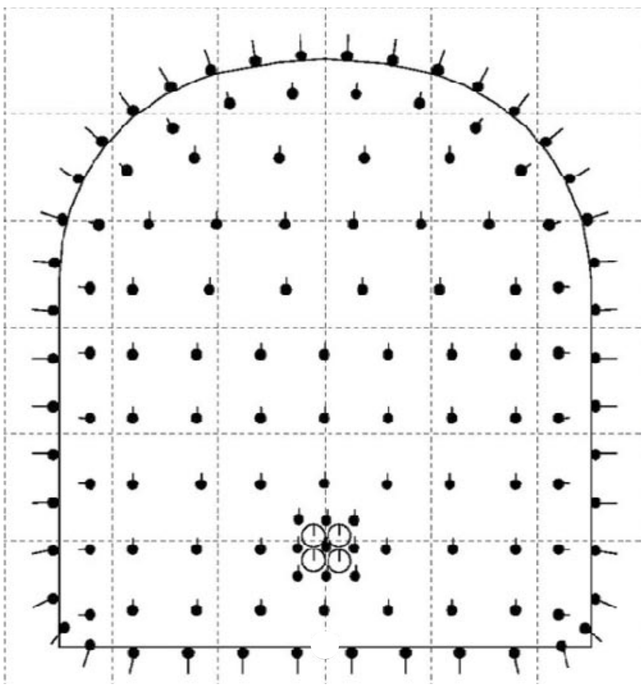


Figure 5-1. *The top heading drill pattern.*

5.2 The bench

Between sections 0/031 and 0/041 a transition zone changes the tunnel shape from a flat floor into an arched floor showed in Figure 5-2. Both the shape and the blast design of the bench were aiming at a much more careful blasting than normal. One of the goals of this project was to reduce the blast damage to the bottom as much as possible while still using the drilling and blasting method. As can be seen in Figure 5-3 the bottom row and helper row is designed the same way as the wall and roof contours in the top heading. See also Appendix 1.

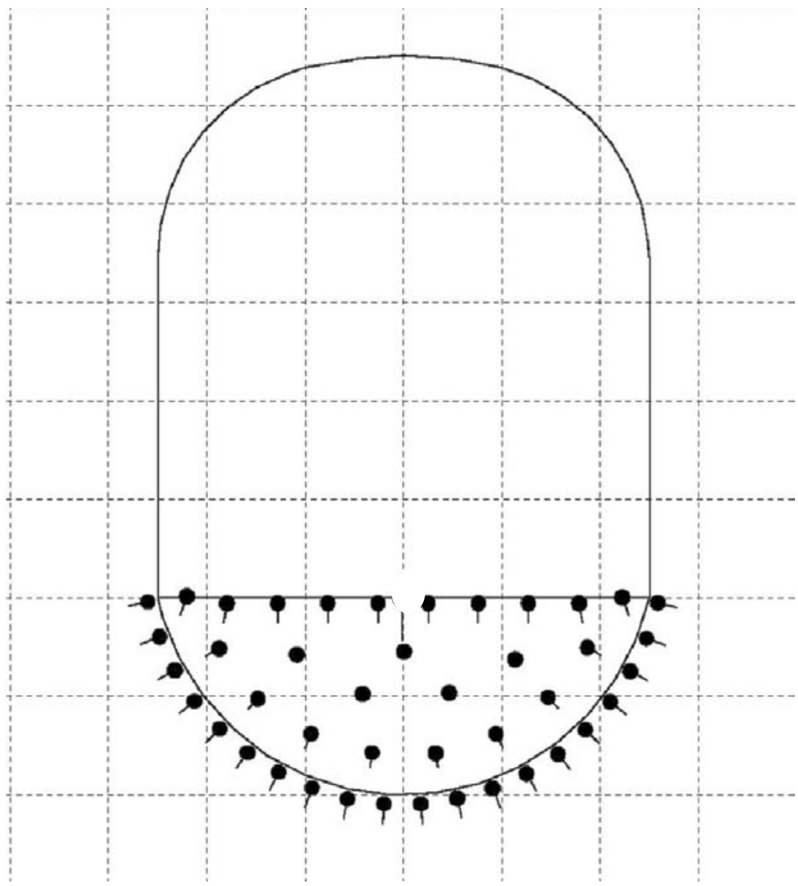


Figure 5-2. *The bench drill pattern including top heading bottom row.*

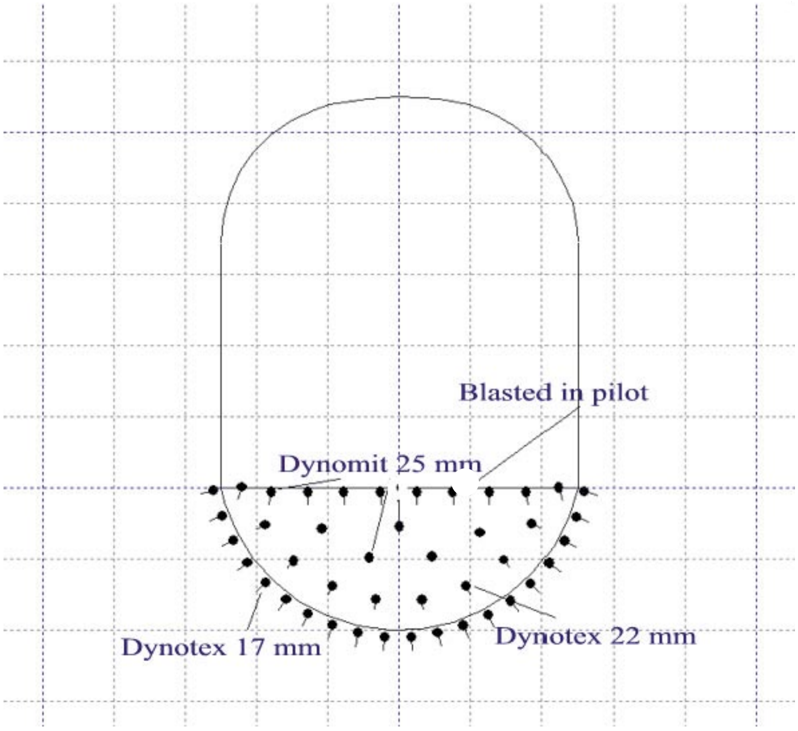


Figure 5-3. *The charging plan of bench rounds.*

5.3 Explosives

The types and charge sizes of the explosives were taken from the manufacturer's standard assortment /www.dynonobel.com/. Only packaged explosives were used.

The types and sizes that were used can be found in Table 5-1 (more in Appendix 1):

Table 5-1. Explosives used in the TASQ tunnel.

Brand name	Diameter x Length (mm)	Weight (kg)	Field of application
Nobelprime	15 x 150	0.03	Primer
Dynomit*	29 x 380	0.32	Bottom charge
Dynomit	30 x 380	0.38	Bottom charge
Dynorex	29 x 1,110	0.94	Column charge
Dynorex	25 x 1,110	0.73	Column charge
Dynotex 1	22 x 1,000	0.37	Column charge
Dynotex 1	17 x 460	0.095	Column charge

* Used when size 30 x 380 mm was out of suppliers stock

For perimeter and helper holes a small diameter pipe charge was used with the brand name Dynotex from Dyno Explosives. Cut, lifters and stoping holes were charged with Dynomit or Dynorex from Dyno Nobel. To initiate the perimeter charge a small primer was used, the Nobel Prime. For further information regarding the characteristics of the explosives please refer to Table 5-2.

Table 5-2. Some explosives used and their characteristics.

Explosives	Components	Density (kg/m³)	VOD[1] (m/s)	Water resistance
Dynomit	Nitro glycol, ammonium nitrate.	1,400	3,000- 6,000	Excellent
Dynorex	Nitro glycerine, nitro glycol, ammonium nitrate		>4,500	Good
Dynotex 1	Nitro glycol, ammonium nitrate	1,000	2,400	Limited
Nobel Prime	Nitro glycol, PETN, ammonium nitrate	1,500	>6,000	Excellent

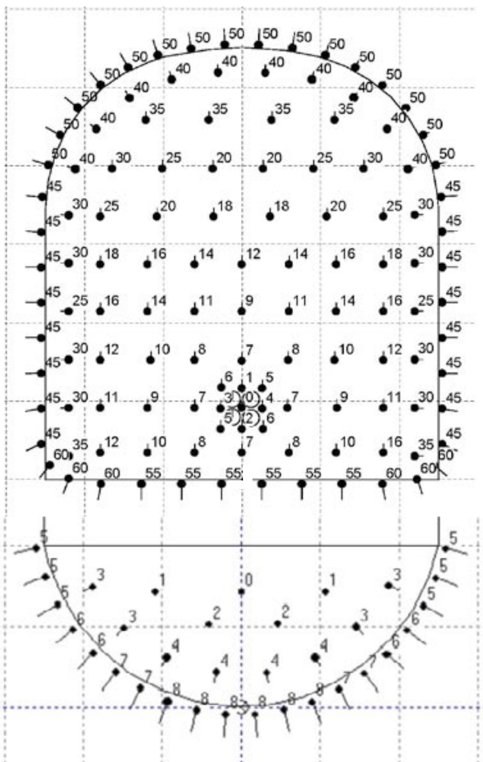
[1] Detonation velocity

5.4 Initiation system

Two types of initiation systems were used. In all rounds except the last 3 top heading rounds Nonel detonators initiated the charges. The 3 final top heading rounds formed a separate test with electronic detonators. These rounds are more closely described in Section 5.8.

Two different types of Nonel detonators were used, the (ms) millisecond and the long period (tunnel series) detonator. In order to create as many separate numbers as possible it is practicable especially for short rounds also to include ms detonators in the tunnel series, below number 6 (600 ms). The longest delay in the ms series is 500 ms. A combination of the two was only used in the first part of the tunnel where blasting with a divided face was applied (see Section 4.2.2).

Figure 5-4 shows to the left the standard initiation pattern for the top heading and below a typical initiation pattern for the bench. The delay numbers shown refers to the time delays shown in the table to the right.



Delay numbers in the Dyno Nobel Nonel tunnel series.

Delay #	Delay time ms	Delay #	Delay time ms
0	25	14	1400
1	100	16	1600
2	200	18	1800
3	300	20	2075
4	400	25	2500
5	500	30	3000
6	600	35	3500
7	700	40	4000
8	800	45	4500
9	900	50	5000
10	1000	55	5500
11	1110	60	6000
12	1235		

***Figure 5-4.** The initiation pattern for top heading and bench. The delay numbers refer to time delays shown in the table to the right.*

5.5 Stemming and fly-rock protection

In order to reduce the risk of having explosive cartridges and plastic pipe charges thrown out of their blast holes before the time of detonation, clay cartridges were used as stemming to confine the charges.

Later, when the face moved away from the portal area the clay cartridges were changed to polystyrene plugs.

One of the main concerns in the first part of the tunnel was to prevent fly rock from penetrating into the main level at –450 m. The protection consisted of four steps.

1. A steel plate was vertically positioned in front of the cut of the round.
2. Rock material was placed in front of the steel plate in order to keep it in place.
3. Heavy rubber mats were hung at the tunnel entrance.
4. Heavy rubber mats point protected sensitive cables and installations on the –450 m level (see Figure 3-2).

This system of protection was used for all top-heading rounds. At first the bench rounds had a cover of rubber mats but after some initiation problem, probably caused by damage to the Nonel tubes by the handling of the mats, the rounds were blasted without any covering.

5.6 Equipment

One Atlas Copco Rocket Boomer rig has drilled blast-, grouting and probe hole. The three-boom 353 ES from 1997 is equipped with 16" BUT 35 feed beam with COP 1838 hydraulic rock drills, the ECS drilling system and Bever guidance control. The outer booms are equipped with the RAS rod-adding system to drill grouting and probe holes. The COP 1838 drill is designed for high-speed drilling in the 45–102 mm diameter hole range.

There are several manufactures of guidance control equipment. The Rocket Boomer rig was equipped with a system from Bever Control AS. It provides a production control in blast tunnelling with drilling and profile guidance. The main features of this drilling guidance system is: tunnel laser navigation, drill rod direction monitoring and control, drill pattern display and recording, drill depth control, planning and reporting on a desktop PC.

For service drilling an Atlas Copco water-flushed pusher leg rock drill, BBC 16W was used.

Charging the drilled blast holes was done from a scaling platform or from the Rocket Boomer’s charging basket.

A Cat 966 dumper was used for mucking. At the bench with an arched floor a backhoe was used together with the Cat 966. The blasted rock volumes were transported with three or four 3-axled trucks.

The scaling works were done from the same scaling platform as was used for charging.

A robot rig with Putzmeister equipment mounted on a Scania chassis did the wet shotcrete reinforcement.

The grouting works was done from a grouting platform for one-hole grouting with a ZBE100 grouting pump. The platform was equipped with gauge for registration of the pressure and the pumped volume of grout mortar.

5.7 EDZ (Excavation Damaged Zone) Planning

5.7.1 General

Since many years a table is used in Sweden for the judgement of blast damage caused during blasting. Commonly used explosives for such work are listed in order of their equivalent linear charge concentration in terms of kg Dynamex per metre. The table suffers from many shortcomings and has only been verified for a couple of explosives under specific circumstances. A clear definition of damage is lacking too. Furthermore the table does not take in consideration the influence of blast hole pattern, scatter in initiation and coupling ratio. It has never the less been a practical tool for designing smooth blasting.

An intensive research of how cracks are caused by blasting has been carried out at SveBeFo. A large number of holes have been blasted and the cracks in the remaining rock have been examined. Coupling ratio (charged diameter/hole diameter), spacing, water in the holes, scatter in the initiation and the influence of different explosives on crack lengths are some of the examined factors. With the knowledge of the effects of these factors a new prediction formula for blast damage has been proposed /Ouchterlony and Olsson, 2003/.

The new formula emanates from measured crack lengths in granite. Compensation factors for decoupling, spacing, initiation, water in holes and the rock are included in the formula. One important factor included in the formula is the variation of the detonation velocity due to decoupling in wet or dry holes.

The damage zone is defined as the maximum crack length for the cracks emanating from the half-casts. The formula is:

$R_c = R_{co} \cdot F_h \cdot F_t \cdot F_v \cdot F_b$ where (5-1)

R_{co} is a reference crack length and F_h etc are correction factors for spacing (h), interval time (t), wet holes (v) and type of rock (b) with and its natural cracks.

5.7.2 The test section

The first possible test area for the formula was the TASQ-tunnel. It was decided to reserve the last three rounds for testing one of the factors in the formula. The factor to be tested was the influence of interval time. Earlier tests by SveBeFo have shown that simultaneous initiation gives the shortest cracks /Olsson and Bergqvist, 1997/. With ordinary caps, like Nonel, it is impossible to achieve simultaneous initiation due to the high scatter in the initiation time. With electronic detonators it is possible to have simultaneous detonation of the holes.

After a site visit early in the project it was decided to use the same drilling- and charging plan as Skanska had used for the other rounds. The hole spacing for the contour holes was 0.4 m and the holes were charged with 17 mm Dynotex, see more details in Table 5-4.

Table 5-4. Drilling and charging specifications of the contour holes for the test rounds.

Length of rounds	4.5 m
Spacing contour holes	0.4 m
Charging of contour holes	2 Nobelprime 15×150 mm+10 Dynotex 17×460 mm
Charge weight per contour hole	1.01 kg

For the last three rounds it was planned to use electronic detonators in the contour holes. Three variations of simultaneous detonation were planned. Simultaneously detonations of many contour holes would be preferred to obtain few and short cracks in the remaining rock. However due to vibration limitations in the ABSE-project only five contour holes could be simultaneously initiated. In this manner the initiation of the contour holes were divided in different groups. Within each group the caps should detonate simultaneously.

The electronic detonator system in this test was i-kon® from Orica /www.i-konsystem.com /. The system consists of three hardware components; the detonator, the Logger and the Blaster, see Figure 5-5. The detonator can be programmed from 1 to 8,000 ms and each detonator has a unique factory assigned ID-number. The Logger is capable of storing data for up to 200 detonators and for assigning each detonator a delay time. The Logger also tests the detonators connected to the circuit and measure leakage. The Blaster is the piece of control equipment which arms and fires the blast.

As a special function control it was decided to measure the real interval time from the electronic detonators. A number of bore holes were therefore drilled into the tunnel wall at a close distance from the front. Accelerometers were grouted inside the holes and the acceleration measured by a transient recorder. More details may be found in a separate report /Nyberg et al. 2005/.

Test round 1

The plan for this round was to initiate all the contour holes (roof and walls) with electronic detonators. The rest of the holes in the round should be initiated with Nonel. Figure 5-6 shows the initiating plan.

Figure 5-5. *The i-kon system.*

Test round 2

The plan for this round was to compare electronic initiation with ordinary Nonel in the same round. The contour holes on the right side of the tunnel and the roof should be initiated with electronic detonators. The contour holes on the left side should be initiated with Nonel, see Figure 5-7. The contour holes in the bottom were planned to have Nonel initiation.

Test round 3

Simultaneous initiation has proved to give shortest cracks and has a very positive effect according to the new SveBeFo crack formula /Olsson and Ouchterlony, 2003/. The purpose of this round was to examine small time differences in initiation time and its effect on the crack length. The initiation time was varied in steps in the walls. First some simultaneous holes then a 1 ms step to the next hole followed by a 2 ms step to the next hole then a 4 ms step and finally a 13 ms step, see Figure 5-8. The upper parts of the walls and the roof had simultaneous initiation. The contour holes in the bottom were planned to have Nonel initiation.

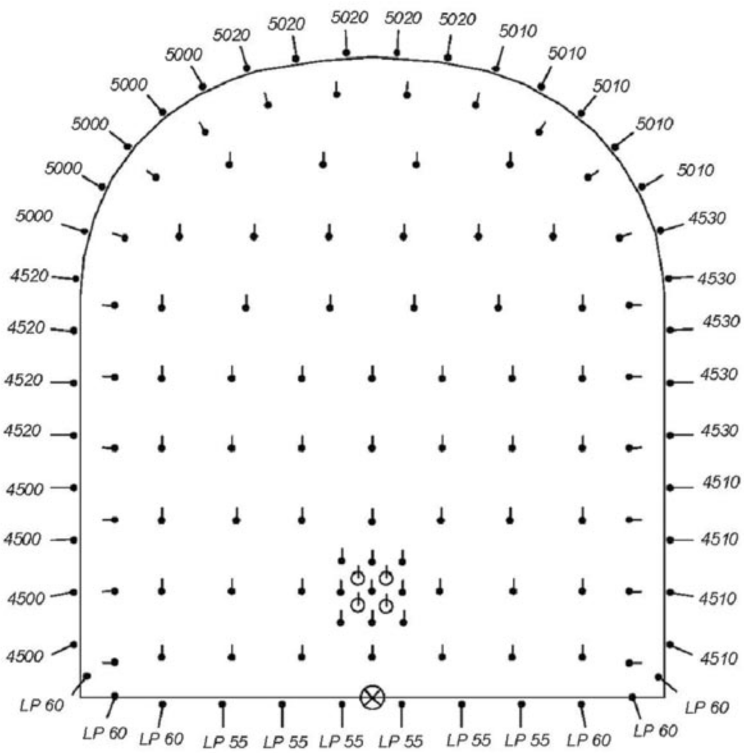


Figure 5-6. Planned contour initiation of Test round 1.

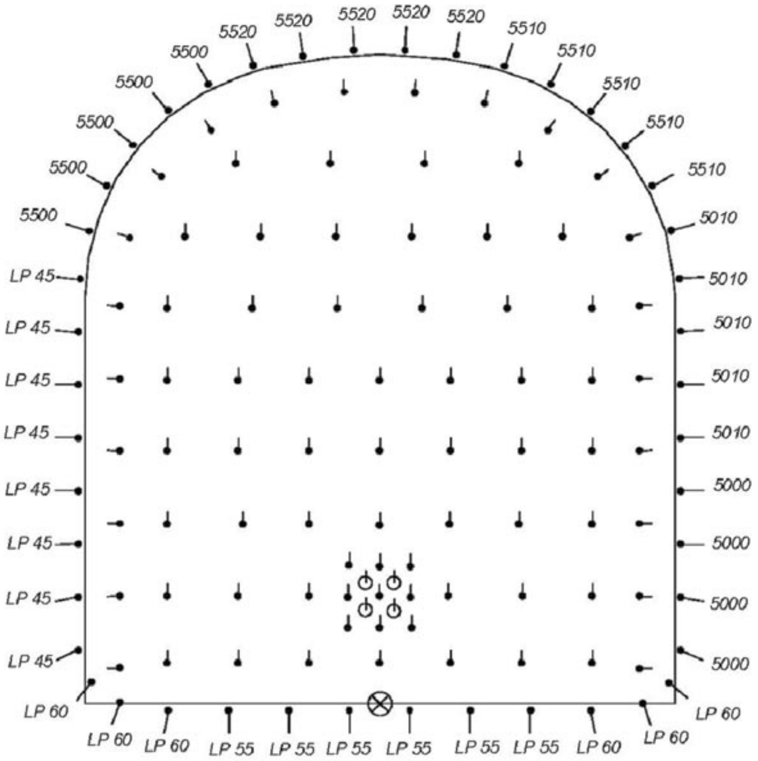


Figure 5-7. *Planned contour initiation of Test round 2.*

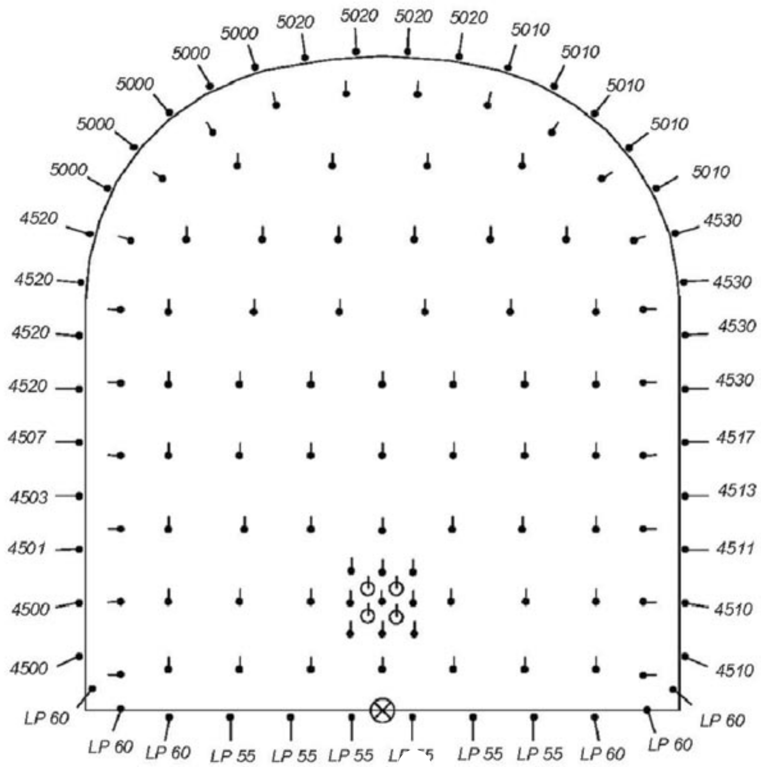


Figure 5-8. *Planned contour initiation of Test round 3.*

6 Excavation

6.1 Adjustments in production planning

Due to restrictions in the tender dossier Skanska had to make some adjustments in the production planning. Round blasting wasn't allowed before 10 pm and the mucking had to be done before 7 am. All transports of blasted rock were carried out during the night so it shouldn't interfere with the ordinary activity in Äspö Hard Rock Laboratory. In practice the rounds where blasted between 10 pm and 2 am. The mucking couldn't start before the ventilation had cleared all the gases after a blast which took one to two hours. If mucking started earlier the gas plug reduced the visibility in the tunnel to only a couple of meters.

The shift consisted of one drill operator/rock blaster (two during a period), one mechanic and one reinforcement/grouting labourer. The mucking and transport was done by a sub-contractor.

Two grouting shields were performed at sections 0/049 and 0/059. This report will not describe the grouting experience. For more information regarding this experiment please refer to /Emmelin et al. 2004/.

The roof reinforcement in the tunnel consisted of 50 mm thick fibre reinforced shotcrete and 25 mm thick un-reinforced shotcrete layers. The walls were un-supported. No rock bolts were installed.

6.2 Drill and blast records

The tunnel portal starts at approximately section 0/010. In the first part of the tunnel, the tunnel section was divided into two or more blasts. The drilled length started at 1.5 m at the portal and reached 4 m in the round before 0/016. In round 11 the helper and contour rows from section 0/012 to 0/016 were blasted in round 11. From section 0/016, round 12 and on the round lengths were between 3.9 and 4.5 m with a few exceptions. No limitations regarding charge weight was experienced. In rounds 15, 16 and 22 the blasting resulted in misfires, see Section 6.3.1. Rounds 19 to 22 divided the section into a top heading and a bench, see section 5.2. In round 25 to 27 the tunnel convergence was measured and they had shorter hole depths and hence a shorter advance rate. Round 32 to 34 were blasted with electronic detonators, see Section 6.3.8. Round 35 was the first bench round. In round 41 initiation problems occurred probably caused by damages to the Nonel from the handling of the protection mats. Round 42 was a re-blast of some holes from round 41. Round 46 was a re-blast of some of blasting of the lower contour holes remaining from round 45. See more in Figure 6-1.

Table 6-1 is a summary of explosive consumption. In Appendix 1 the number of cartridges and explosive type/brand name for different hole types are described. The accuracy in follow-up of cooperative charge weight is good for the different types of holes, cut, stoping, helpers, contour and lifters. During charging work, the rock blaster has done some adaptation due to the geology, fracture systems at the face and experiences during the blast hole drilling. Therefor there are some uncertainties in the total charge weight values for each round, approximately 1–4 kg per round.

The bench rounds have been charged in the same cautious manner as the helper and the contour rows at the top heading, se more in section 5.2.

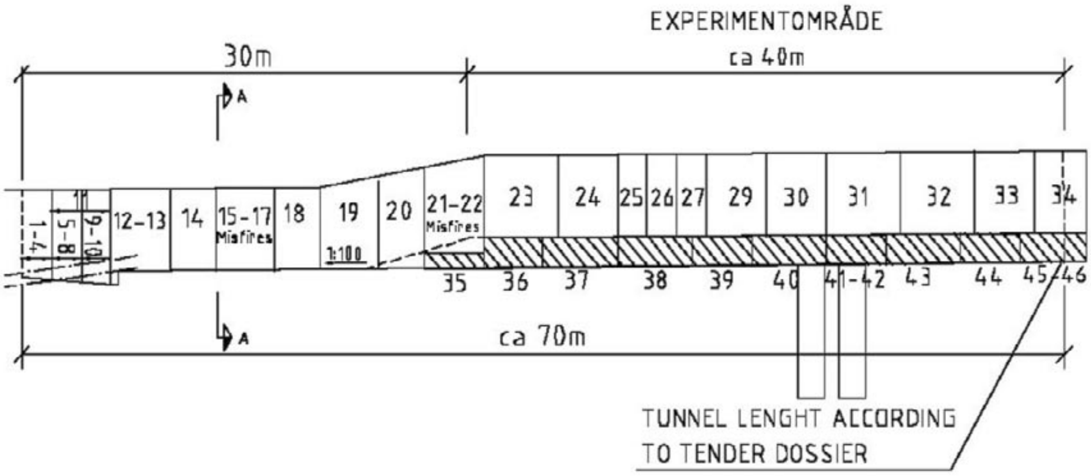


Figure 6-1. The tunnel profile and the round numbers.

Table 6-1. Summary of explosive consumption. See Appendix 1 for more details.

Round no	Start section	Date	Number of holes	Empty holes	Hole depth (m)	Cooperative charge weight (kg)	Total charge weight (kg)
1	0/010	2003-04-15	1	4	1.5	0.39	0.4
2	0/010	2003-04-15	4		1.5	0.78	3.0
3	0/011	2003-04-22	8		1.5	1.11	8.9
4	0/012	2003-04-24	24		1.5	1.11	26.6
5	0/012	2003-04-27	7	4	2.0	1.31	9.2
6	0/012	2003-04-29	11		2.0	0.82	9.0
7	0/012	2003-04-29	21		2.0	1.19	25.0
8	0/012	2003-05-05	13		2.0	2.11	22.8
9	0/014	2003-05-07	9	4	3.0	2.57	23.1
10	0/014	2003-05-07	38		3.0	1.46	50.0
11	0/012	2003-05-08	78		4.0	1.83	100.9
12	0/016	2003-05-10	43	4	3.9	2.57	96.1
13	0/016	2003-05-11	92		3.9	2.57	107.5
14	0/020	2003-05-13	125	4	3.0	2.21	176.9
15	0/023	2003-05-14	125	4	4.0	2.94	244.3
16	0/023	2003-05-14	116			2.57	216.4
17	0/023	2003-05-19	100			2.57	175.3
18	0/027	2003-05-21	125	4	4.0	2.94	261.9
19	0/030	2003-05-22	125	4	4.0	2.94	261.9
20	0/034	2003-05-23	133	4	4.0	2.94	275.2
21	0/037	2003-05-24	138	4	4.5	3.68	301.7
22	0/037	2003-05-25	102			3.32	221.6
23	0/041	2003-05-26	125	4	4.5	3.32	280.2
24	0/046	2003-05-27	125	4	4.0	2.95	255.8
25	0/050	2003-06-15	125	4	2.3	1.84	148.2
26	0/052	2003-06-16	125	4	2.4	1.84	148.2
27	0/054	2003-06-17	125	4	2.4	1.84	148.2
28	Portal floor	2003-06-24	11		4.6	1.6	217.8

Round no	Start section	Date	Number of holes	Empty holes	Hole depth (m)	Cooperative charge weight (kg)	Total charge weight (kg)
29	0/056	2003-06-24	125	4	4.1	2.94	260.4
30	0/060	2003-06-28	125	4	4.5	3.63	279.2
31	0/064	2003-06-29	125	4	4.5	3.63	282.8
31 B	Portal pillar	2003-06-29	11		2.0	0.54	5.9
32	0/069	2003-07-01	124	4	4.5	4.10	259.8
33	0/074	2003-07-02	125	4	4.5	4.34	282.6
34	0/078	2003-07-02	125	4	4.0	4.34	282.6
Bench							
35	0/037	2003-07-11	17		4.4	1.47	19.0
36	0/041	2003-07-12	32		4.6	1.81	40.4
37	0/045	2003-07-12	32		4.4	1.81	40.4
38	0/050	2003-07-13	32		4.4	1.81	40.4
39	0/055	2003-07-13	30		3.7	1.62	36.4
40	0/059	2003-07-14	27		4.6	1.81	33.3
41	0/064	2003-07-14	28		4.5	1.81	35.0
42	0/064	2003-07-15	10			1.81	13.3
43	0/068	2003-07-15	28		4.6	1.81	35.6
44	0/073	2003-07-15	28		4.4	1.81	35.6
45	0/077	2003-07-16	27		3.9	1.62	33.4
46	0/077	2003-07-17	10			0.44	4.4

6.3 Practical experiences

The practical experiences described in this section refer both to the top heading and the bench rounds. If not, this will be understood from the text.

6.3.1 Misfires

At three occasions the blasting resulted in misfires. The definition of misfire is when a major part of the rock mass is left unbroken after the blast and that many holes of the blastholes are intact. In this report a misfire involves more than 10–20% of the total volume and that many of the blast holes are intact.

The rounds showed in Table 6-2 are considered as misfires.

Table 6-2. Misfires.

Round no	Section	Date	Comment
15	0/023	2003-05-13	Section 23
16	0/023	2003-05-14	Re-blast of round 2003-05-13
21	0/037	2003-05-24	Section 37

Round no 15

Round no 15 was drilled 4.0 m deep and the maximum charge weight per hole was 2.94 kg charged in the cut holes.

The result from blast no 15 regarding breakage shows that the first and second quadrants round the cut and the holes above the cut had broken deeper inside the round but not entirely all the way out to the free face.

A deeper analysis of the vibration records in Figure 6-2 shows that the second hole in the cut is missing. Normally one missing charge in a cut with 4 large holes should not reduce the possibility of establishing a good opening for the rest of the round.

Round no 16

Blast no 16 was the re-blast of round no 15. Difficulties in charging the holes were an effect of the previous misfire. Some holes were badly damaged and the explosives were difficult to charge.

The result showed a further propagation of the broken area above and around the cut but still with an unsatisfactory end result.

As showed in Figure 6-3 very low amplitudes were registered in the vibration monitoring record for the first 7 holes in the round.

Round no 21

Round no 21 was drilled to a length of 4.5 m and had its maximum charge per hole, 3.3 kg in the cut holes.

The cut holes hade broken out of the face. Some holes below and to the right of the cut and some holes above the cut had broken out further in but not all the way out to the free face. No disturbances were found in the initiation sequence of the cut, see Figure 6-4. All holes in the first critical part of the initiation showed full functionality.

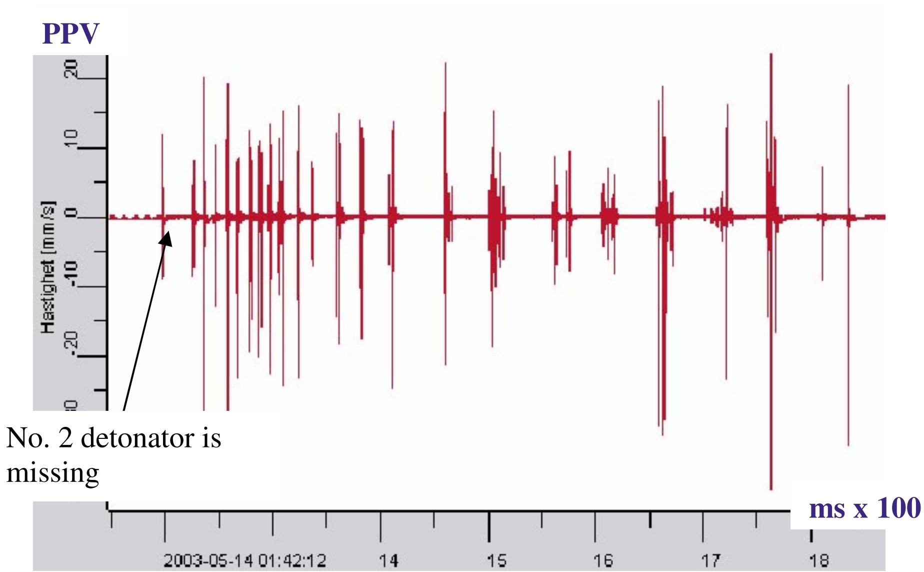


Figure 6-2. *The time history for blast round no 15. Each wave package in the left part of the time window corresponds to a detonating hole.*

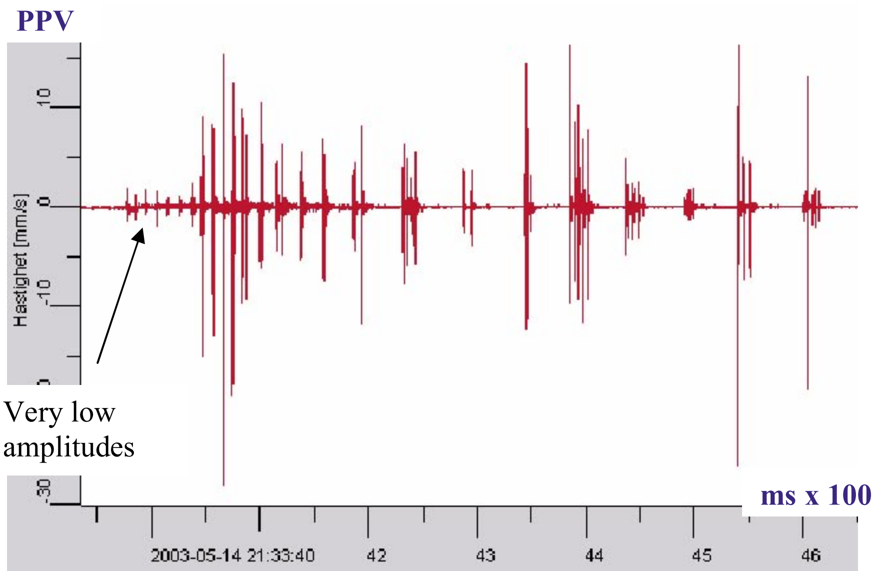


Figure 6-3. *The time history for blast round no 16. Each wave package in the left part of the time window corresponds to a detonating hole.*

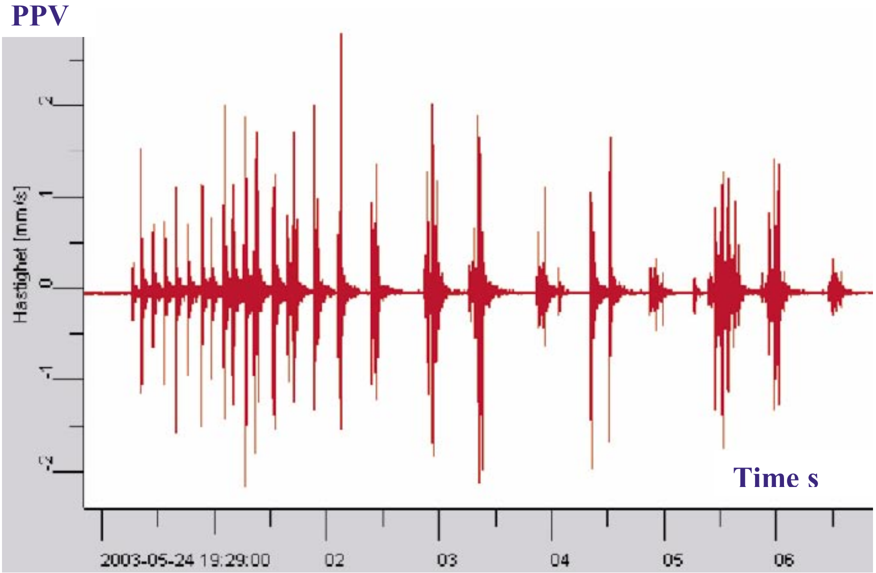


Figure 6-4. *The time history for blast round no 21. Each wave package in the left part of the time window corresponds to a detonating hole.*

Conclusion

According to the registrations available from the TASQ tunnel and other experiences from blast control monitoring there are not enough disturbances in the initiation sequence of the cut that would have caused the misfires. Most of the cut holes have demonstrably broken out from near the bottom all the way to the tunnel face.

The drill pattern was designed primarily for very cautious and careful blasting with small charge weights in order to conform to the restrictions on vibrations and fly-rock. Small burdens and distances were used. The drilling pattern should not be the cause of misfires.

The specific charge for a 4 m long blast round is over 2 kg/m^3, which is relatively high compared to a "normal" tunnel round. This fact should reduce the probability for undercharging being the reason for misfires. It is more interesting to study how the charges were distributed along the blast holes.

The geology in the misfire areas is rather consistent with the most of the exposed rock in the TASQ tunnel. Fracturing and lithology varies not significantly in this part of the tunnel. The rock mass quality, expressed as a Q-index is around 135 respectively 70 in the two misfire sections, Figure 2-4. The dominant NW trending fracture set (set 2) is parallel to the major horizontal stress, and perpendicular to the tunnel orientation. The fracturing pattern trended to form slabs of the fragmented rock, rather than irregular shaped block, Figure 3-2. The actual geological conditions may have contributed to the misfires due to the high confinement in the plane perpendicular to the tunnel. If so may the specific charge have been too small.

Eearlier experiences from the excavation of the ramp of the Äspö access ramp show relatively high specific drilling and charge values, which may point at a fairly hard blasted rock mass. A quite heavy bottom charge was necessary in order to achieve a clean breakage but a short uncharged part of the hole was also of great importance in order to break out into the free face. If not, the result was a remaining "curtain" of rock, see Figure 6-5. To meet both these conditions a high specific charge is necessary. Both a large bottom charge and a long pipe charge become necessary.

The charging plan used for the TASQ tunnel was foremost adapted to short rounds and initially therefore the bottom charges were short compared to the pipe charges when the round length increased to 4 m or more. Later on when the restrictions on the vibration levels so permitted the length and weight of the bottom charge was increased.

The final conclusions are:

- The cut worked satisfactory.
- The drilling pattern was correct.
- The specific charge was high enough.
- The bottom charge was probably too small, i.e. too short for blast rounds over 4.0 m.

6.3.2 The effect of clay stemming

As has been mentioned in Section 6.3.1 on "Misfires", it may be relevant to expect the clay plug to take an active part not only in the confinement of the charge preventing fly-rock, but also in causing the rock curtain that has been experienced at several occasions. As can be seen in Figure 6-5 the broken rock is clearly visible behind the rock curtain. This picture was taken from one of the first blasts. In this case the rock curtain did not lead to any complications. It was broken when the next section of the face was blasted.

Figure 6-5. *An example of a rock curtain.*

6.3.3 Scaling

The initial scaling included in the regular cycle gave only small volumes of scaled rock material, which mainly came from the face. Larger volumes were experienced at section 0/049-0/064 due to the rock structure. Vertical structures almost perpendicular to the tunnel axis forming layers broke quite easily, but were only occurring locally.

The real need for roof scaling appeared soon after the round was blasted. This occured especially at the face and when the rock structure was vertical and perpendicular to the tunnel (fracture set 2) which may indicate an opening of the normally closed fracture set due to re-distribution of the stresses.

6.3.4 Transition zone

The change over between the top heading and the bench, where the tunnel floor changed shape, was made with three rounds in the transition zone. The first round, no 20, with a flat floor was a normal top heading but with a higher roof section. At round no 21 (and 22 because of misfire) the upper part of the arched bench was excavated in the same round as the top heading. Round no 23 was the first top heading above the bench. Round no 35 was the first bench blast only having holes in the lower part of the arched section, see more in Figure 6-6.

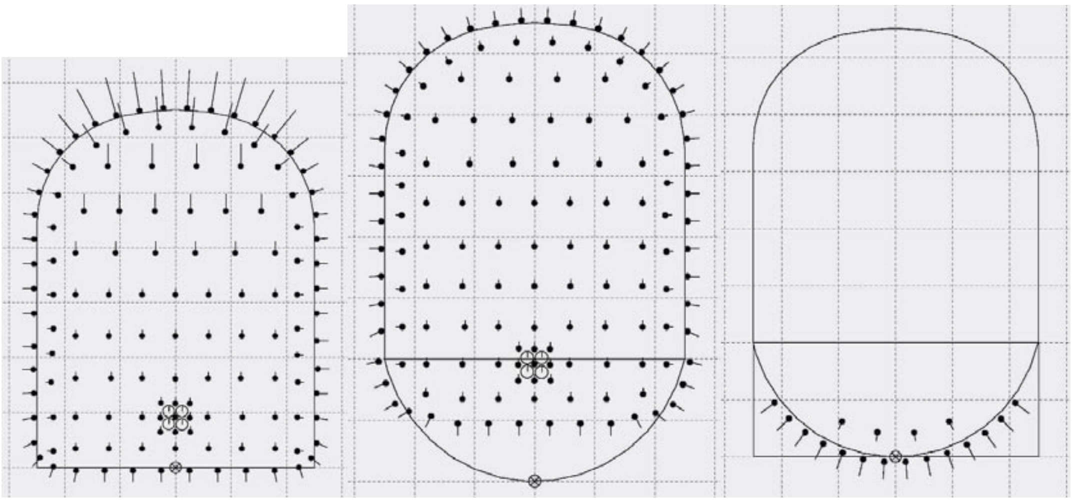


Figure 6-6. The figure shows at left, round no 20, section 0/034 and in the middle round 21 and 22 at section 0/037. The first bench round no 35 at section 0/037 at right.

6.3.5 Zone on the walls between the top heading and the bench

Visible signs on the tunnel walls clearly show that the transition zone between the top heading and the bench is more affected by the blasting than the rest of the wall. Figure 6-7 shows the transition zone. The lower corner holes of the top heading must be charged as contour holes. The bottom corners of the section were chamfered in order to reduce the confinement of the corner holes. Despite these measures the wall seem to have been affected although only a more thorough investigation of the zone would show if blast damage has penetrated further into the wall than the blast damage below and above the zone.

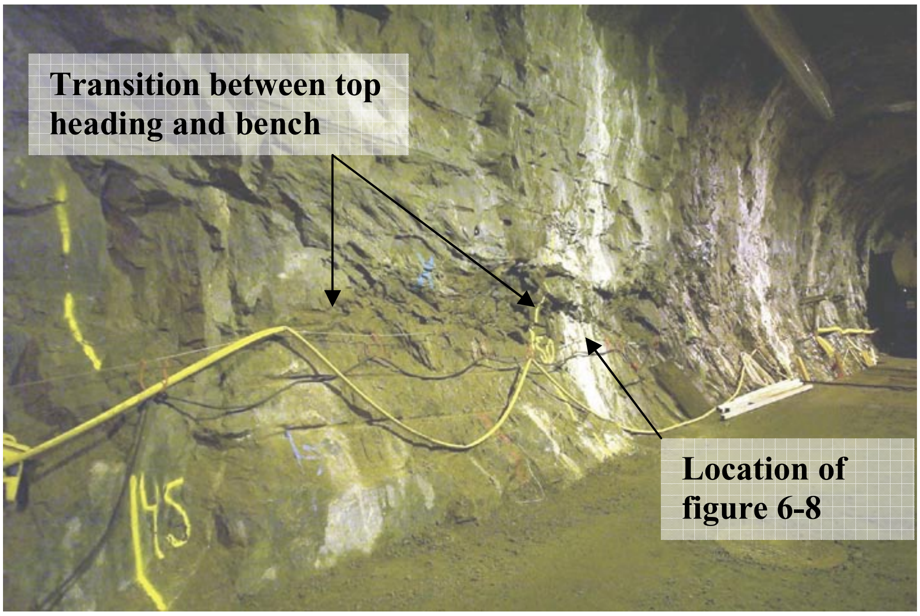


Figure 6-7. The visible signs of the transition zone on the tunnel wall between the top heading and the bench on the left hand side wall.

***Figure 6-8.** A grout hole at section 0/050.7, location is indicated in Figure 6-7. The white slab in the bottom of the borehole is cement from the grouting. The hole was not fully filled with the grout slurry, and the cement has sedimented in the hole. Borehole diameter is 64 mm.*

An scenario is that boreholes from the first grout fan have affected at least part of the blast results. The tunnel face at section 0/050 (round no 25) was the starting position of the first pre-grouting, /Emmelin et al. 2004/. One borehole for pre-grouting was observed being not sufficiently filled with grout, Figure 6-8. The presence of an open grout hole during drilling for a blast round may influence the drilling precision, or provide alternative possibilities for the gasses to fragment the rock more uncontrolled. This however does not explain the first half of the transition zone in Figure 6-7, but indicates a possible cause for local excavation damage.

6.3.6 Water filled blast holes

Water filled blast holes reduces or even eliminates the effect of decoupling. If the decoupling effect is a part of the blast design in order to keep blast damage as small as possible the presence of water may effectively ruin this design.

In general, no water was present in the top heading holes. However, in some cases drilling water may have filled some holes.

In the last bench round, blast no 46, the bottom holes were water filled. Therefore, a saw cut in this round should be compared with a bottom cut from another round blasted with dry holes.

6.3.7 Specific charge

The specific charge, kg of explosives/m^3 of excavated rock, can be calculated in different ways. The total amount of explosives for a complete project can be used. This calculation includes all re-blasts and all spillage. The use of explosives in one round or in a number of rounds can also be used to represent the specific charge. The specific charge can be presented as the total weight of all explosives or it can be normalised to one reference explosive taking into account the density and some measure of the strength of the explosives. In Table 6-3, the specific charge is presented in three different ways.

The specific charge, kg of explosives/m^3 of excavated rock, has been high in the TASQ tunnel. Top heading rounds have been consuming 2–2.5 kg/m^3 and the bench rounds 1.2–1.3 kg/m^3. This fact could be seen as contradictory due to the nature of the tunnelling. Although the tunnel is blasted in a careful and cautious way the consumption of explosives is high. One explanation could be that since the dense drilling pattern distributes the explosives more evenly in the rock mass fine fragmentation will be a result of the blast. Fine fragments means higher energy input compared to coarser rock material and this could be caused by a higher specific charge.

A comparison with the 18 m^2 Zedex tunnel (Zone of Excavation Disturbance Experiment) undertaken at Äspö at the –420 m level in 1994–1996 shows a similarity in specific charge. For location, see Figures 1-1 and 6-10. The tunnel, shown in Figure 6-9 was excavated with focus on cautious blasting and on testing methods for quantifying the damaged zone around the tunnel. The specific charge for the Zedex tunnel varied between 2.25 and 2.5 kg/m^3 indicating that the explosives consumption for the TASQ tunnel was not deviating from smooth blasted tunnels in the same area.

Table 6-3. Three different ways to calculate specific charge.

Calculations	Specific charge kg/m^3
Total amount of all explosives	2.8
Re-blasts excluded	2.5
Normalised to Dynamite	2.4

Figure 6-9. *The circular shaped Zedex tunnel has a diameter of 5.0 m.*

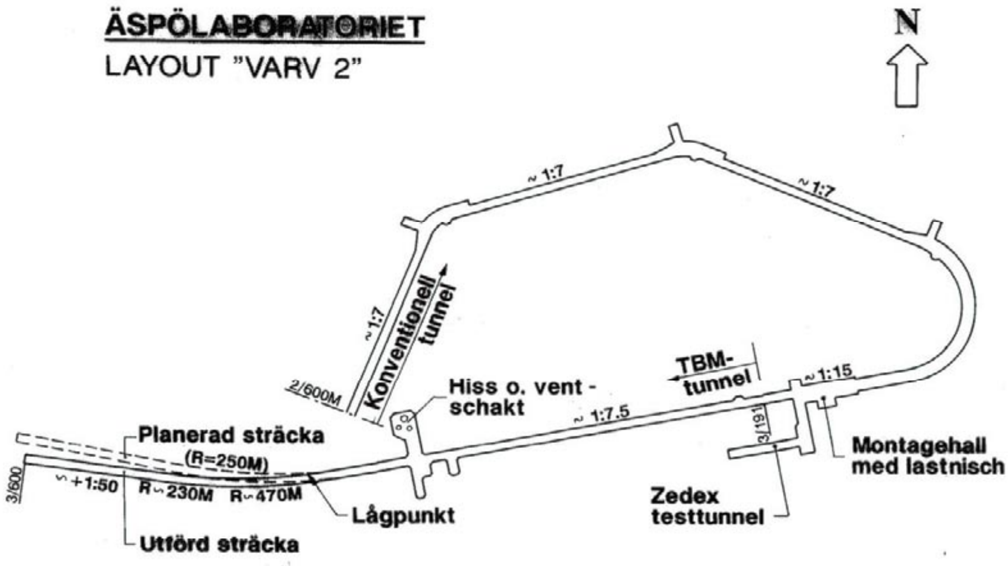


Figure 6-10. *The last lap of the Äspö access ramp and the Zedex tunnel.*

When the ramp in Äspö was excavated in 1995 the area of the ramp was 25 m^2 in the straight parts and 43 m^2 in the curved (from section 2/600 down to the assembly hall for the TBM). The specific charge based on theoretic volume and consumed amount of explosives varied from 2.15 to 2.33 kg/m^3 niches included, see Figure 6-10.

6.3.8 Electronic detonators

Electronic detonators were used in the contour holes of the 3 final top heading rounds. Some practical aspects in the handling and hooking up are worth mentioning. The technical information on the electronic detonator is described in Section 5.8.2 of this report.

When connecting each detonator to the harness wire a small snap-connecting block is used. The block at the end of the detonator wire must first be opened; the trunk line must be positioned perpendicular to the snap connector and finally promptly closed. The tolerance in positioning the trunk line wire in the connector is fairly small. The combination of reduced illumination and protection gloves makes the operation somewhat insecure. The risk of getting conducting substances like nitrates from explosives is probably high. A more practical connector would certainly ensure a safer hook-up procedure. Each detonator had to be tested before connecting the next one. This procedure was time consuming.

6.3.9 The Bever control system

The drill rig Jumbo used in the TASQ tunnel was equipped with the Bever guiding system. This system has been described in section 5.7 of this report. One of the functions of this system is to guide the operator when a new collaring of a drill hole is made. The Bever system gives the operator information on the position of the drill bit and the alignment of the feeder beam. He also sees the actual point of collaring on his monitor, see Figure 6-11. The drill bit position and alignment of the feeder beam is presented as a vector on the screen. Theoretically, the operator will find his correct starting position and feeder angle by superimposing the drill plan vector with his actual rig vector. However, the system should be used with a few precautions. Due to the mechanical characteristics of the rig system – carrier, boom, feeder, hydraulics etc the accuracy of the system is fairly good but not excellent. No guidance system can today meet the demands from many of the Swedish

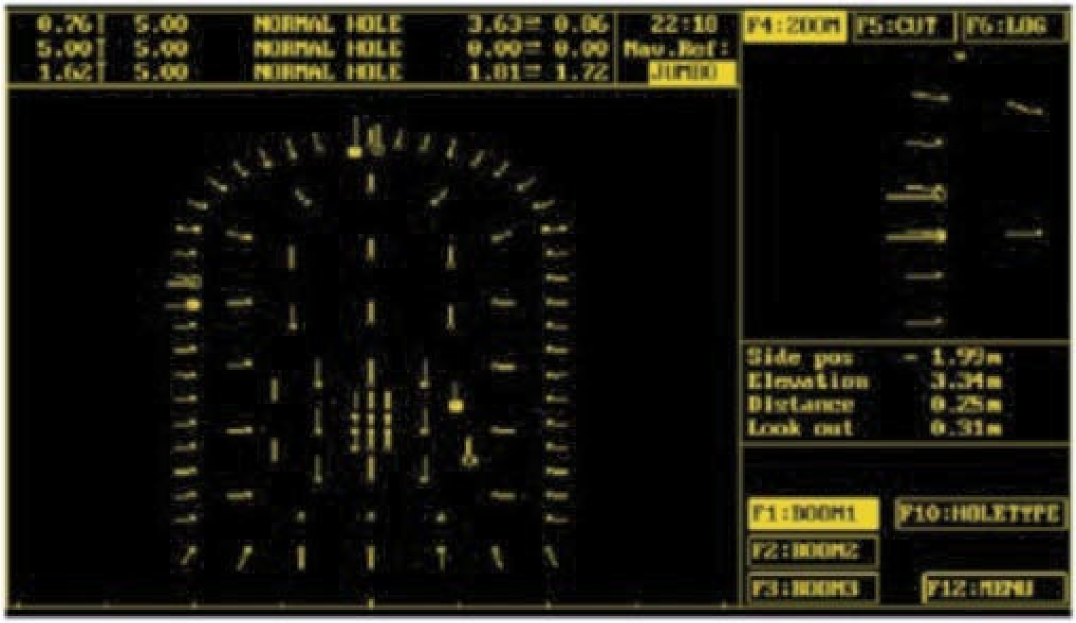


Figure 6-11. *The drill plan as displayed for the operator.*

owners of tunnel contracts regarding drilling accuracy. The reason for such high demands may originate in the conviction that these systems are correlated to the demands, which is not true. Manufacturers are aware of this situation and improvements are to be expected in the near future. Atlas Copco presented an article in Tunnels & Tunnelling in July 2003 giving their view on "The use and misuse of the logging system".

The Bever logging system gives a presentation of how each hole is drilled in the round and it also includes the in-accuracy of the rig system. This must be taken into account when studying the bore logs from the TASQ- or any other tunnel.

6.3.10 Contour drilling of the bench

The requirement for an increased drilling accuracy in the floor row raises new demands on drilling equipment and operator skills. Normally the look-out angle for the floor holes is allowed to be larger than for the wall and roof holes. The allowed zone of damage is also larger. In the TASQ tunnel the same condition was valid for all contour holes including the floor holes. The difference in drilling floor holes compared to other contour holes is that the drill steel is not visible to the operator. In order to make the collaring of the hole as close to the previous floor as possible the feeder beam must be positioned up side down with the drill steel facing down towards the floor. As long as the on-board systems for positioning and alignment of the drill holes are lacking the high degree of accuracy needed for a fully non-manual drilling, the result relies on the skill of the operator. When the operator looses the possibility of a visible contact with the drill steel, it will also reduce the chances of reaching a proper result. This situation definitely calls for better guidance tools in the future.

The smaller look-out angle will create problems with preventing water from filling the bottom holes. It will reduce the volume of the pool in front of the tunnel face and a smaller volume of water could be contained. The distance between the collar of the hole and the water surface is reduced. As a consequence dewatering at the tunnel face must be improved.

6.3.11 Vibration control

A transformer approximately 16 m from the portal and a firewall to the elevator floor approximately 10 m from the portal had vibration restrictions stated in the tender dossier. The firewall is made of lightweight concrete blocks. Basically, the size of the round was not limited by low imposed limitations in vibration levels but more by the needs involved in the protection of fly-rock and air shock wave damage. On a few occasions the measured vibration level on the transformer exceeded the vibration limit. A separate report treats the vibrations experiences of the project /Nyberg et al. 2005/.

6.3.12 Quality system

The contractor's quality system is normally reviewed by an internal audit. In the APSE project the work in the tunnel was based on a system where the operators regularly check their own work mainly by signing check lists and reports. Furthermore, the project and production managers where generally in the tunnel during drilling and charging. The project was also supported by technical staff especially during the more critical operations. Internal control was therefore more extensive than in the normal project. Skanska was also subjected to a Quality Auditing by SKB.

6.4 EDZ Experiences – Blasting of the test sections

Due to lack of Nonel caps the planned interval times for the contour holes had to be re-planned. The actual bore plan from the log and the real interval settings are shown in Figure 6-12 to 6-14.

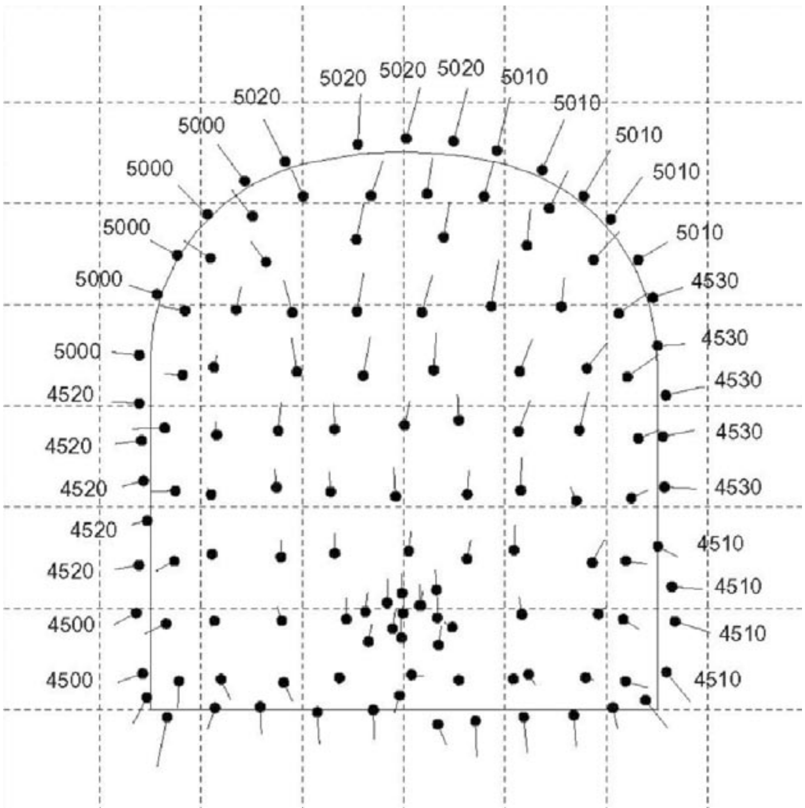


Figure 6-12. *Hole trajectories and initiation of the contour holes in Test section 1; round 32.*

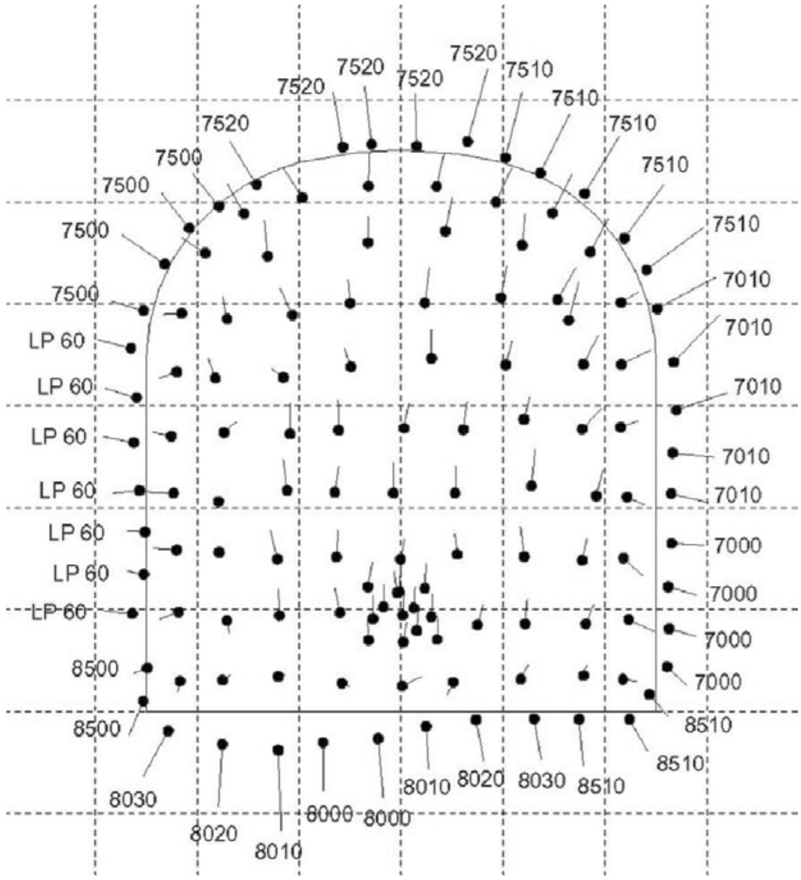


Figure 6-13. *Hole trajectories and initiation of the contour holes in Test section 2; round 33.*

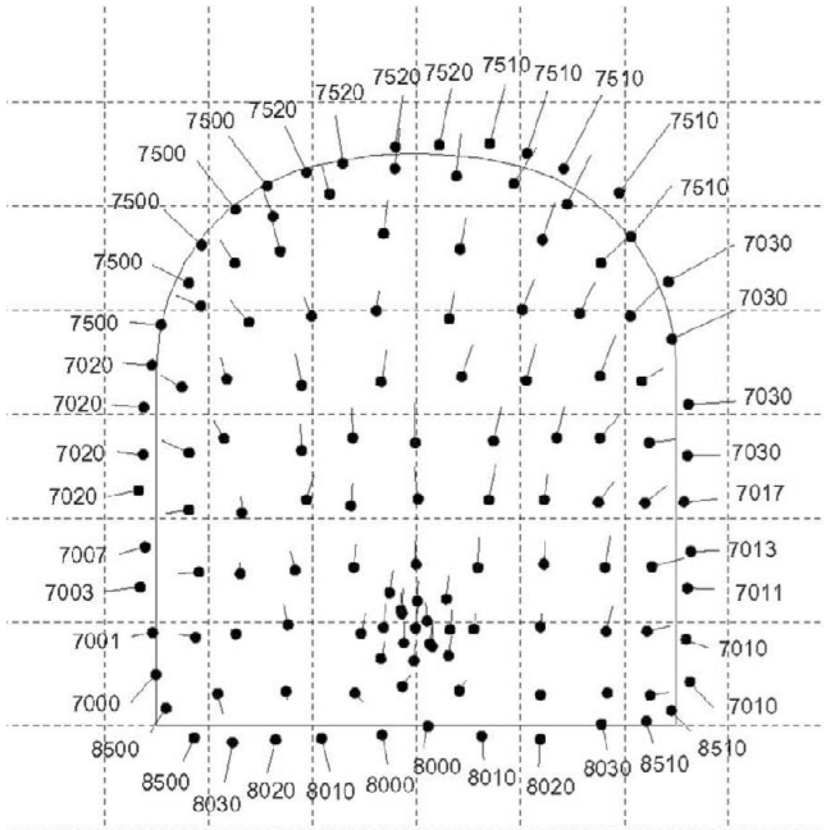


Figure 6-14. *Hole trajectories and initiation of the contour holes in Test section 3; round 34.*

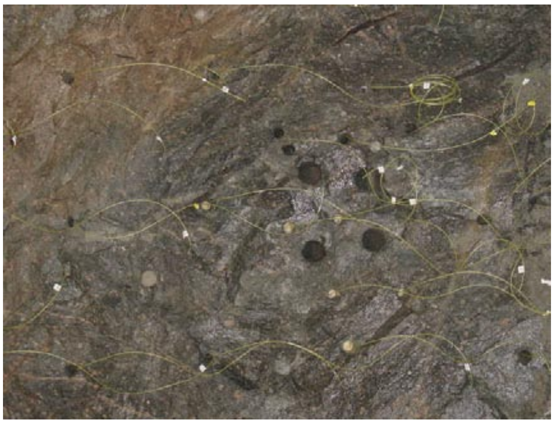


Figure 6-15. Cut holes.

Figure 6-16. Programming the caps.

Figure 6-17. Connecting the electronic caps.

Figure 6-18. Connecting all caps.

Figure 6-15 shows the cut holes for one of the test rounds. The electronic caps were programmed after being put into the holes as shown in Figure 6-16.

All the Nonel tubes were collected in bunches and the bunches were connected by a loop of detonating cord. A starting electronic cap initiated both the detonating cord and contour holes. Figure 6-17 and 6-18 shows the connection of the caps.

6.4.1 Test round 1

No changes from the planned initiation were made see Figure 6-12. The accelerometers were grouted in a hole placed in the right hand wall (facing the tunnel face) at a distance of 19.5 m from test round 1. The cables from the gauge hole were placed inside a steel tube as a protection against fly rock.

Blasting

As the round was to be fired there was an error message from one of the electronic caps in the contour, (hole no 23 in the upper left hand wall). The round was fired and after mucking the non-detonating explosive and the cap could be removed. The cap was sent to Orica for examination.

The result of the blast was good from a fragmentation point of view. However, the amplitud of the air blast was higher than normal, probably due to explosives replacing stemming material in the blast holes.

In most of the previous rounds the rock was better on the right side. However there were some slickensides due to the NE trending fracture set (set 1) on both sides of this round. Therefore the numbers of half casts in the wall was lower than expected. The highest number of half casts was found in the roof.

6.4.2 Test round 2

Due to lack of Nonel-caps more electronic detonators had to be used. Other interval numbers than the planned were also used as Figure 6-13 shows. The interval time for the electronic caps here starts at 7,000 ms compared to planned 5,000 ms but the time steps between the contour holes were not changed. Electronic caps were also used for the bottom holes (in the lifters). The Nonel LP caps were used as planned in on the left hand wall but caps number 45 (4,500 ms) were replaced by caps number 60 (6,000 ms).

Blasting

There were some error messages as this round was to be fired too. One of the electronic caps had too low a voltage and the error message from the other cap was "Not programmed". The round was fired and after mucking the non-detonating explosive and caps could be recovered. The caps were sent to Orica for examination.

The result of the blast was good. However, in addition to the problem with the two electronic caps, there were also two non-detonated Nonel initiated holes in the left hand wall.

Figure 6-19 shows the half casts in the roof after blasting of this round and Figure 6-20 shows the half casts in the right hand wall. The highest number of half casts was in the roof. The right hand wall with electronic caps had much higher number of half casts than the left hand wall charged with Nonel initiated explosives.

Figure 6-19. *Round 2. Half casts in the roof. Notice spacing variation.*

Figure 6-20. *Half casts in the right hand wall of round 2.*

6.4.3 Test round 3

The initiation plan of this round was also changed due to lack of Nonel-caps. Instead of starting the contour intervals at 4,500 ms as planned, the starter hole was given the initiation time 7,000 ms, see Figure 6-14. The time steps between the contour holes were not changed. Electronic caps were used for the bottom holes.

Blasting

Due to the number of misfires in round 1 and round 2 Orica replaced the I-kon blasting control unit. After the change there were no misfires.

The result of the blast was very good with the highest number of half casts of all the three rounds; see Figure 6-21 which shows the result in the roof.

It should be noticed that the long look-out distances in the right abutment in general did not result in abnormal overbreak. The increased look-out was drilled on purpose in order to compensate for hole deviation due to geological conditions.

This shows that a modern positioning and alignment system is not a guarantee for high drilling accuracy.

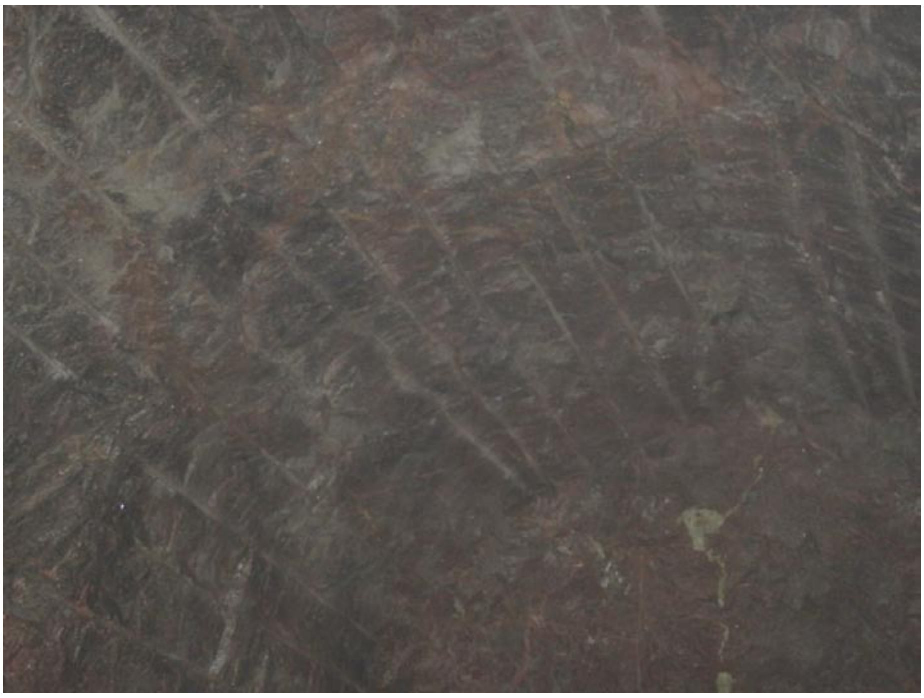


Figure 6-21. *Half casts in the roof of round 3.*

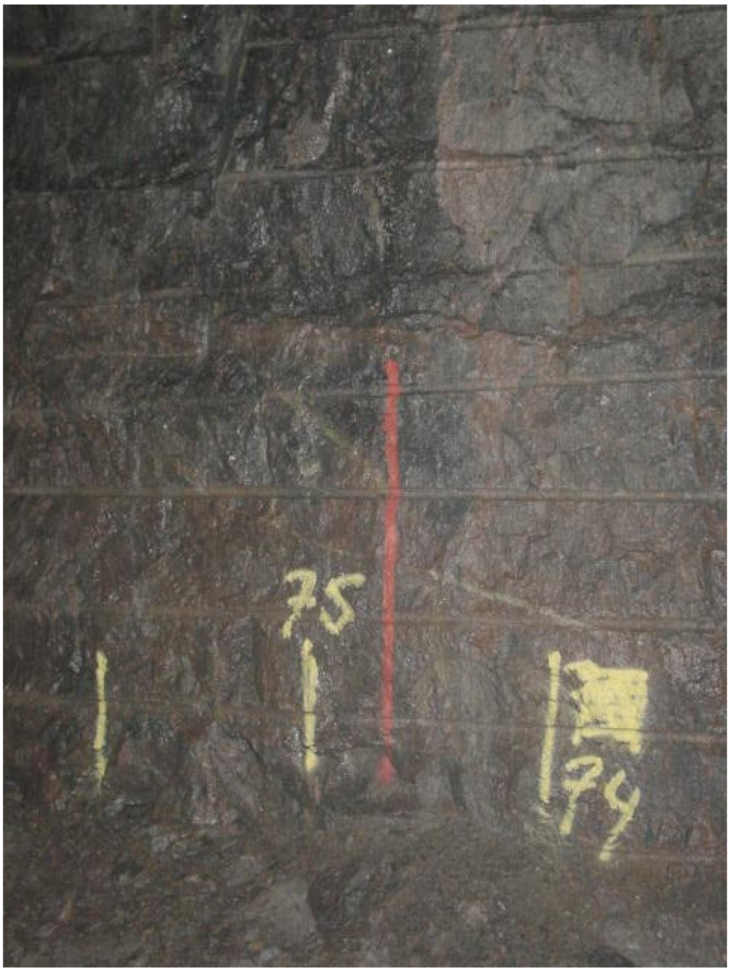


Figure 6-22. *Right hand wall from test round 2.*

7 Results

7.1 Profiling/overbreak

The tunnel contour was surveyed after completion. Some results are shown in Figure 7-1. The contour of the lower part of the floor is not correct, because some muck was still left on the floor up to section 0/060 when the profiling was carried out. Sections 0/040 and 0/045 are situated in the inner parts of rounds, the other ones within 2 m from the collar of a round. The difference in over-break is due to the look-out angle. The transition zone between the tunnel and the bench shown in Figure 6-7 is visible in the lower part of the left wall in sections 0/045 and 0/050. The "undulating" contour of the left hand wall in section 0/070 is visible on the wall, it is obvious that the contour drilling was not aligned for that round. It is also indicative that there is a tendency for over-break in the left abutment in several sections.

To further study the accuracy in drilling precision all visible half pipes in the walls were surveyed at the starting and end points. The difference between the start- and end point was calculated. The results are presented in Figure 7-2. The absolute over-break in the walls is not possible to see, because the collar position relative the theoretical contour is not considered. Since all half pipes are not visible through the full length of a round the graph in Figure 7-2 under-estimating the true over-break in the walls to some degree. Because the large number of visible half pipes is this error probably small. This study could not include the roof, because it was covered by shotcrete. However, Figure 7-1 indicates that the overbreak in the roof is less than in the walls.

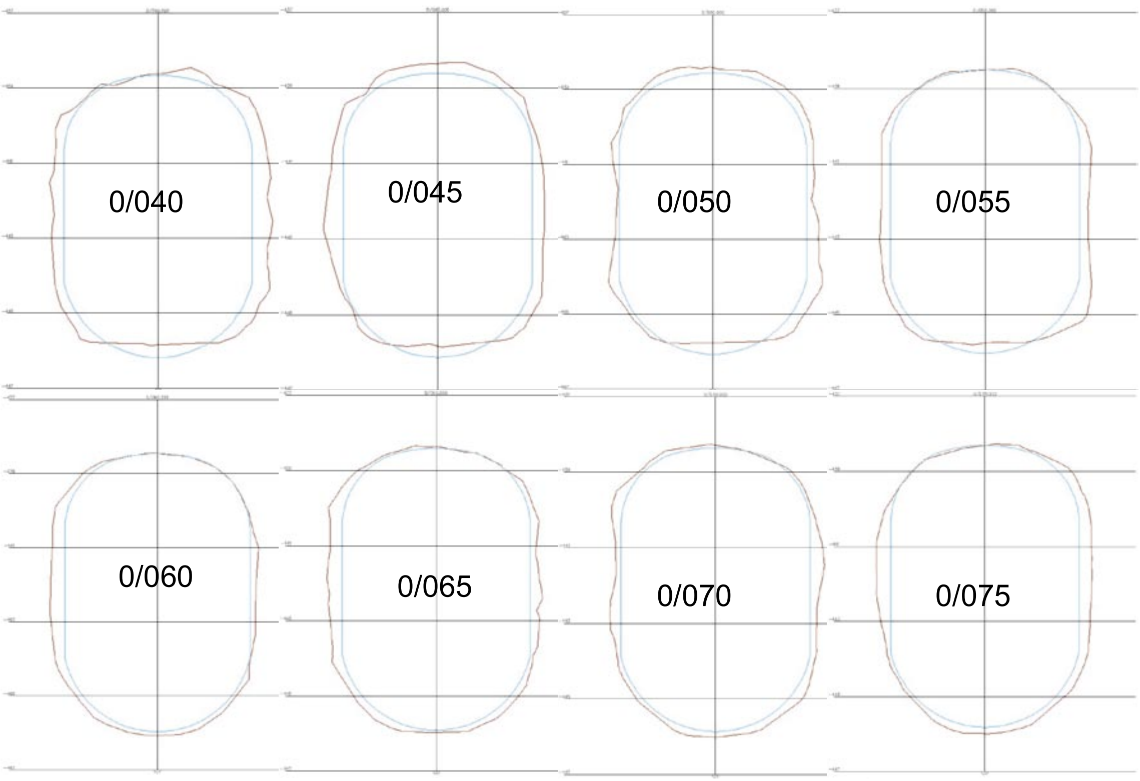


***Figure 7-1.** Results of profiling of the tunnel contour.*

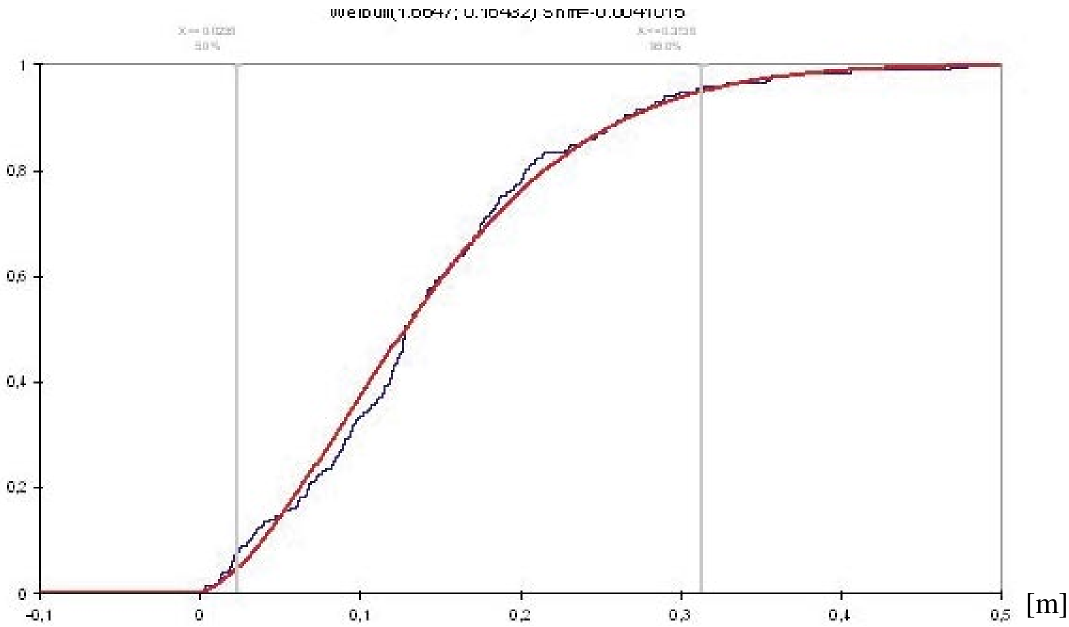


__Figure 7-2.__ Cumulative distribution of look-out angle for visible half pipes in the walls. 0.3 m was the maximum acceptable over-break according to the requirements. 94% of the visible contour holes in the walls stayed within this limit.

7.2 Observed damage in cores

Soon after the tunnel was completed 13 cores were drilled in the floor for detailed planning of the APSE experiment, and for instrumentation. The bore holes were 6–7 m deep and primarily located around sections 0/064 to 0/066 m. The core were logged and sonic velocities were measured /Staub et al. 2004/. Possible induced fracturing was found to a depth of normally not more than 0.3 m. However, it is not possible to determine if observed fractures in the 51 mm cores are induced, or if they consists of natural horizontal fractures that are somewhat sheared because of the high stresses in the floor. Examples of the results are given in Figure 7-3.

In addition, when drilling the two 1.8 m diameter holes in the floor for the APSE experiment /Andersson, 2003/, the upper part of the floor could be studied. The observations from the large size bore holes supports the observations from the 51 mm cores, Figure 7-4.

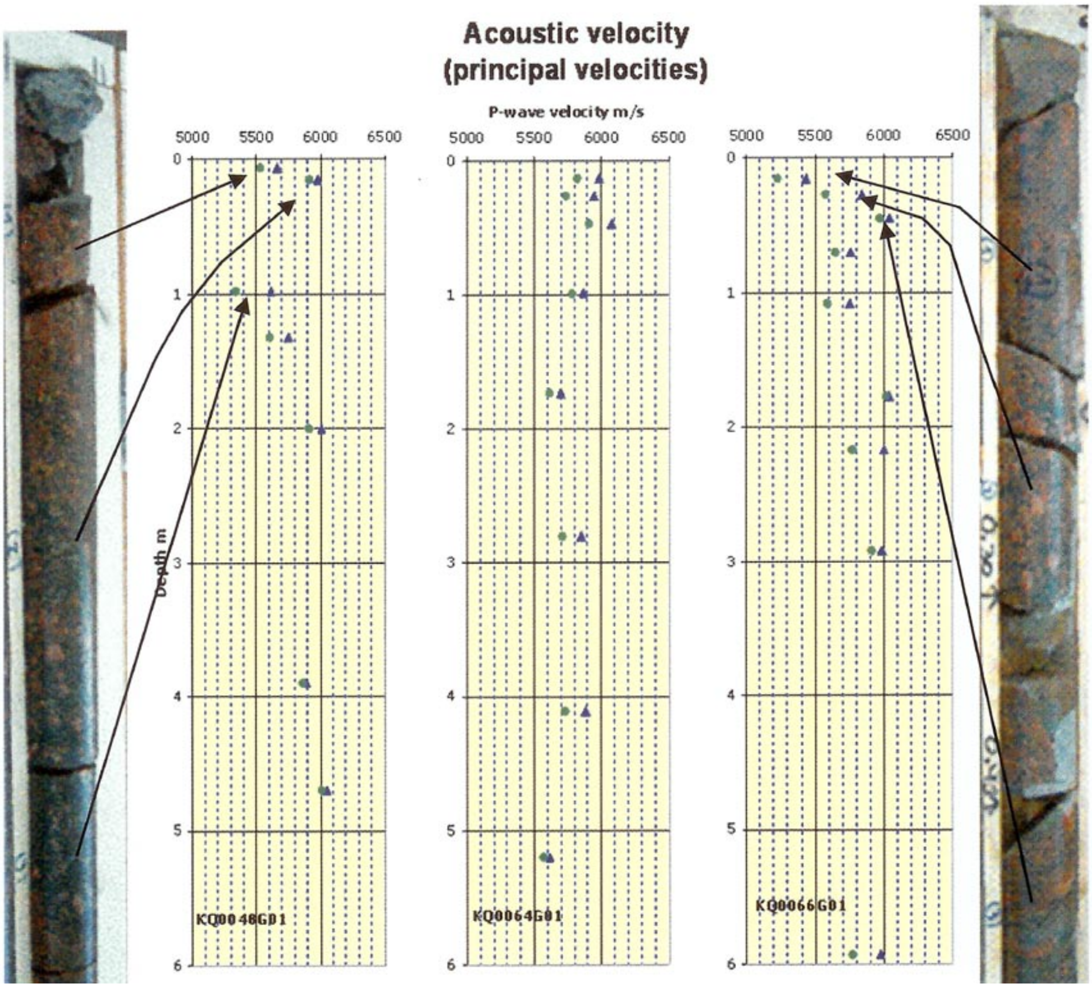


Figure 7-3. *Two cores from the floor of the TASQ tunnel with results of measured sonic velocity as well. There are no indications for damage below approximately 0.3 m /Bäckblom et al. 2004/.*

Figure 7-4. *Observations of the upper part of a 1.8 m diameter bore hole. The upper part is a concrete slab that was casted to provide a smooth surface for collaring of the bore hole. The slab is approximately 0.3 m thick.*

7.3 EDZ

7.3.1 Testing technique for EDZ

The technique used has been developed by SveBeFo /Olsson and Reidarman, 1995/ and consists of cutting 0.5 m deep slots in the tunnel wall or floor. The slots are made by sawing a number of parallel cuts into the rock. Normally two horizontal cuts and 4–5 vertical cuts have to be sawed. The distances between the cuts are adjusted to the force required to loosen the rock between the cuts. Then the slices are priered lose by wedges and removed. Finally a dye penetrant is sprayed on the cleaned surface causing the cracks to appear very clearly.

7.3.2 Cracks in general

For the identification and measuring of blast induced cracks there has to be at least one visible half-pipe. A typical crack pattern is shown in Figure 7-5 where there are some cracks originating from the half-pipe (blasting cracks) and others cracks, induced or natural, inside the rock mass. The "induced" cracks in this study are cracks that are likey not natural. but does not be in contact with a half-pipe of the contour holes. They could be blast or stess induced. The lengths of the blasting cracks are influenced by the explosive itself, the coupling ratio (diameter of charge/diameter of hole), the delay time, spacing, natural cracks, the rock and presence or absence of water.

Water in the hole might increase the length of the cracks by a factor of 3–4 times /Olsson and Ouchterlony, 2003/. Induced cracks are cracks starting near the tunnel contour but not originating from the bore hole itself. These cracks are probably caused by the high stresses and released by the blasting process or by the redistribution of stresses due to the excavation of the tunnel. The natural cracks are determined by the geological conditions. They could be opened or filled by some minerals. During the blasting process these cracks might expand. The natural cracks often have the same direction.

In order to use the penetrants successfully the slots have to be comparatively dry which often is the case for the walls but could be tricky in the floor.

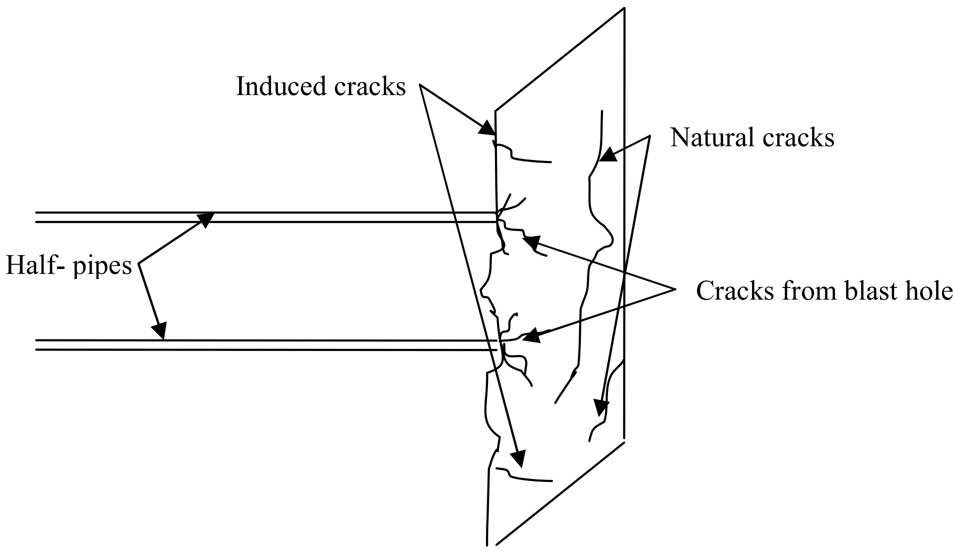


__Figure 7-5.__ Typical crack pattern.

7.3.3 Locations of the slots for studies of the EDZ

The results from blasting together with estimated costs for sawing ended up with a proposal of 5 slots in the wall and one slot in the floor. The different slots, their location and the purpose of the slots are shown in Table 7-1.

It was planned to do all the testing on one occasion but due to very high stresses the rotary saw got stocked during the first cut in the floor. To distress the section some vertical holes were drilled in the floor on both sides of the tunnel, see more in chapter 7.3.4; floor section.

Table 7-1. Locations of saw cuts.

Section	Round no	Initiation type	Tunnel side	Bench side	Purpose
Test section 1	32	Electronic	Right		EDZ at the end of a round EDZ near a bottom charge
Test section 2	33	Electronic	Right		EDZ at start of a round EDZ near a bottom charge
		Nonel		Right	EDZ from a bench hole
Test section 3	33	Electronic	Right		EDZ from a column charge
		Nonel		Right	EDZ from a bench hole
Test section 4	34	Electronic	Left		EDZ from a column charge
Transition zone	24	Nonel	Left		EDZ in transition zone
Horizontal	33	Electronic	Right		EDZ // to the tunnel
Floor	43	Nonel			EDZ in the floor

7.3.4 Testing EDZ

Test section 1, round 32

A slot was made in the right hand wall of the tunnel at the end of round 32. The right wall of the slot shows the crack pattern from the column charge and the left wall shows the corresponding picture from the bottom charge, see Figures 7-6 and 7-7. Three half-pipes are crossed by the slot. All of the holes were initiated simultaneously.

Figure 7-6. *Right slot wall.*

Figure 7-7. *Left slot wall, upper part.*

Generally there was too much of water on these surfaces for a proper analysis. However, on the left wall the influence of a confined hole is visible. There are many long cracks from this hole compared to the upper half-pipe with almost no cracks. Another reason for the long cracks is that the hole probable was partly filled with water when blasted. The drill rig in the TASQ-tunnel used water to flush away the cuttings. This hole, together with many other measured boreholes, was not horizontal. They were dipping 1–3 degrees forward and were most likely partly filled with water when they were charged. There are some shorter cracks from the other half-pipes; the longest is around 20 cm. There are also some blast induced cracks and natural cracks. More details may be found in Appendix 3 and in Section 7.3.5.

Test section 2, round 33

A slot was made on the right hand wall of the tunnel at the start of round 33. Three visible half-pipes are crossed by the slot. Figure 7-8 and 7-9 shows the result from the right wall and Figure 7-10 and 7-11 shows the result from the left wall. All of the holes were initiated simultaneously.

In the upper part of Figure 7-8 cracks from a bottom charge are shown. This is a very typical crack pattern from a confined charge with symmetrical cracks all around the hole. There are also a number of cracks from the half-pipes; the longest is around 20 cm. On the left wall, the cracks from the half-pipes have about the same length. But there are also longer cracks; the longest around 40 cm which seems to originate from the half-pipes. These cracks might also be classified as blast induced cracks. Most of the half-pipes in this section pointed downward which could indicate that the holes were partly filled with water when blasted. There are also many natural cracks in this section. Most of them have a more vertical plunge. One of these cracks was an open crack. More details may be found in Appendix 3 and in Section 7.3.5.

Figure 7-8. *Cracks in upper part of right slot wall.*

Figure 7-9. *Cracks in lower part of right slot wall.*

Figure 7-10. Cracks from left slot wall.

Figure 7-11. Cracks from upper part of left slot wall.

Test section 3, round 33

A slot was made on the right hand wall of the tunnel in the first part of round 33. Three half pipes were crossed by the slot and the longest crack from them was some 20 cm. Most of the cracks originating from the half pipes were shorter with crack lengths < 15 cm, see Figure 7-12. Most of the half-pipes in this section also pointed downwards which could indicate that the holes were partly filled with water when blasted. There are also a number of induced cracks as well as some natural vertical cracks but generally there seem to be fewer cracks in this section compared to section 1 and section 2.

Figure 7-12. Cracks from right slot wall.

Figure 7-13. Cracks from the bench, right slot wall.

Figure 7-14. Cracks from left slot wall.

Figure 7-15. Cracks from the bench, left slot wall.

It is also obvious that there are fewer cracks from the bench than from the tunnel, see Figure 7-15. The reason for that could be:

- Another stress situation around the tunnel during the bench blast (blast induced cracks were reduced).
- The bench holes observed in the lower part of the wall seem to be more horizontal (smaller risk of being filled with water).

Test section 4, round 34

The first plan was to make a slot in the right hand wall were there were more half pipes but unfortunately that could not be done. A slot was therefore made in the left hand side of the tunnel in the middle of round 34. There was a lot of water dripping from the roof in this section. Some of the half-pipes had a negative slope and these holes could therefore have been waterfilled when blasted. Figure 7-16, 7-17, 7-18 and 7-19 shows the crack pattern in this section. Three half pipes were crossed and the longest cracks were 50–60 cm long. One explanation might be that some of these cracks have been induced during the blasting as they do not orginate from the bore holes.

However these two long cracks may well be classified as induced cracks. Above the folding rule in Figure 7-17 some open cracks could be seen. These cracks probably have opened during the sawing. There are also some blast induced cracks but there seem to be fewer of them compared to Test section 2. There are very few natural cracks in this section.

Test section 5, Round 24, Transition zone

This slot was made in the transition zone between the tunnel and the bench in the left hand wall. This area seems to be more affected by the blasting than other parts of the tunnel, se more in Section 6.3.5. There are no half-pipes in this area so the slot had to be extended to cover one half-pipe above the zone and another half-pipe under the zone.

Figure 7-16. Cracks from the right slot wall.

Figure 7-17. Open cracks in the right slot wall.

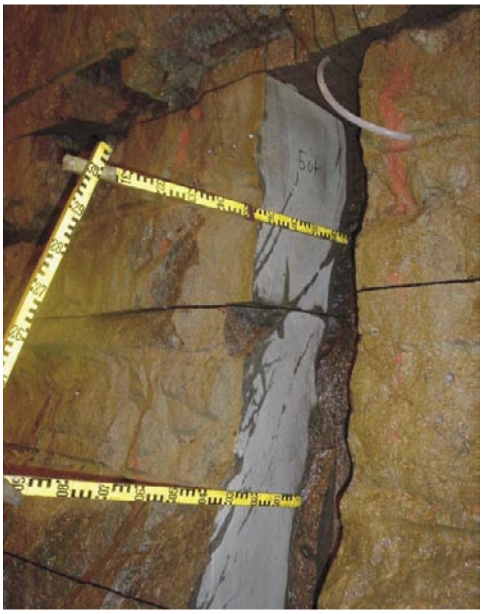


Figure 7-18. Cracks from the left slot wall.

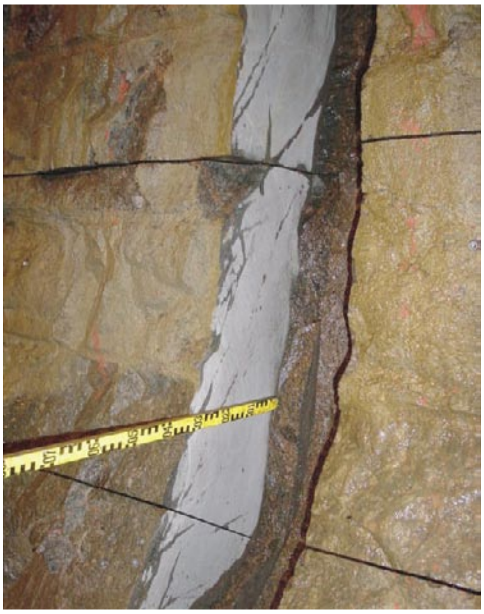


Figure 7-19. Cracks from the left slot wall.

In the transition zone all of the holes were initiated by Nonel. There are two half-pipes one from the tunnel and the other from the bench, see Figure 7-20 and Figure 7-21. There are no visible cracks from the half-pipes in the tunnel part. From the half-pipe in the bench part are two visible cracks with a crack length less than 10 cm. Blast induced cracks as well as natural cracks are common in this section, see Figure 7-22 and Figure 7-23. The number and lengths of cracks in this transition section doesn't differ from the situation in the other test sections.

Figure 7-20. *Cracks from right wall.*

Figure 7-21. *Cracks from the tunnel and bench.*

Figure 7-22. *Cracks in the left wall.*

Figure 7-23. *A zoom in of the upper part.*

Test section 6, Horizontal slot

A horizontal slot was made in the right hand wall of the tunnel between test section 2 and 3 and the purpose was to find out if there is a continuous damage zone due to blasting parallel to the tunnel wall. This slot was made very narrow with only two horizontal cuts.

There were a number of parallel cracks going from the surface and into the wall, see Figure 7-24. The distances between these cracks are approximately 10 cm. Some of these cracks are found also on the tunnel wall. One of the cracks shows up as a filled thin zone on the wall.

Figure 7-24. *The horizontal slot.*

Figure 7-25. *Vertical right cut.*

Figure 7-26. *Vertical left cut.*

There are also some blast induced cracks in the vertical cuts in Figure 7-25 and Figure 7-26 like the cracks in the previous vertical cuts. The crack pattern from the upper surface is different, see Figures 7-27 and 7-28. There are many cracks near the tunnel surface, a long parallel to the surface and a shorter one pointing into the slot. These cracks should also have been found on the lower surface but this area probably was sliced away by the cutting of the slot. The system of parallel cracks going into the wall that was found on the lower surface could not be found on the upper surface. There are only a few cracks going further into the slot and these cracks are not parallel to each other.

The crack pattern in this horizontal slot shows no indication of a continuous damage zone parallel to the tunnel wall.

***Figure 7-27** and **Figure 7-28.** The upper surface of the horizontal slot.*

Test section 6, Floor

The slot was cut in the floor of bench round 43, see Figure 7-29. This was in the area with the two vertical drilled experimental holes. In this area the tunnel was free from the road bed. Due to very high stresses the saw weel got stuck in the floor during the longitudinal cutting. A number of holes then had then to be drilled on each side of tunnel to destress the area. Five vertical slices were cut from the floor and in three of them the cracks could be examined, see Figure 7-30 to 7-32. Up (the floor) is towards the ruler in the figures. The other two slices broke up in smaller parts.

Slice no 1 fell apart into two parts. To the left of the upper part some concrete could be seen. There are two vertical cracks with a length of 8 cm and 12 cm and some horizontal cracks. In the lower part there are more cracks both horizontal and vertical. All of the cracks seem to be induced or natural cracks. From the second slice (Figure 7-31) there are fewer cracks. Notice that this slice is facing the other way. There are no visible cracks from the half-pipe (to the left of the upper slice). The right hand side of this part shows a layer of concrete. In the lower part there are some horizontal natural or induced cracks, some of them seem to have been open.

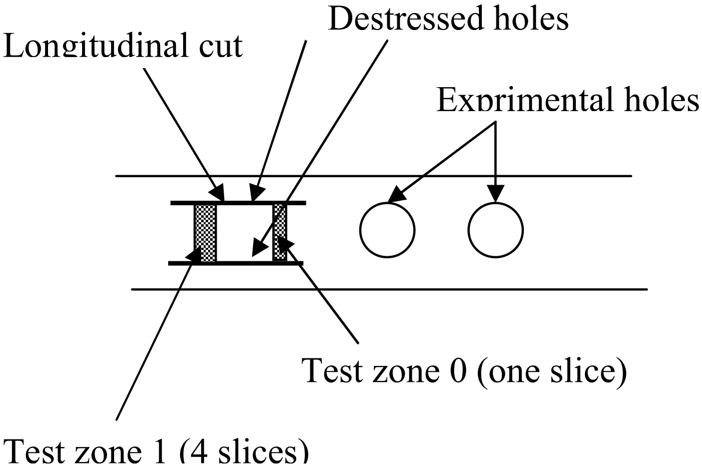


***Figure 7-29.** Test area for EDZ in the floor (tunnel face to the left).*

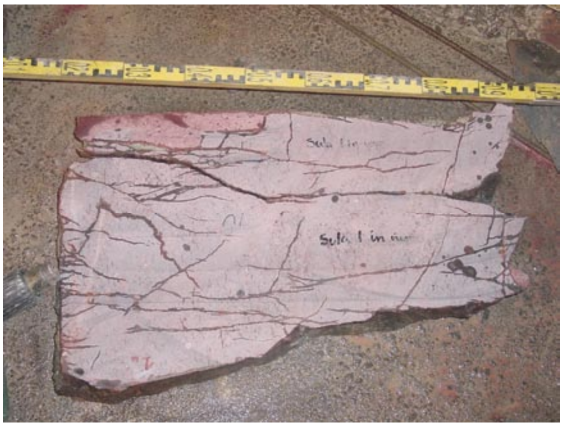


Figure 7-30. *Slice no 1 (first slice from left to right in Test zone 1).*

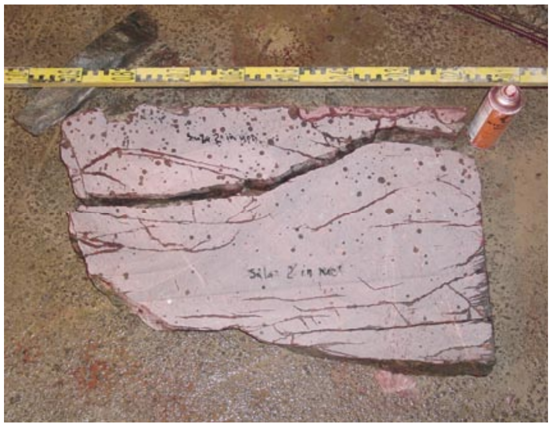


Figure 7-31. *Slice no 2 from Test zone 1.*

Figure 7-32. *Slice no 3 from Test zone 1.*

Figure 7-33. *Slot no 0.*

The crack pattern on slice no 3 in Figure 7-32 looks similar those on the other slices. There are no visible cracks from the half-pipe (to the right of the upper part). There is only one crack going from the surface into the upper part and the length of this crack is 5 cm. In the lower part there are some horizontal natural or induced cracks. From slot no 0 some parallel, nearly horizontal cracks could be seen, see Figure 7-33. There is also one vertical crack but this seems to be a natural crack as it is also visible on the upper surface. There are no cracks emanating from the half-pipe.

A saw cut was also made along the tunnel and the result could be seen in Figure 7-34. There are a few vertical cracks with a length of some 20 cm and also an inclined natural crack. There is no evidence of any continuous damage zone.

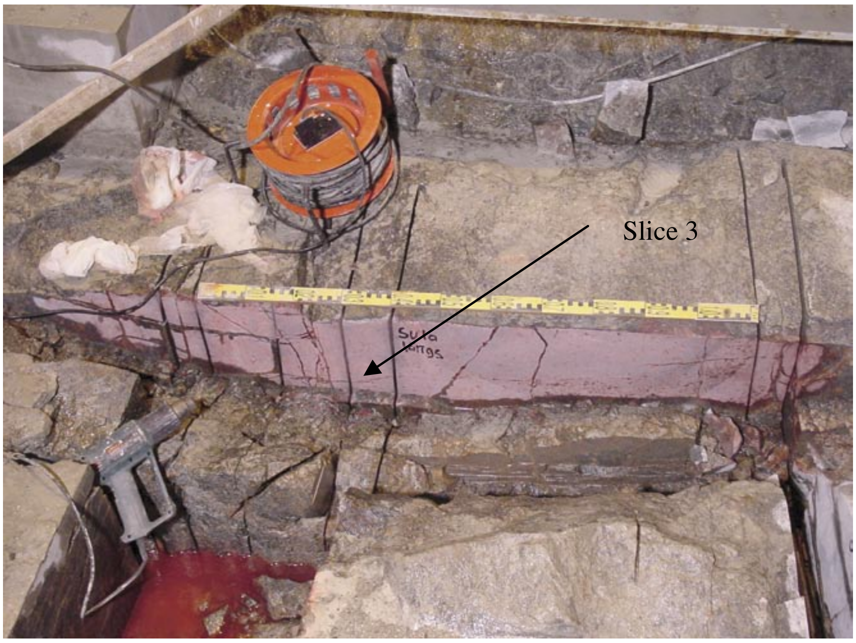


Figure 7-34. A saw cut along the tunnel direction.

7.3.5 Summary of the EDZ-tests

In the EDZ test cracks from 8 slots have been examined (five vertical slots in the wall, one horizontal slot in the wall and 2 slots in the floor). The definition of various cracks observed in this study is given in Section 7.3.2. Lengths of blast induced cracks and their orientations have been measured by Split, a fragmentation analysis computer program with a built in measurement analysis modulus. All analyzed photos use a reference ruler as reference to set the scale. The angle is measured from a theoretical horizontal plane, see Figure 7-35. For this reason the measured angle is dependent on the camera position and the angle of the cracks should not be directly compared to the angle of cracks in another section. It should only be used as a relative indicator.

In Appendix 2 the number of cracks and the average crack lengths are shown as a function of the crack direction. These figures presented should be considered as approximately figures rather than exact. The largest number of induced and natural cracks seems to point in the direction 60–120 degrees. In the vertical slots most of the induced cracks therefore have a diagonal direction (pointing upwards) as Figure 7-35 shows. The cracks originating from blast holes point in all directions.

A summary of the result from the EDZ-tests is shown in Appendix 3. Here the test section, blasthole initiation, type of cracks, length of cracks (maximum, average and scatter) and some specific notes are presented. It is obvious that water in holes strongly affects the crack length. This could be the explanation why electronical detonators seem to cause longer cracks compared to holes initiated with Nonel. A great number of tests in quarries and tunnels, performed by SveBeFo /Fjellborg and Olsson, 1996; Olsson and Ouchterlony, 2003/, have very obviously shown that simultaneous detonation with electronic detonators always creates shorter cracks than Nonel-initiation when the same conditions apply.

Table 7-2 shows a summary of measured crack lengths from column charges initiated by the different initiation methods, used in the TASQ tunnel. For the 17 mm Dynotex charges the average crack length varies from 0 to 22 cm for electronic detonators and from 0 to 10 cm for Nonel.

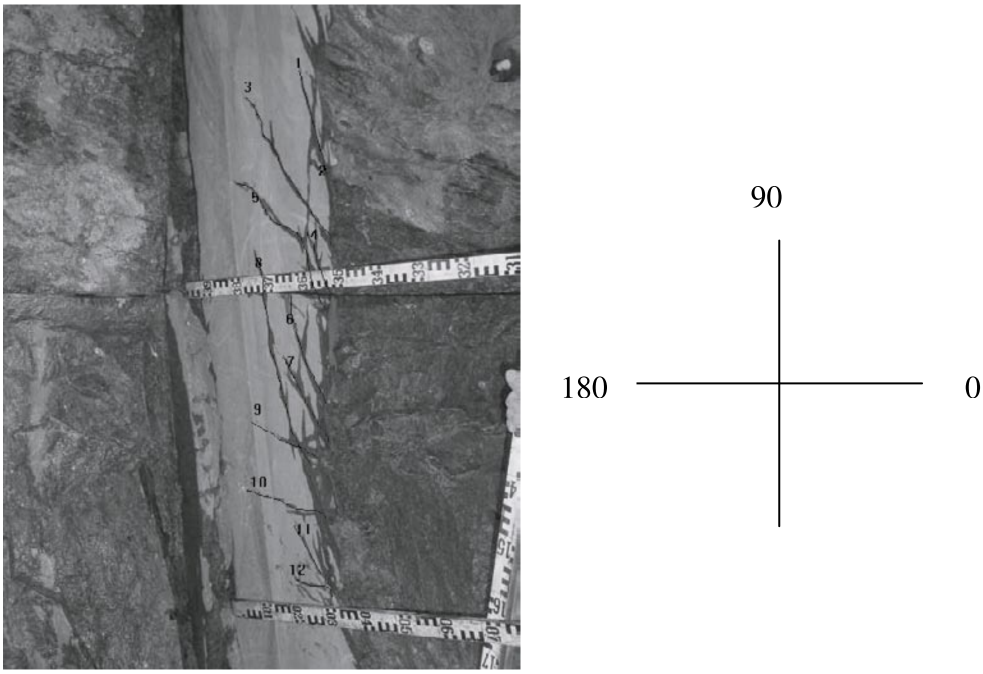


Figure 7-35. How crack angles are measured.

Table 7-2. Blast induced crack length in tested sections and from different initiation.

Explosive	Section	Initiation	Crack length in cm Minimum	 Maximum	 Average
Dynotex 17	Tunnel	Electronic	0	22	14
Dynotex 17	Tunnel	Nonel	0	0	0
Dynotex 17	Bench	Nonel	0	10	5
Dynotex 17	Floor	Nonel	0	0	0

In conclusion of the EDZ tests in the TASQ tunnel could be summarized as:

- Unexpectedly long crack lengths were obtained from holes simultaneously initiated with electronical detonators.
- Unexpectedly short crack lengths were obtained from holes initiated with Nonel detonators.
- Shorter cracks and fewer cracks were created in the left hand wall than in the right wall.
- Most of the induced cracks in the vertical slots seem to have a diagonal direction (pointing upwards) in the lower part of the wall.
- Most of the induced and natural cracks seem to be in the direction 60–120 degrees relative the vertical walls.
- Fewer cracks were observed from the bench than from the tunnel (due to stress, hole straightness, water and perhaps confinement).
- No cracks originating from blast holes in the floor.
- No evidence of a continuous damage zone parallel to the tunnel wall was found.

Many of the examined contour holes in the wall pointed downwards. Consequently there may be reason to believe that the holes may have been water filled causing longer cracks. Contradictory, it is concluded that there were no visible cracks beneath in the downward dipping floor holes.

8 Conclutions

8.1 Learning curve

How long time will it take for a tunnel crew to reach full capacity, both in quantity and quality? This is of course a complicated question due to all different factors involved. On one hand, when a well synchronised crew is sent from one project to another together with the same pieces of equipment used in the previous project the time for reaching a good capacity is strongly reduced compared to the start up of a project where personnel and equipment is pooled together from different projects and conditions. The latter is the most likely to happen. The job opportunities today for underground personnel are not so much related to a construction company but more to a project.

The underground crew of today may be jointed by persons originating from different companies sometimes with different working procedures and organisation, which will increase the time required for reaching full advance. The type of project will also influence the start-up efficiency and implementation of quality plans. A technically more complicated project will be more dependent on the special skills of key personnel than a more straight-on project. If a key person is missing in the team it will show a reduced productivity.

The example shown in Figure 8-1 is from a large hydropower project in a fairly remote part of the world. It also fits quite well with highway tunnels in Hong Kong. A local work force was hired and educated during the start-up which may have slowed down the raise time of the curve but still, in this case the calculated 80% of a theoretical advance was reached while in other projects this level is never attained. This issue must be dealt with within the risk assessment work. Other projects when an entire organisation together with their equipment is moved from one tunnel project to another, facing the same geological scenario the start-up time would of course be shorter. One must also keep in mind that the start-up process is not only taking place at the tunnel face but also at workshops, at offices and along all logistic activities involved in a larger tunnelling project.

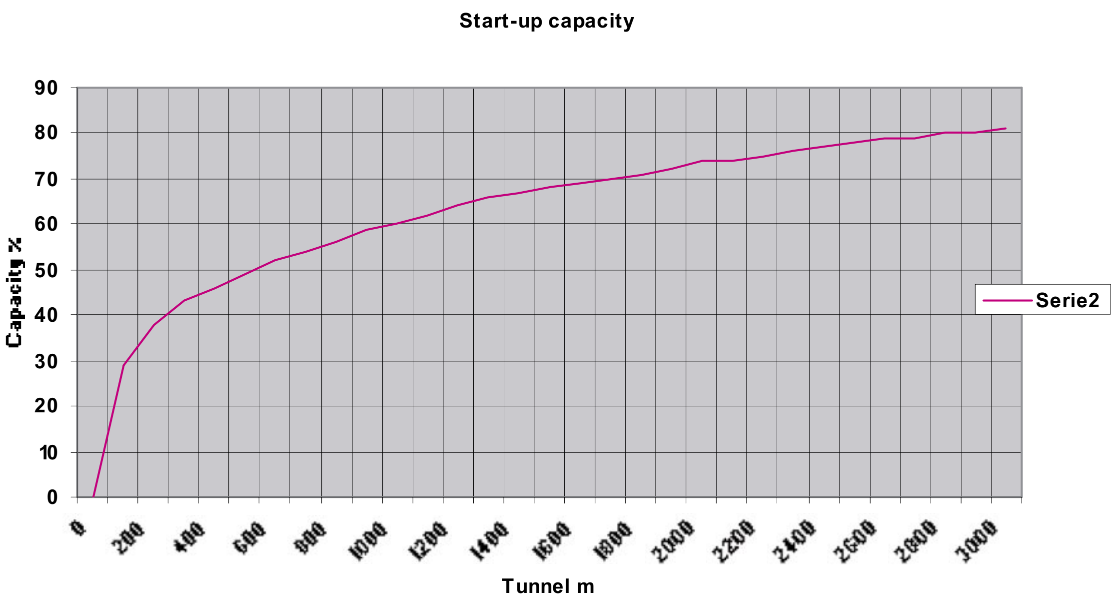


Figure 8-1. *A diagram of advancee per round versus tunnel length. 80% is considered to be practically attainable maximum capacity.*

Seen in the light of this diagram the 80 m long TASQ tunnel was not even a warm-up period for the tunnel crew especially since the TASQ tunnel contained different tunneling disciplines, portalling, top heading and benching. However, in the APSE project the difficulties where highly visible to the tunnel crew. The closeness to different types of installations, to the shaft area and the research activities made everyone aware of the importance of caution and that quality goes before speed. The start-up progress in this kind of project must be focused more on quality than on speed and consequently the diagram in Figure 8-1 is not fully applicable for the TASQ tunnel.

8.2 Best available technology

The answer to the question if the tunnel excavation could have been carried out in a better way is yes. The best available technology was not used in the project basically because it was not ordered. A normal technology would be the proper description of present the tunnelling works.

The contract documents define the tolerances of excavation as class 2 according to the Swedish Construction Directives AMA-98 where the final excavated geometry is compared to the theoretical excavated geometry. The directives identify 3 classes where class 2 could be considered as normal and class 1 as the highest tolerance class.

The exception from class 2 was the benching passing the test area. For this part the class 1 was ordered but with even higher demands. No hole in the contour line was permitted to end more than 0.30 m outside of the theoretical tunnel section and no hole deviation exceeding 10 mm/m was permitted. The drill hole deviation tolerance of 10 mm/m is not defined by AMA-98 but was added by SKB as well as the distance of 0.3 m outside the theoretical tunnel section where AMA-98 class 1 permits 0.7 m for a single hole. In order to regularly fulfil these two demands many specialised drilling arrangements would have been needed. Even today's best drilling equipment for tunnelling is a compromise. It is built to drill different types of holes, different hole sizes, it must be flexible, movable and not too expensive. The fact is that the tunnelling technology can almost always be improved but at the cost of an increasing price tag.

Best available technology in a future tunnel with similar requirements is proposed to include:

- A special training period before the excavation of the tunnel. To build a team, to build up skill in using new equipment and to inform all involved personnel about the project background and critical activities.

- New equipment with the latest technology would increase the accuracy in drilling but not up to the demands earlier mentioned. Possible measures in order to succeed could be a more stable feeder beam, the use of steel stabilisers and manual alignment of the drill steel. Such equipment is available and used for special long-hole drilling but not proper today for conventional tunnelling.

- A large variety of explosive types and dimensions may improve the blasting result but may increase the time for the charging procedure and raise the costs. In order to reduce damage especially in the bottom holes water must be prevented from filling the holes. To ensure a blast hole almost free of water, special pre-decoupled charges may have to be manufactured. These charges consist of closed pipes slightly smaller in diameter than the hole itself. Inside the pipe the charge is regularly centred and de-coupled. The pipe is placed in the hole and locked to prevent from slipping out of the hole due to the water pressure.

- Access to an electronic detonator system could improve tunnelling by:
  - A higher safety against simultaneously detonated charges due to more available delay numbers. The need for surface delay solutions in large tunnel sections with a large number of holes will decrease.
  - A better result from contour blasting, reduced damage zone and smoother tunnel surface, could be achieved if a simultaneous initiation of contour holes is made.
  - A reduction of ground vibrations at longer distances has been shown with the use of selected delay times using electronic detonators.

To find the best available technology when the focus is beamed on reduction of blast damage and high quality tunnel contours, the main effort should be to develop or modify the drilling equipment and to educate and encourage the organisation to perform as well as possible.

Modification of equipment must be made on a long-term relationship with the manufacturers.

Care of the organisation must be taken on different levels like:

- Very high awareness of the project goals.
- Continuous education and training.
- Allow free scope for ideas and R&D projects aiming at improving the tunnelling works.
- Keep the organisation intact, with a low turnover as a result.

8.3 EDZ

The excavation of the TASQ tunnel with the primarily focus on the APSE experiment brought in some requirements that are unusual for underground projects, at least in Sweden. The main focus was on the quality of the excavated floor. The rock mechanic conditions included rock stresses that were not extreme, but significantly elevated by the tunnel design, compared to other tunnels at the AHRL.

- To excavate with top heading and bench gives significantly lower damage in the floor compared to ZEDEX experiences /Olsson and Reidarman, 1995/, even less than in the roof and walls. This is primarily caused by the difference in specific charge in the contour holes for the floor (25 mm Emusion cartridges respectively 17 mm Gurit cartridges). Further development in blast design is needed to enable similar results in terms of a small EDZ in the floor without excavation of a separate bench.

- For the “average” tunnel construction, based on current Swedish practice the observed excavation damage is similar to that observed in the ZEDEX D&B tunnel 8 years ago.

- A systematic use of an electronic initiation system in the contour seems to be promising or a further reduction of the extent of the EDZ.

- Large drill hole deviations have caused significant local damage.

- The reasons for local significant larger extension of the EDZ are well understood. They are most likely manageable in a systematic QA program during excavation. To implement quality plans in the common organisation for a tunnelling project may require special care and requires reasonable acceptance times from the staff to get a fast rising learning curve.

- The look-out angle and the distribution of specific charge along each round causes a discontinuous EDZ along the tunnel. It is therefore concluded that the impact of the EDZ on hydraulic conductivity along the tunnel is very small, because it is manageable through D&B design and QA control during excavation.

- The stresses around the TASQ tunnel were sufficiently high to cause some shearing of natural fractures within the EDZ. This is an effect of the actual design, which aimed at elevating the stresses as high as possible, especially in the floor. This was done by orientation of the tunnel, tunnel height to span ratio and the circular floor. A more horse-shoe shaped with a smaller height to width ratio tunnel would significantly decrease induced stresses in the floor. It is however important to consider that an alignment of a tunnel at some angle to a dominant fracture set will reduce the risk for stress induced opening of natural fractures over longer stretches of the tunnel, especially in a relatively high stress environment.

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Appendix 1

Charging plan for TASQ tunnel

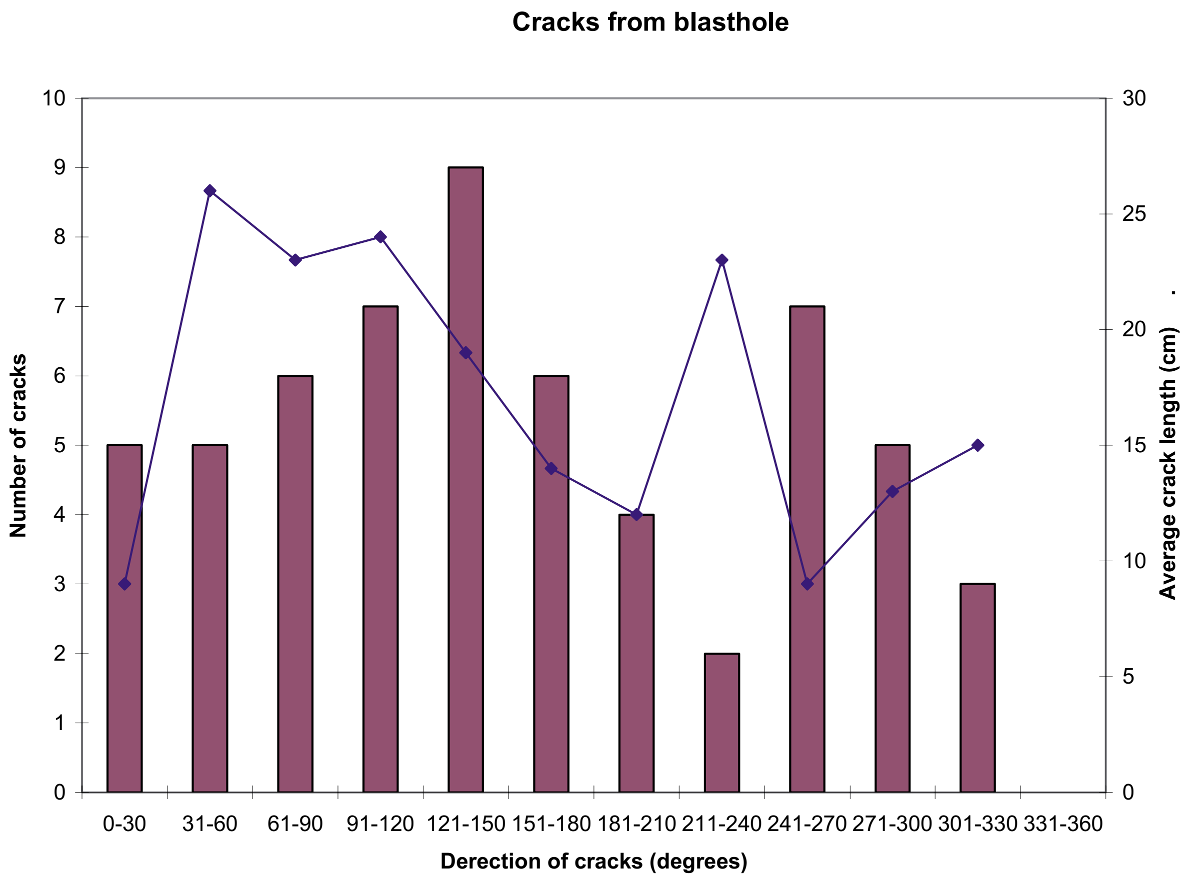
Round	Hole depth (m)	Type of hole	No of cartridges, explosives type (diameter in mm x length in mm) Bottom charge. A primer type charge, Noble prime, is used as bottom charge in some rounds.	Column charge	Total charge (kg/hole)
1	1.5	Cut	1 Nobelprime 15×150	1 Dynotex 22×1,000	0.4
2	1.5	Cut	1 Nobelprime 15×150	2 Dynotex 22×1,000	0.78
3	1.5	Cut	1 Dynomit 30×380	1 Dynorex 25×1,110	1.11
4	1.5	Stoping	1 Dynomit 30×380	1 Dynorex 25×1,110	
5	2.0	Cut		1.8 Dynorex 25×1,110	1.31
6	2.0	Stoping	0.25 Dynomit 30×380	1 Dynotex 22×1,000 + 0.5 Dynorex 25×1,110	0.82
7	2.0	Stoping	0.25 Dynomit 30×380	1.5 Dynorex 25×1,110	1.19
		Lifter	0.25 Dynomit 30×380	1.5 Dynorex 25×1,110	1.19
8	2.0	Stoping	0.5 Dynomit 30×380		0.19
		Stoping	0.75 Dynomit 30×380	1.5 Dynorex 25×1,110	1.38
		Stoping	1 Dynomit 30×380	1.5 Dynorex 25×1,110	1.48
		Stoping	0.75 Dynomit 30×380	2.5 Dynorex 25×1,110	2.11
9	3.0	Cut	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
10	3.0	Stoping	1 Dynomit 30×380	1.25 Dynorex 25×1,110	1.29
		Lifter	2 Dynorex 25×1,110		1.46
11	4.0	Stoping	1 Dynomit 30×380	1 Dynorex 25×1,110 + 2 Dynotex 22×1,000	1.83
		Helper	1 Dynomit 30×380	3 Dynotex 22×1,000	1.46
		Contour	0.25 Dynomit 30×380	8 Dynotex 17×460	0.86
		Lifter	Same as stoping holes		1.83
12 + 13	3.85	Cut	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
		Stoping	1 Dynomit 30×380	1 Dynorex 25×1,110 + 2 Dynotex 22×1,000	1.83
		Helper	1 Dynomit 30×380	3 Dynotex 22×1,000	1.46
		Contour	0.25 Dynomit 30×380	8 Dynotex 17×460	0.86
		Lifter	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
		Lifter	1 Dynomit 30×380	1 Dynorex 25×1,110 + 2 Dynotex 22×1,000	1.83
14	3.0	Cut	1 Dynomit 30×380	2.5 Dynorex 25×1,110	2.21
		Stoping	1 Dynomit 30×380	2 Dynorex 25×1,110	1.84
		Helper	1 Dynomit 30×380	2 Dynotex 22×1,000	1.10
		Contour	0.25 Dynomit 30×380	6 Dynotex 17×460	0.66
		Lifter	1 Dynomit 30×380	2 Dynorex 25×1,110	1.84
15	4.0	Cut	1 Dynomit 30×380	3.5 Dynorex 25×1,110	2.94
Misfire		Stoping	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
		Helper	1 Dynomit 30×380	3 Dynotex 22×1,000	1.46
		Contour	0.5 Dynomit 30×380	8 Dynotex 17×460	0.95
		Lifter	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
16	4.0	Cut	– – – – –		
Misfire		Stoping	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
		Helper	1 Dynomit 30×380	3 Dynotex 22×1,000	1.46
		Contour	0.5 Dynomit 30×380	8 Dynotex 17×460	0.95
		Lifter	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57

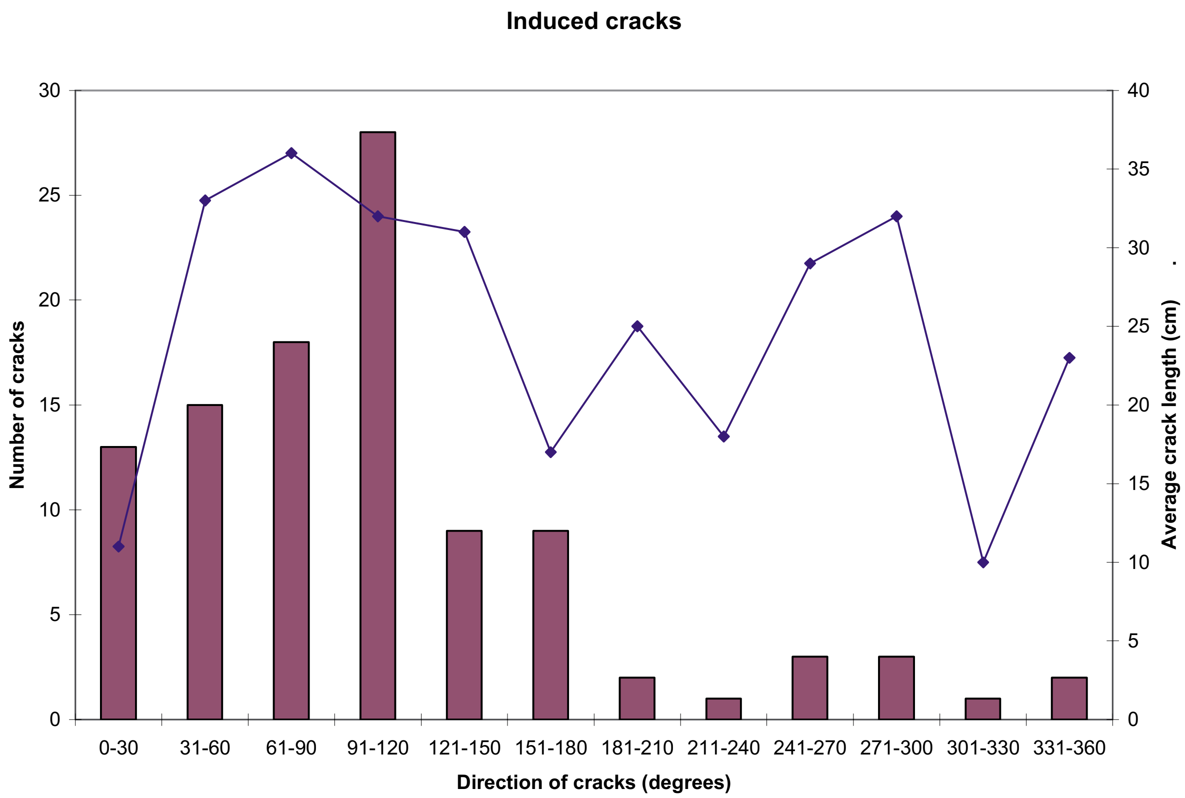
Round	Hole depth (m)	Type of hole	No of cartridges, explosives type (diameter in mm x length in mm)		Total charge (kg/hole)
			Bottom charge. A primer type charge, Noble prime, is used as bottom charge in some rounds.	**Column charge**	
17	4.0	Stoping	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
		Helper	1 Dynomit 30×380	3 Dynotex 22×1,000	1.46
		Contour	0.5 Dynomit 30×380	8 Dynotex 17×460	0.95
		Lifter	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
18	4.0	Cut	1 Dynomit 30×380	3.5 Dynorex 25×1,110	2.94
		Stoping	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
		Helper	1 Dynomit 30×380	1 Dynorex 25×1,110 + 3 Dynotex 22×1,000	2.19
		Contour	0.5 Dynomit 30×380	8 Dynotex 17×460	0.95
		Lifter	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
19	4.0	Cut	1 Dynomit 30×380	3.5 Dynorex 25×1,110	2.94
		Stoping	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
		Helper	1 Dynomit 30×380	1 Dynorex 25×1,110 + 3 Dynotex 22×1,000	2.19
		Contour	0.5 Dynomit 30×380	8 Dynotex 17×460	0.95
		Lifter	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
20	4.0	Cut	1 Dynomit 30×380	3.5 Dynorex 25×1,110	2.94
		Stoping	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
		Helper	1 Dynomit 30×380	1 Dynorex 25×1,110 + 3 Dynotex 22×1,000	2.19
		Contour	0.5 Dynomit 30×380	8 Dynotex 17×460	0.95
		Lifter	1 Dynomit 30×380	3 Dynorex 25×1110	2.57
21	4.5	Cut	1 Dynomit 30×380	4 Dynorex 25×1,110	3.68
Misfire		Stoping	1 Dynomit 30×380	3.5 Dynorex 25×1,110	2.94
		Helper	1 Dynomit 30×380	1 Dynorex 25×1,110 + 3 Dynotex 22×1,000	2.19
		Contour	0.5 Dynomit 30×380	9 Dynotex 17×460	1.05
		Lifter	0.25 Dynorex 25×1,110	4 Dynorex 22×1,110	1.62
22	4.5	Cut	– – – – –		
		Stoping	2 Dynomit 30×380	3.5 Dynorex 25×1,110	3.32
		Helper	1 Dynomit 30×380	1.5 Dynorex 25×1,110 + 3 Dynotex 22×1,000	2.56
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
		Lifter	0.25 Dynorex 25×1,110	4 Dynorex 22×1,110	1.62
23	45	Cut	2 Dynomit 30×380	3.5 Dynorex 25×1,110	3.32
		Stoping	2 Dynomit 30×380	3 Dynorex 25×1,110	2.95
		Helper	1 Dynomit 30×380	4 Dynotex 22×1,000	1.82
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
		Lifter	4 Dynorex 25×1110		2.92
24	4.0	Cut	2 Dynomit 30×380	3 Dynorex 25×1,110	2.95
		Stoping	2 Dynomit 30×380	2.5 Dynorex 25×1,110	2.59
		Helper	1 Dynomit 30×380	4 Dynotex 22×1,000	1.82
		Contour	2 Nobelprime 15×150	9 Dynotex 17×460	0.92
		Lifter	4 Dynorex 25×1,110		2.92
25	2.3	Cut	1 Dynomit 30×380	2 Dynorex 25×1,110	1.84
		Stoping	1 Dynomit 30×380	1.5 Dynorex 25×1,110	1.48
		Helper	1 Dynorex 25×1,110	1 Dynotex 22×1,000	1.09
		Contour	2 Nobelprime 15×150	5 Dynotex 17×460	0.54
		Lifter	2 Dynorex 25×1,110		1.46

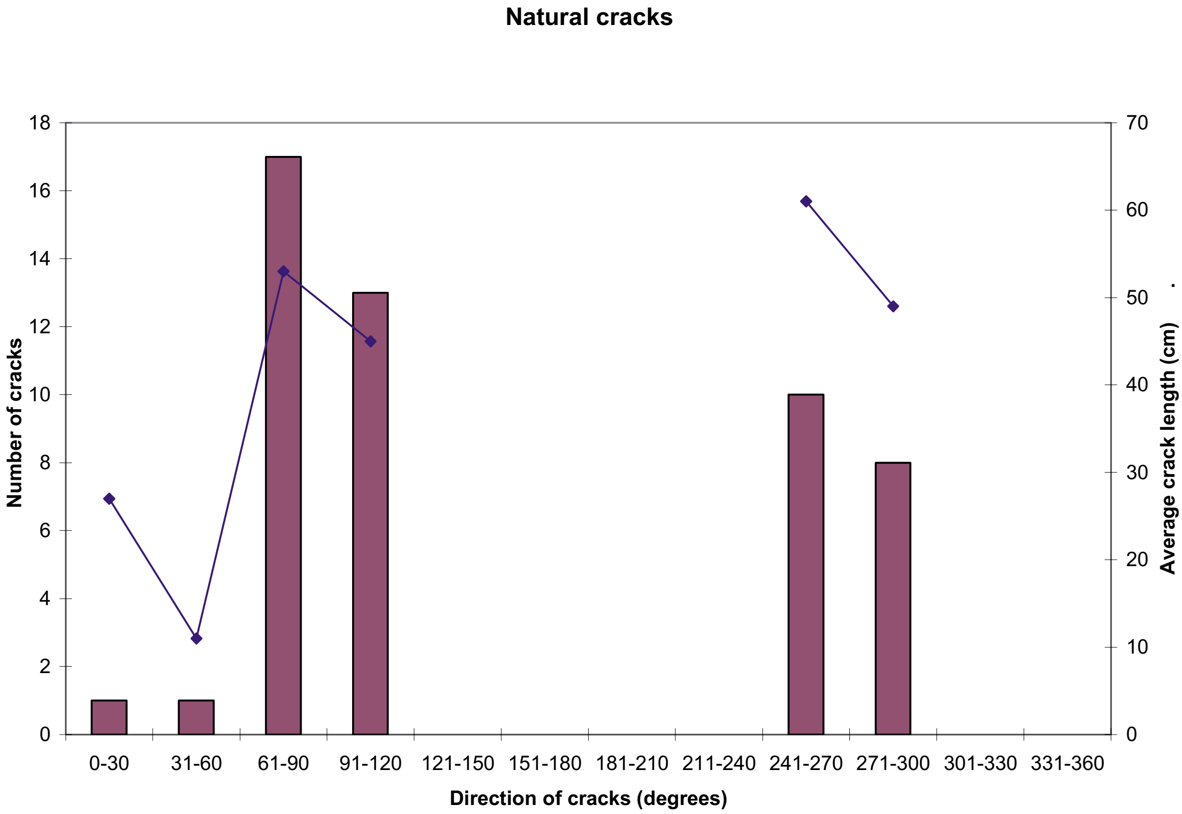
Round	Hole depth (m)	Type of hole	No of cartridges, explosives type (diameter in mm x length in mm)		Total charge (kg/hole)
			Bottom charge. A primer type charge, Noble prime, is used as bottom charge in some rounds.	**Column charge**	
26	2.3	Cut	1 Dynomit 30×380	2 Dynorex 25×1,110	1.84
		Stoping	1 Dynomit 30×380	1.5 Dynorex 25×1,110	1.48
		Helper	1 Dynorex 25×1,110	1 Dynotex 22×1,000	1.09
		Contour	2 Nobelprime 15×150	5 Dynotex 17×460	0.54
		Lifter	2 Dynorex 25×1,110		1.46
27	2.4	Cut	1 Dynomit 30×380	2 Dynorex 25×1,110	1.84
		Stoping	1 Dynomit 30×380	1.5 Dynorex 25×1,110	1.48
		Helper	1 Dynorex 25×1,110	1 Dynotex 22×1,000	1.09
		Contour	2 Nobelprime 15×150	5 Dynotex 17×460	0.54
		Lifter	2 Dynorex 25×1,110		1.46
28	4.6	Lifter	0.25 Dynorex 25×1,110	4 Dynorex 22×1,110	1.62
29	4.1	Cut	1 Dynomit 30×380	3.5 Dynorex 25×1,110	2.94
		Stoping	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
		Helper	1 Dynomit 30×380	1 Dynorex 25×1,110 + 3 Dynotex 22×1,000	2.19
		Contour	0.5 Dynomit 30×380	8 Dynotex 17×460	0.95
		Lifter	1 Dynomit 30×380	3 Dynorex 25×1,110	2.57
30	4.5	Cut	2 Dynomit 30×380	1.5 Dynomit 29×1,110 + 2 Dynorex 25×1,110	3.63
		Stoping	1.5 Dynomit 29×1,100	2 Dynorex 25×1,110	2.87
		Helper	1 Dynomit 29×380	4 Dynotex 22×1,000	1.76
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
		Lifter	4 Dynorex 25×1,110		2.92
31	4.5	Cut	2 Dynomit 30×380	1.5 Dynomit 29×1,110 + 2 Dynorex 25×1,110	3.63
		Stoping	2 Dynomit 30×380	3 Dynorex 25×1,110	2.95
		Helper	1 Dynomit 29×380	4 Dynotex 22×1,000	1.76
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
		Lifter	4 Dynorex 25×1,110		2.92
32	4.5	Cut	4 Dynomit 29×380	3 Dynomit 29×1,110	4.10
		Stoping	3 Dynomit 29×380	3 Dynorex 25×1,110	3.15
		Stoping	1 Dynomit 29×380	3.5 Dynorex 25×1,110	2.88
		Stoping	2 Dynomit 29×380	3 Dynorex 25×1,110	2.83
		Stoping	5 Dynomit 29×380	4 Dynotex 22×1,000	3.04
		Stoping	10 Dynomit 29×380		3.20
		Helper	1 Dynomit 29×380	4 Dynotex 22×1,000	1.76
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
		Lifter	4 Dynotex 22×1,000		1.44
33	4.5	Cut	4 Dynomit 30×380	3 Dynomit 29×1,110	4.34
		Stoping	3 Dynomit 29×380	2 Dynomit 29×1,110 + 1 Dynorex 25×1,110	3.75
		Stoping	2 Dynomit 29×380	1 Dynomit 29×1,110 + 1.5 Dynorex 25×1,110	2.68
		Helper	1 Dynomit 29×380	4 Dynotex 22×1,000	1.76
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
		Lifter	4 Dynotex 22×1,000		1.44

Round	Hole depth (m)	Type of hole	No of cartridges, explosives type (diameter in mm x length in mm)		Total charge (kg/hole)
			Bottom charge. A primer type charge, Noble prime, is used as bottom charge in some rounds.	Column charge	
34	4.0	Cut	4 Dynomit 30×380	3 Dynomit 29×1,110	4.34
		Stoping	3 Dynomit 29×380	2 Dynomit 29×1,110 + 1 Dynorex 25×1,110	3.75
		Stoping	2 Dynomit 29×380	1 Dynomit 29×1,110 + 1.5 Dynorex 25×1,110	2.68
		Helper	1 Dynomit 29×380	4 Dynotex 22×1,000	1.76
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
		Lifter	4 Dynotex 22×1,000		1.44
35	4.5	Helper	1 Nobelprime 15×150	4 Dynotex 22×1,000	1.47
Bench		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
36	4.4	Stoping	0.5 Dynorex 25×1,110	4 Dynotex 22×1,000	1.81
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 Nobelprime 15×150		1.01
			10 Dynotex 17×460		
37	4.4	Stoping	0.5 Dynorex 25×1,110	4 Dynotex 22×1,000	1.81
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 Nobelprime 15×150		1.01
			10 Dynotex 17×460		
38	4.4	Stoping	0.5 Dynorex 25×1,110	4 Dynotex 22×1,000	1.81
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 Nobelprime 15×150		1.01
			10 Dynotex 17×460		
39	3.7	Stoping	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 Nobelprime 15×150	8 Dynotex 17×460	0.82
40	4.6	Stoping	0.5 Dynorex 25×1,110	4 Dynotex 22×1,000	1.81
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
41	4.5	Stoping	0.5 Dynorex 25×1,110	4 Dynotex 22×1,000	1.81
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 nobelprime 15×150	10 Dynotex 17×460	1.01
42	4.5	Stoping	0.5 Dynorex 25×1,110	4 Dynotex 22×1,000	1.81
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
43	4.6	Stoping	0.5 Dynorex 25×1,110	4 Dynotex 22×1,000	1.81
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
44	4.4	Stoping	0.5 Dynorex 25×1,110	4 Dynotex 22×1,000	1.81
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 Nobelprime 15×150	10 Dynotex 17×460	1.01
45	3.9	Stoping	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
Bench		Helper	0.25 Dynorex 25×1,110	4 Dynotex 22×1,000	1.62
		Contour	2 Nobelprime 15×150	8 Dynotex 17×460	0.82
46	2.0	Contour	2 Nobelprime 15×150	4 Dynotex 17×460	0.44
Bench					

Appendix 2


Cracks from blasthole
Number of cracks
Average crack length (cm)
0
1
2
3
4
5
6
7
8
9
10
0
5
10
15
20
25
30
0-30
31-60
61-90
91-120
121-150
151-180
181-210
211-240
241-270
271-300
301-330
331-360
Derection of cracks (degrees)


Induced cracks
Number of cracks
Average crack length (cm)
0
5
10
15
20
25
30
0
5
10
15
20
25
30
35
40
0-30
31-60
61-90
91-120
121-150
151-180
181-210
211-240
241-270
271-300
301-330
331-360
Direction of cracks (degrees)



Natural cracks
Number of cracks
Average crack length (cm)
Direction of cracks (degrees)
0
2
4
6
8
10
12
14
16
18
0
10
20
30
40
50
60
70
0-30
31-60
61-90
91-120
121-150
151-180
181-210
211-240
241-270
271-300
301-330
331-360

Appendix 3

Section	Tunnel	Bench	Charge	Initiation	Timing (ms)	Half-pipe (no)	Pipe	Type of cracks	Number of cracks (no)	Max. crack length (cm)	Average length (cm)	Scatter (max/min) (cm)	Notes
1	Yes		Column	Electronic	4,510	3		Blast	7	21	14	0/21	
			Column/bottom	Electronic	4,510		Yes	Blast	12	42	20	11/42	No breakage/ Water in hole
			Bottom	Electronic	4,510		Yes	Blast	12	14	7	3/14	No breakage
								Induced	2	30	24	17/30	
								Natural	7	93	70	37/91	
2	Yes		Column	Electronic	7,000	2		Blast	5	20	14	8/20	Water in hole?
			Column	Electronic	7,000	3		Blast	5	22	11	3/22	Water in hole?
								Induced	15	72	36	12/72	
								Natural	22	58	37	11/58	
3	Yes		Column	Electronic	7,000	3		Blast	6	20	11	9/20	Water in hole?
		Yes	Column	Nonel	3,500	1		Blast	3	10	5	0/10	
								Induced	47	54	20	6/54	
								Natural	7	97	44	11/97	
4	Yes		Column	Electronic	7,000	1		Blast	0	0	0		
					7,001	1		Blast	3	64	48	26/64	Water in hole?
					8,500	1		Blast	2	56	24	0/56	Water in hole?
								Induced	9	35	25	19/35	
								Natural	2	129	93	57/129	
Transition	Yes		Column	Nonel	4,500	1		Blast	0	0	0		
zone		Yes	Column	Nonel	3,000	1		Blast	4	31	16	5/31	Water in hole?
								Induced	29	66	38	7/66	
								Natural	9	114	63	17/114	
Horizontal	Yes		Column					Blast	0	0	0		
section								Induced	12	57	18	3/57	
								Natural					
Floor		Yes	Column	Nonel	5–8	1		Blast	0	0	0		
								Natural/Ind	17	63	29	2/63	