R-14-08

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March 2014

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ISSN 1402-3091
SKB R-14-08
ID 1433311

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Foreword

This project was conducted in collaboration between Diamond Wire Teknikk AS (DWT), NCC Construction Sweden AB and SKB. Financial support has also been given to the DWT from Innovasjon Norway through the development project "Tårn til blindkutt, rette gulv og tak."

The work was performed 2011–2012 within the excavation work for section "Norrström" in the City Line railway project, Stockholm, where DWT has been a subcontractor to NCC, which has been an important partner in the excavation and was responsible for all practical arrangements on this project as part of their undertaking.

The project would not have been possible to perform without the consent of the Client, Trafikverket, and the contractors' ability to coordinate this project with their regular duties.

Acknowledge to all actors for the opportunity given to follow up this interesting technology.

The report was prepared in collaboration between the authors and Roger Sjöland DWT, Göran Magnell NCC, Göran Ajling NCC and Stefan Sidander NCC.

Stockholm, December 2013

Rolf Christiansson, SKB

This report is a translation of the SKB report R-13-06 "Studie av vajersågning för deponeringstunnlar".

Contents

1 Introduction

The purpose of the project was to demonstrate the technology to saw a tunnel with dimensions of a deposition tunnel. The excavation work covered a 16 m long rectangular tunnel where the contour is shaped by wire sawing. The subsequent excavation of free-sawn rock was performed with conventional drilling and blasting. The purpose of this report is to describe the work done, and to compare techniques and costs for rock excavation for the deposition tunnels made with conventional drilling and blasting versus wire sawing of all or part of the contour.

The project concerns the development of technologies and equipment for rock excavation of tunnels with diamond wire sawing of floor, walls and roof from the face and into the rock. The execution requires both equipment for drilling long straight holes in the rock and equipment for wire sawing rock from the tunnel face and inward in the rock. If a rectangular tunnel is to be sawn, the work will begin with drilling a hole in each corner. In two of these holes, long push rods are put in with pulleys at the ends for the diamond wire and the diamond wire is arranged as a loop that can rotate and be kept under constant tension as the wire cuts into the rock. By repeating the installation of the push rods with pulleys in the drilled holes in the corners of the rectangle it is possible to saw the tunnel floor, sides and roof from outside and in.

The project was carried out during 2011/2012 and was a development project with the participation of NCC Sweden, Diamond Wire Teknikk AS (DWT) of Norway, Innovasjon Norway and SKB Sweden.

The project was conducted in NCC's undertaking Norrström for the City Line in Stockholm. DWT has performed extensive wire sawing for escalators, ventilation shafts etc, in the City Line. DWT has also, on behalf of SKB, for a number of years conducted feasibility studies on the technical and economic advantages for the wire sawing compared to conventional drill and blast of deposition tunnels.

In 2010 DWT took the initiative to implement this development project being funded by the participating parties. The budget for the project is approximately 5 million kronor of which SKB's part is paid by in-kind contribution in the project. Innovasjon Norway has contributed to the funding of the project as a development support to DWT. A condition from Innovation Norway to participate in the financing was that the sawn tunnel could not be used for any commercial purpose but had to be destroyed after completing the project. As the chosen location for the demonstration was one of track tunnels in the City Line, which was then enlarged up to the final size, it was easy to satisfy this condition.

2 Prerequisites

2.1 The test location

NCC's undertaking for the City Line includes parallel track tunnels for trains and it was decided that a section in one of these track tunnels could be used for demonstration of wire sawing of a tunnel corresponding to SKB's planned deposition tunnels in the future Spent Fuel Repository.

In conventional drilling and blasting of the deposition tunnels, these will have a horseshoe shape with flat floor and arched roof. This tunnel shape is difficult to achieve by wire sawing and therefore it is easiest to select a rectangular cross section for this test deposition tunnel. For the demonstration project with wire sawing, the idea was to complete a deposition tunnel with sawn floor, sides and roof but with a limited length of approximately 30 m. The sawing would be conducted in two phases. The motive for sawing in two phases was to demonstrate how a transition between two phases can be performed.

Unfortunately it turned out that the geology of the originally selected position in track tunnel 1 was inappropriate for the project because of fractured rock mass and consequently a new location had to be chosen. The new location meant that the length of the sawn tunnel was limited to about 16 m, but drilling and sawing was carried out despite this in two phases. The existing experience about how geology affects the conditions for wire sawing is in general terms:

- the impact of fractures impact on straightness and direction of the bore and the need for rock, reinforcement,
- rock composition, mainly quartz content, affecting the drilling and sawing efficiency,
- rock stress situation can be a potential risk that the wire gets stuck.

Figure 2-1 shows a plan view of the tunnel system as part of the City Line. The sawn tunnel was carried out as a pilot tunnel in track tunnel 1 between the sections 34+405 and 34+420 which is one of the future train tunnels in the City Line near the Central Station in Stockholm. The test site was chosen based on the availability with respect to other rock work and geological conditions.

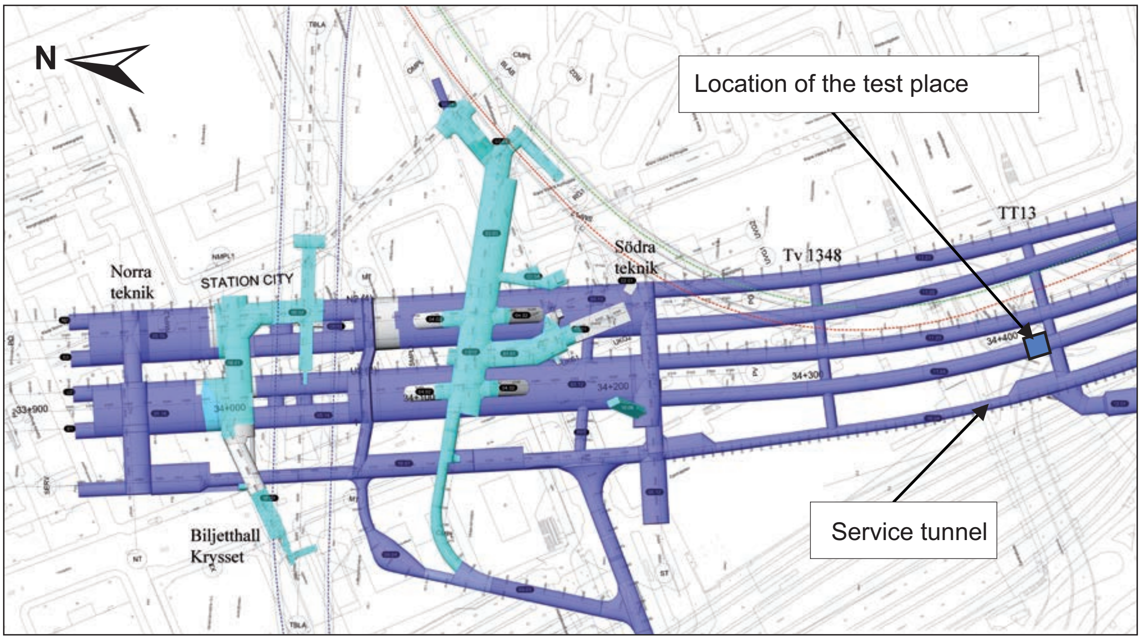


Figure 2-1. Plan view of the subway system in the City line adjacent to the Central Station in Stockholm. The site for the sawing project is marked. Rock coverage is approximately 30 m.

2.2 Geometric conditions

In a real situation in the Spent Fuel Repository deposition tunnels can have a length of up to 300 m. Previous drilling experience indicated that it can be difficult to achieve the required precision in the drilling if the whole deposition length of the tunnel would be drilled in one phase. Drilling and wire sawing is therefore expected to be carried out in phases with a provisional length of about 50 m to meet the tolerance requirements for drilling. This provisional length can probably be increased in the future. To document the geometry obtained at the transition between two sawing phases also this sawing project was made in two phases, Figure 2-2.

The sawn tunnel size was equal to SKB's planned deposition tunnels, width 4.2 m and height 4.8 m. The planned tunnel geometry and phase division are shown in Figure 2-3.

The preliminary demands on tunnel geometry are that the over break should not be more than 30 cm. Holes for the sawing must be carried out with a look-out angle to provide space for the drilling equipment when the next drilling phase begins. Therefore a split of the execution was decided into two phases so that the experience from drilling inside the sawn tunnel geometry could be obtained. The division into phases was done to estimate the size of the step between the phases. The length of phase 1 was about 10.5 m and of phase 2 about 5.5 m. After completed sawing and excavation it was thought that also the step in the floor should be sawn to facilitate vehicle transports in the tunnel, etc. However, this was not performed because of lack of time, but it can easily be done.

The tunnel floor in phase 1 was skewed in order to test the precision of the execution. The idea is that by using of bevelled floor, water can easily be drained away from the flat floor without further action.

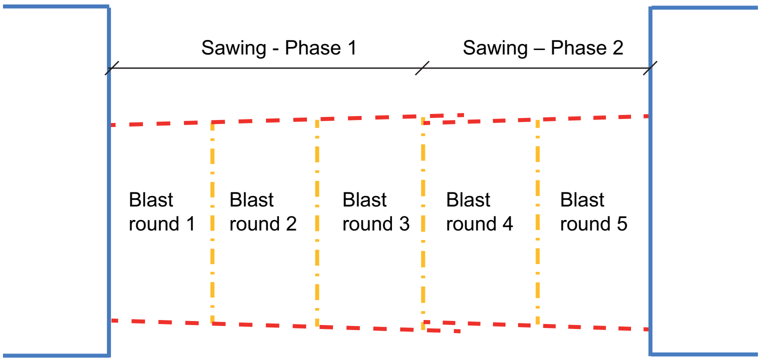


Figure 2-2. *Illustration of the tunnel for the demonstration of wire sawing in the big train tunnel. Blasting of the free-sawn rock was made in three plus two blasting rounds.*

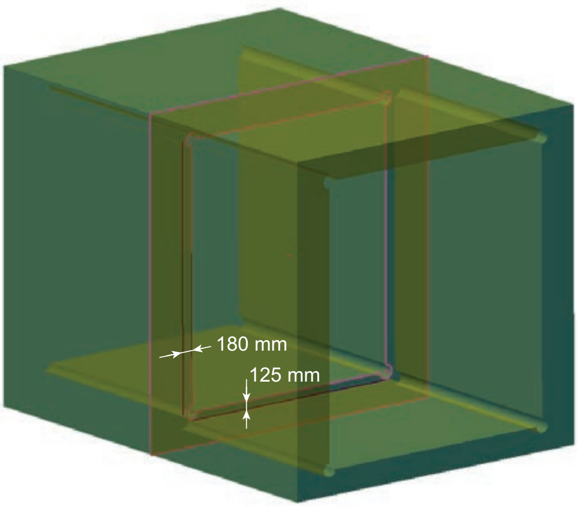


Figure 2-3. *3D illustration of the deposition tunnel as planned for the sawing project. The shown measures refer to the difference of level between phase 1 and 2.*

2.3 Rock mechanical conditions

The selected location has approximately 30 m of rock coverage and in the rock mechanical analysis the sawn tunnel would have sufficient stability without additional rock reinforcement but rock reinforcement could be made if deemed necessary.

Pregrouting of the experimental area was completed before the start of sawing project, which reduced the risk for problems with water inflow to the excavation of the deposition tunnel.

The geology of the area was mapped using cores from the drill holes for wire sawing. These cores were mapped according to the procedures that NCC uses for the works in the City Line project. Core mapping was only performed on cores from the upper left corner and the lower right corner. Compilation of the results of the mapping of the two cores is shown in Figure 2-4.

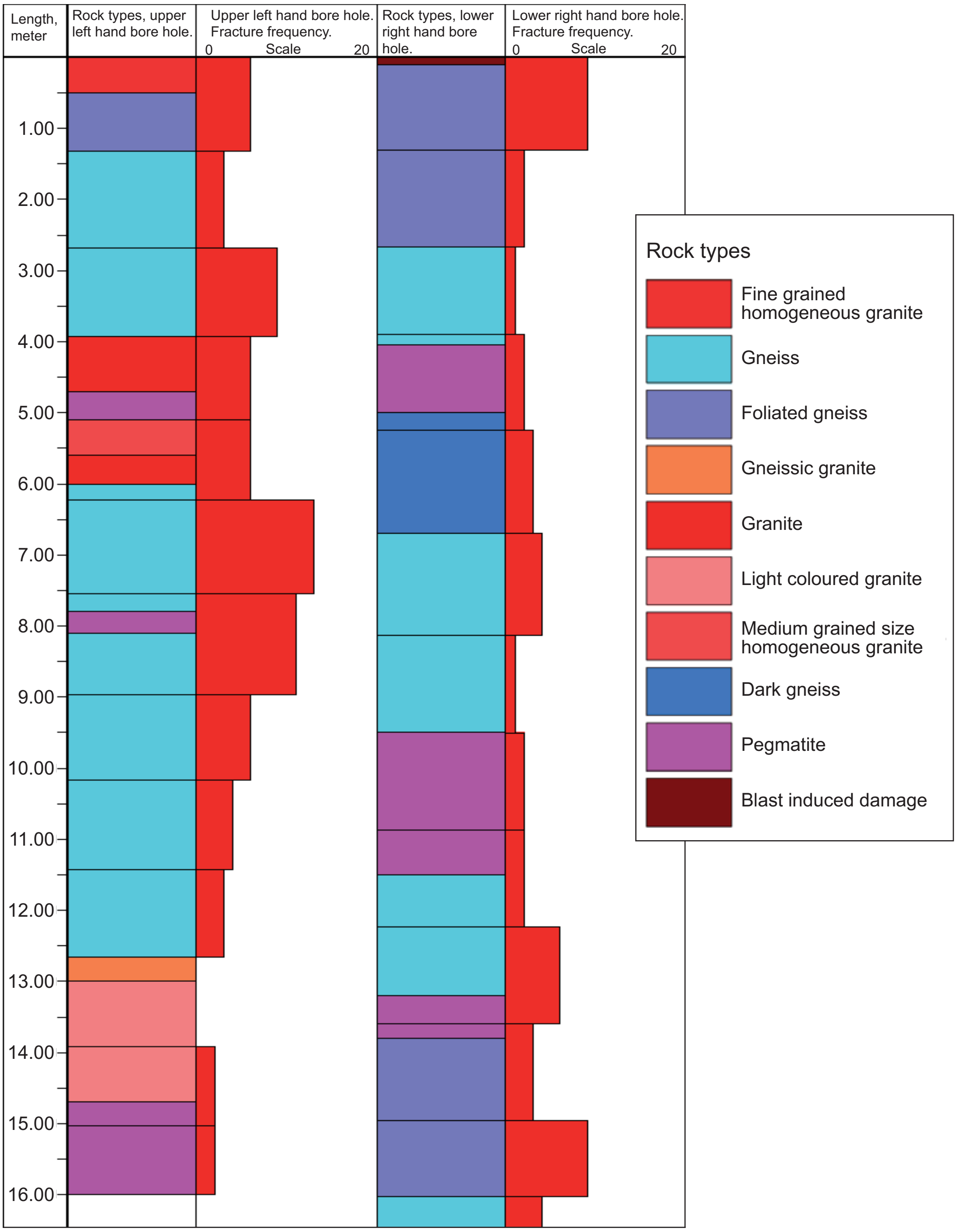


__Figure 2-4.__ Geological mapping of two of the 16 m long boreholes. The column on the left shows the upper left borehole and the right column shows the lower right borehole.

The test area consists of gneiss and foliated gneiss, which are mostly medium-grained. More granitic patches and pegmatite dykes occur as well. The rock is fresh without alterations. Foliation is quite steeply dipping with a east-west strike and is most intense at the end of the tunnel. According to the core mapping the frequency of natural fractures varies from 0 to 9 pieces per core uptake and the RQD value (Rock Quality Designation) is between 53 and 100. There are occasional clay-filled fractures with a fracture aperature of about 10 mm.

Figure 2-5 shows the appearance of some of the cores from the upper left and lower right borehole. The upper left borehole showed a poorer rock quality than the rock in the lower right hole.

The geological mappings that NCC had completed in the adjacent tunnels were also on hand and from those an assessment was made of rock quality to RMR 70–100 (Rock Mass Rating).

To assess the need for any rock reinforcement of the flat roof, a 2D elastic analysis was made for the actual dimension with the following assumptions:

- Depth 30 m = > σ_v = 0.8 MPa.
- σ_H = 2 MPa.
- E = 45 GPa.
- Influence from nearby excavated tunnels was judged to be small.

The results of the calculations are shown in Figure 2-6.

Based on the results of the calculations the following assessments were made:

- It is known from other tunnel projects in the Stockholm area that the bedrock has an excess of horizontal in situ rock stresses. Assuming slight excess of the horizontal stress, even in this relatively shallow tunnel location, a weak arch-forming stress is expected in the tunnel roof.
- The walls, however, will be stress released.
- The fracture distribution in the rock will have great significance for reinforcement needs. The blocky rock mass should be considered because the wall is expected to be subject to stress relaxation that may cause need for rock reinforcement.

Figure 2-5. *Photo of boxes of drill cores from the upper left and lower right borehole of Figure 2-4.*

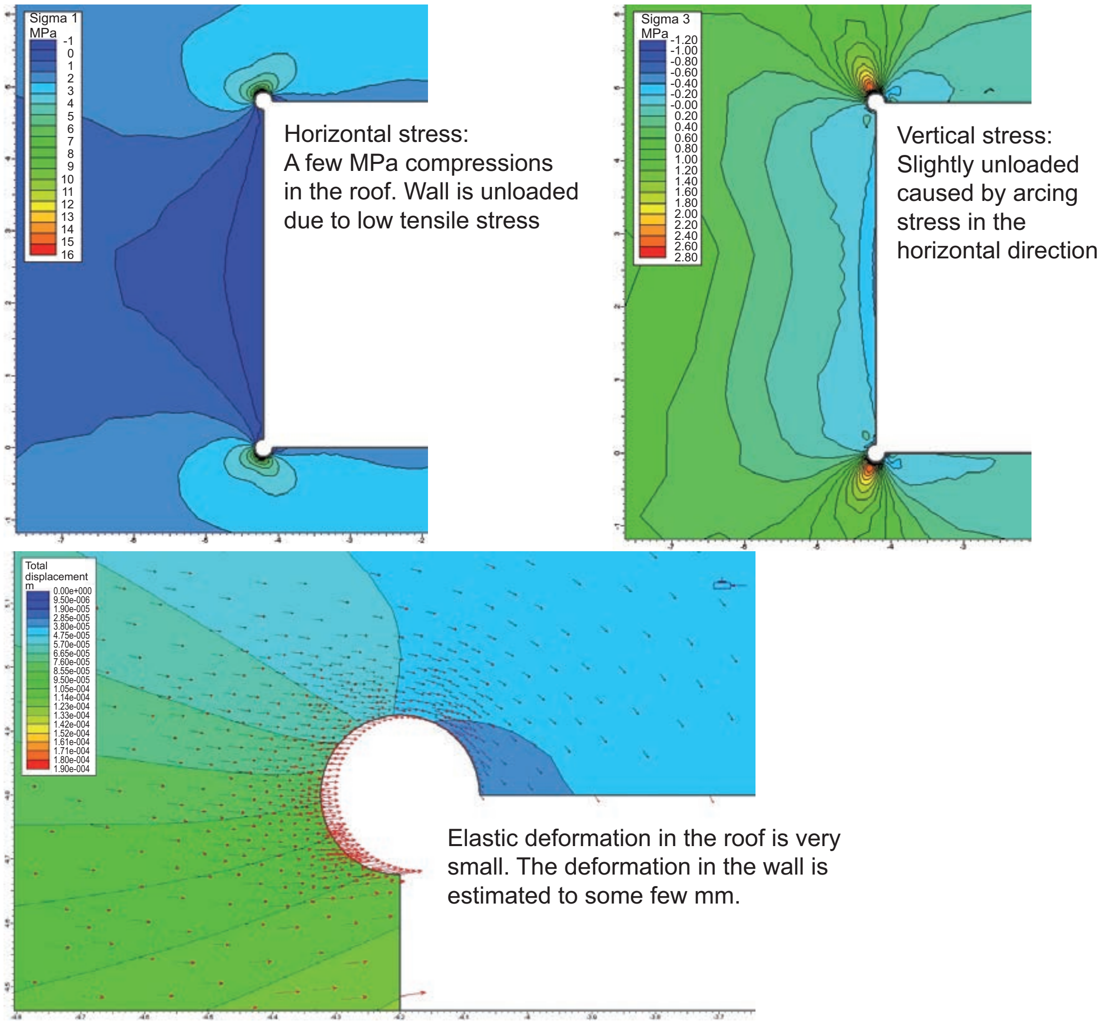


***Figure 2-6.** The upper pictures show the calculated horizontal and vertical stresses. The bottom image shows detail of deformations around the left upper borehole.*

3 Planning

3.1 Drilling and reaming of holes for wire sawing

To perform wire sawing from the front and inwards there must be boreholes into the rock that has sufficient diameter for the push rods with pulleys for the diamond wire to be installed.

Drilling of holes in the four corners of the tunnel is initiated by core drilling. The cores were placed in boxes and mapped. A reaming is made in two step to a final diameter of 255 mm. This diameter is required for inserting the push rods with pulleys for the diamond wire. Figure 3-1 is a photo of the tunnel face of the track tunnel where the four holes of the rectangular deposition tunnel are drilled and reamed.

The drilling of the holes in the four corners of the future tunnel was performed in three steps as follows:

Step 1 – The first drilling is performed as a core drill hole and form pilot holes for the subsequent reaming. To meet the requirements of accuracy for this drilling, a core drill machine Sandvik Onram 1000 was used with a drill diameter Ø 76 mm, see Figure 3-2.

The core drilling machine is modified to ensure an accuracy deviation < 5 cm over a length of drill of up to 60 m. The drilling of the pilot hole with a core drill rig gives the desired precision and it also gives cores to map the rock. The four boreholes in the tunnel corners give very good information about the rock properties and the occurrence of fractures.

Step 2 – Once the pilot hole has been drilled, the equipment is replaced with a down-hole precuasion drill. Through this drilling the hole diameter is increased the from 76 mm to 165 mm. The down-hole drilling was carried out with Geawelltech machine specially adapted and modified so that the boom can be bolted to the carrier of the core drilling equipment. This solution implies that one set-up and scaling is enough for the core borehole and when reaming is done with down-hole drilling after exchanging the boom.

Step 3 – Changing to bigger down-hole hammer and drill bit to ream the hole diameter to 255 mm.

Figure 3-3 shows the reamed hole and the drill bit used for the down-hole drilling.

This project required a refurbishment of the drilling equipment from a left-hand to a right-hand machine to enable the area change between two drill rounds to be as small as possible. In a production situation, a right-hand and a left-hand engine will be used to avoid this modification.

Figure 3-1. *Photo of the front of the track tunnel with the four holes in the corners of the rectangular deposition tunnel drilled.*

Figure 3-2. Drilling machine, Onram 1000, was used for drilling the core holes.

Figure 3-3. Photo of the reamed hole and drill bit used.

3.2 Wire sawing of floor, walls and roof

After drilling of the four holes for phase 1 the sawing into the rock was carried out as "blind cut" in four steps:

1. Sawing of the floor,
2. sawing of one wall,
3. sawing of the other wall, and finally
4. sawing off the roof.

The installation of the equipment was done by installing the push rods with the pulleys for the diamond wire in the left and the right bottom holes. At the same time, a diamond wire was drawn through the pulleys and was joined to loop over the drive unit.

As the wire penetrates the rock the drive unit stretches the wire by moving away from the face along a rail. This continues until the sawing reached the pulleys in the bottom of the drilled holes.

The next step was to install a push rod in the upper left corner and the push rod with pulley for the bottom hole was rotated 90 degrees. The installation of wires and cutting of the rock was made in the same manner as in the sawing of the floor.

Figure 3-4 is a 3D-model showing the design of the equipment used for sawing the floor, walls and roof. The illustration shows the arrangements for the sawing of the floor. When sawing the walls and roof, extra pulleys will be needed for the diamond wire.

Figure 3-5 shows photographs of the machine for stretching and driving the diamond wire and a close-up picture of the left bottom corner hole with sections from sawing with the wire.

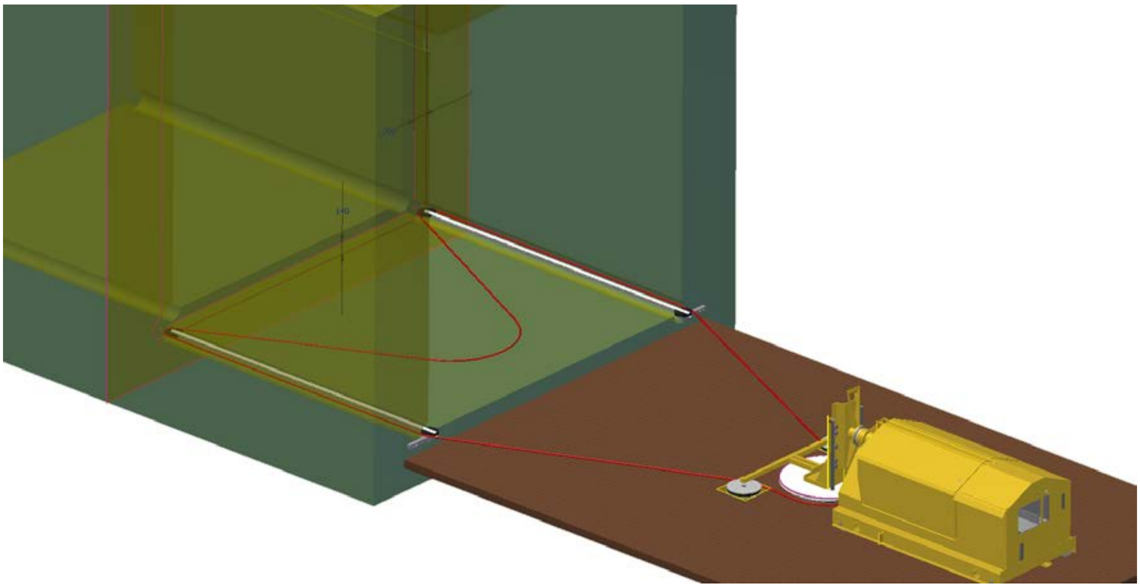


Figure 3-4. *3D model of a sawn tunnel with two drilling and sawing phases and the equipment with pulleys for the wire mounted on fixed push rods used for sawing of the floor, walls and roof.*

Figure 3-5. *Photo of the equipment for driving the diamond wire and to the right there is a close-up picture of the lower corner hole after the sawing has been made.*

3.3 Excavation of the free-sawn rock

When the sawing of phase 1 was completed, blasting was performed of the free-sawn rock in three blast rounds. Drilling and blasting plans were developed for about 3.5 m long rounds and took into account that the floor, sides and roof of the tunnel were free-sawn which affects the drill and the blast plan.

After that the rock in phase 1 was removed, drilling was carried out for the wire sawing of the remaining part of the tunnel. After completed sawing blasting is done of the remaining portion of the tunnel with two blast rounds.

After sawing the rock inside the sawn contour has to be blasted away. To make the task easy it was determined to try adjusting the drilling and charging to NCC's regular drill and charge plans. NCC operates pump emulsion as the main explosive in the City Line project and therefore it was decided to use this explosive also in the "Wire Sawing project". It was also decided to use Nonel LP detonators. Table 3-1 shows some typical burdens, hole spacing and charging configurations. All holes in this plan have a dimension of 48 mm and 0.2 kg DynoRex 32 mm was used as bottom charge together with emulsion. The round length for NCC's regular tunnel blast was 3.5 m and the tunnel area 80 m^2.

Using these data the drilling and charge plan was constructed as in Figure 3-6 and the data are shown in Table 3-2. The distance from the sawed contour to the drilling plan contour was 0.5 m. As the tunnel contour was wire sawn the blast round contour holes did not need to account for a look-out angle.

The drill plan includes a total of 62 blast holes in accordance with Table 3-2 of diameter 48 mm and 4 opening holes with a diameter of 102 mm. The length of blast rounds were 3.5 m.

The specific drilling and charging is shown in Table 3-3. There is apparently a big difference in the values depending on the area concerned, area limited of drilled contour or area limited of the sawed contour.

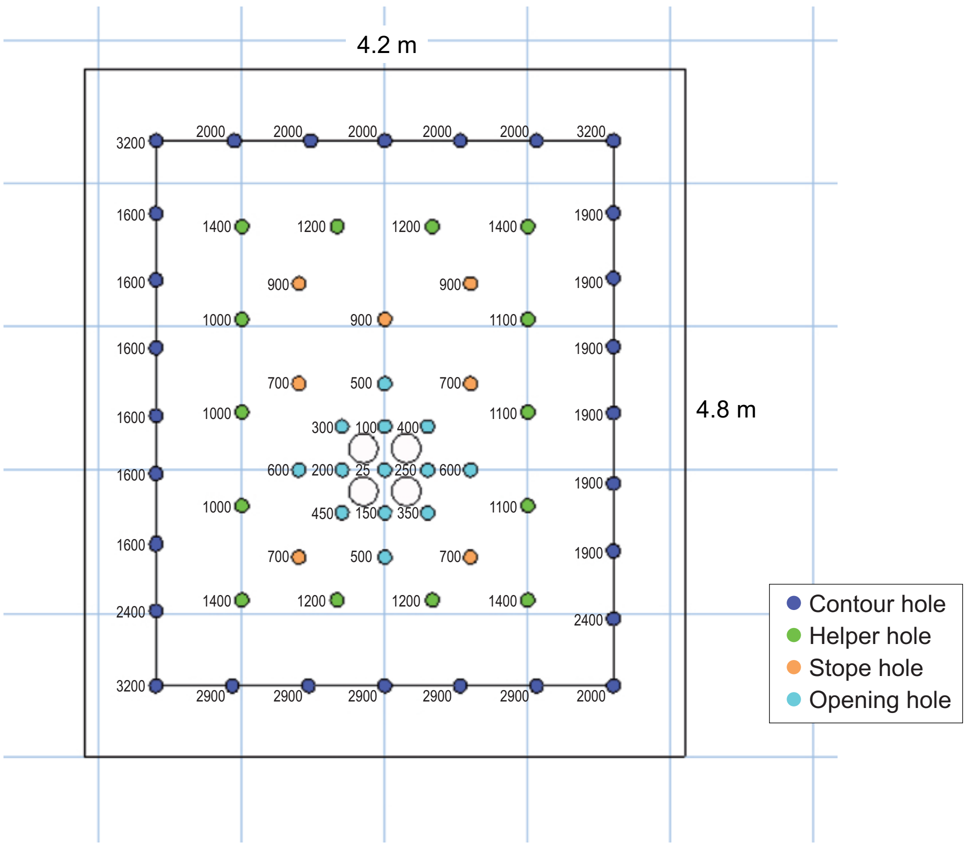


***Figure 3-6.** Drill and blast plan for phase 1. The delay between detonations specified in milliseconds.*

Table 3-1. NCC drill and blast data for a tunnel area of 80 m².

Hole type	Hole spacing (m)	Burdens (m)	Explosive	Specific charge (kg/m)
Contour	0.6	0.6	SSE	0.4
Helper	0.8	0.9	SSE	0.8
Stope hole	1.0	1.2	SSE	1.6

Table 3-2. Drilling and blasting data for the wire tunnel.

Hole type	No of holes	Hole spacing (m)	Burdens (m)	Explosive	Charge (kg/m)
Contour	28	0.4	0.6	SSE	0.35
Helper	14	0.65	0.4	SSE	0.8
Stope hole	7			SSE	0.8
Opening	13			SSE	1.6

Table 3-3. Specific drilling and charging track tunnel, phase 1.

Alternative	Area	Specific drilling	Specific charge
Inside drilled contour	12.16	5.1	3.8
Inside sawn contour	20.16	3.1	2.3

3.4 Planning of vibration measurements

To investigate if the vibration levels actually become lower when blasting against a sawn slit it was determined to measure vibrations at several points along the surrounding tunnels, see Figure 3-7. Four geophones with measurements in three directions were set up in each of the parallel tunnels thereby determining the vibration levels. Readings would then, as a reference, be compared with measured vibrations from conventional tunnelling in the Äspö HRL (Nyberg et al. 2009). A proper comparison with vibrations from the NCC's regular tunnel blasting at the site could however not be made as corporative charge differs, and the distance between the blast and measuring point above ground gives another condition.

The company Ansvarsbesiktning AB carried out vibration measurements from test blasts. The real position of the test blasts and the measurement points are shown in Figure 3-8. The measuring units in the parallel tunnels were three-axis geophones INFRA V12.

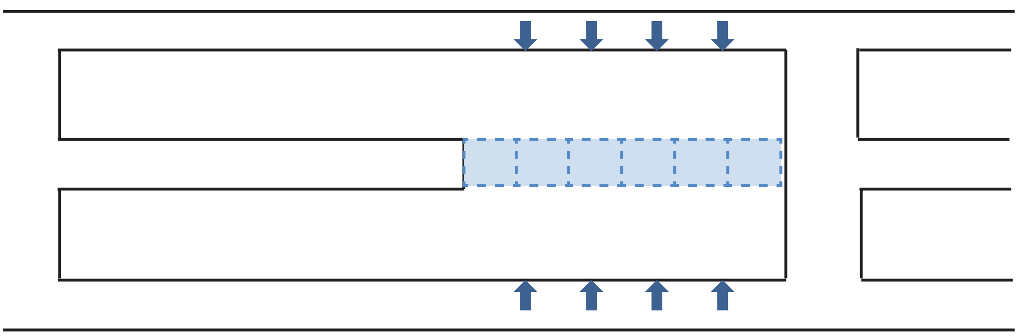


Figure 3-7. Planned location for vibration measurements.

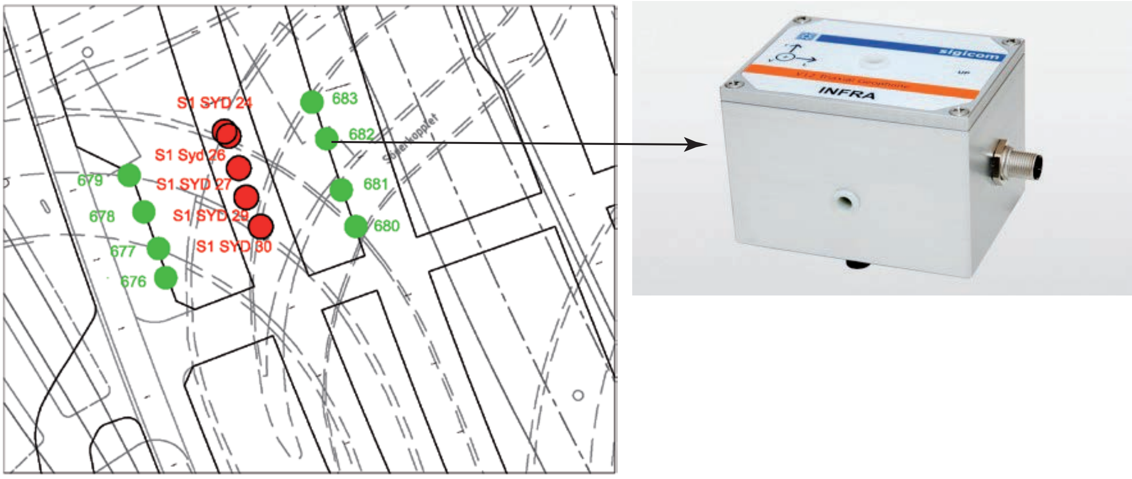


Figure 3-8. *Measuring points (marked in green), test blasts (marked red).*

3.5 Removal of rock plug for the study of blast damages

Wire sawing of the floor, walls and roof give a very smooth contour and the free-sawing of surfaces provides conditions to minimize the risk of blast damage. In order to show that the wire sawing really minimizes the damage zone from the blasting a rock block, a "plug", was wire sawed from the tunnel floor. The plug that was taken out was 8-sided with each side 75 cm and 75 cm depth, see Figure 3-9, and it has then the same diameter, 1.75 m, as a deposition hole.

The location of the plug was in the middle between the walls and about 7 m from the tunnel starting point, which is in the transition between blast 2 and blast 3.

The deposition hole has an inner diameter of 1,750 mm and the bentonite has an outer diameter of 1,650 mm. The ideal would be to provide a nearly circular hole with wire sawing. The dimensions in Figure 3-9 is given by the demand that the holes for the wire sawing of the bottom of the deposition hole must lie within deposition hole diameter, since this operation is done as a final step in the completion of the deposition hole. Eight holes for wire sawing is a compromise that leads to the remaining plug having a diameter of 1,430 mm, but not 1,650 mm that would be desirable, but it was sufficient to see how flat the deposition hole bottom could be sawn.

Therefore it was considered interesting to make the removal of the rock plug as shown in Figure 3-9, as part of discussions on the levelling of the deposition hole bottom, where there is a requirement that the surface must not slope more than 1 mm over the hole diameter.

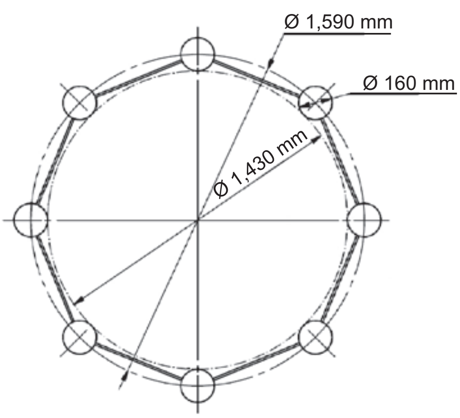


Figure 3-9. *Plan view of the rock plug. The 8 boreholes required to implement wire sawing is inside the deposition hole diameter of 1,750 mm and then the showed measurements are obtained. The depth of the plug was about 0.7 m.*

4 Implementation

4.1 Drilling for wire sawing

The establishment of drilling in phase 1 of the holes for sawing was simple because there was room enough at the face for the drilling machine drive unit as illustrated in Figure 4-1. For the lower corner of the holes no scaffolds were required. Drilling of the upper holes required erecting of a scaffold to obtain a work platform at the correct level.

In performing the corresponding drilling for phase 2 provisional scaffolding was also required to provide a working platform at the desired height, see Figure 4-2.

Figure 4-1. *Phase 1 – The left image shows the drilling of the core drill hole Ø 76 mm. By changing the boom on the drilling unit (right picture) the reaming of the borehole core in two steps was made, from Ø 76 mm to Ø 165 mm, then from Ø 165 mm to Ø 255 m. No new scaffold was required for these reaming steps.*

Figure 4-2. *Phase 2 – In order to perform the drilling of the holes on the upper level, the construction of fixed scaffolds and platforms was required. In the actual production of deposition tunnels, the fixed platforms will be replaced with mobile lifting platforms to achieve a rational work.*

4.2 Wire sawing

During the sawing no unexpected problems occurred but when DWT performed a test for determining the maximum production capacity of sawing equipment that caused a wire break. The test consisted of varying the speed and tension of the diamond wire in order to optimize the rate of cutting at sawing.

A wire break created extra working re-establishing the sawing equipment and the diamond wire, see Figure 4-3.

Furthermore, there was an error in the measurement of the length of the wire that caused the cut in the roof not going right up to the pulleys. This did not come to light until the blasting of the rock. It meant that the push rods with pulleys had to be reinstalled in order to complete sawing of phase 1.

Even when sawing the walls the sawing was suspended before the wire came in line with pulleys and consequently also the walls in phase 1 were sawn too short.

4.3 Excavation of free-sawn rock

Drilling and blasting was performed, as well as sawing in two phases. Phase 1 consisted of three rounds and phase 2 of the two rounds. In phase 1 the planned drilling plan was used but with some varying charge types. In phase 2, a different drilling and charge plan was used to avoid some problems that arose in the blasting in phase 1.

The following represents an overview of the drilling and blasting work. A detailed description is presented in Appendix 1.

Figure 4-3. *During the sawing in phase 2 a break of diamond wire occurred (vertical yellow line in the middle of picture at the rock surface).*

4.3.1 Phase 1

Round No 1

The blasting was done 2012-02-15, and the result was a 'cabin', i.e. the blast went well at the very back but the rest remained as a "plug". In the "plug" there was unexploded emulsion in many holes while opening holes had gone out nicely. The remaining plug was reloaded and blasted; now with good results, see Figure 4-4. The rock of the tunnel face was coarse blocky but the sawn contour of the walls, roof and floor was very nice.

Round No 2

Round No 2 was executed 2012-02-23 and the result was a mixture of some oversized fragments and fine fragmenting, see Figure 4-5. The blasting was fairly successful in the bottom. In some holes in the oversized fragments unexploded explosive was left. After unloading a boulder was left with unexploded explosive.

Figure 4-4. *Photo of the face after blasting and loading of round No 1.*

Figure 4-5. *The left photo shows the face after blasting of round No 2 and the photo on the right shows the face after removal of the loos rock material.*

Round No 3

Round No 3, which was loaded with cartridges, was blasted 2012-03-12. After the explosion, there were many oversized fragments and many unexploded charges were thrown out. A reblast had to be done in a few holes then being charged with ½ 25 mm DynoRex/hole. The blast now went into the bottom of the centre section while there were still knobs standing in the sides, see figure 4-6. One problem that became apparent was that the roof was not wire sawn deep enough but there remained a wedge in the roof of the tunnel.

4.3.2 Phase 2

The ambition to keep predetermined look-out angle was maintained for time reasons only in the tunnel floor. In practice the upper part of the tunnel became too narrow. During the drilling for round No 4 it was discovered that even the sawing of the walls were too short. Consequently phase 2 began with the drilling of four new holes for wire sawing and subsequent sawing. For blast in phase 2 only cartridge explosives were used.

Round No 4

Round No 4 was short. It was blasted 2012-04-05 with good results, see Figure 4-7.

Figure 4-6. *Photo of the face after the blasting of round No 3. The right figure shows also the residual crescent-shaped rock knob in the roof.*

Figure 4-7. *Photo of the face after blasting and loading of round No 4.*

Round No 5

Round No 5 was the last blast that was done in this project. The round with a length of 4.5 m, was made 2012-05-14 with good results. The blast caused a break through.

4.3.3 Experiences from tunnelling

The tunnel excavation took much longer than planned. Round No 1–3 in phase 1 followed the planning and consisted of standard tunnelling with parallel holes. However, more reblasting was needed with sometimes large oversized fragments despite a high specific charge. This was probably due to the following factors:

- The tunnel was cross-cut by many distinct fracture planes.
- The gas from the explosives was efficiently ventilated through the four coarse corner holes for wire sawing causing that the generated gas couldn't contribute to the normal fragmentation and throwing out of the rock.
- The wire sawn slots also released the pressure from the explosive gases.

The combination of these factors reduced the borehole pressure, released the gas pressure and prevented effective fragmentation of the rock, see Figure 4-8.

It was therefore decided to change the drilling and charge plan during phase 2. The drilling and charge plan with the oblique holes towards the sawn slit worked better and Round No 4-5 could be done with good results.

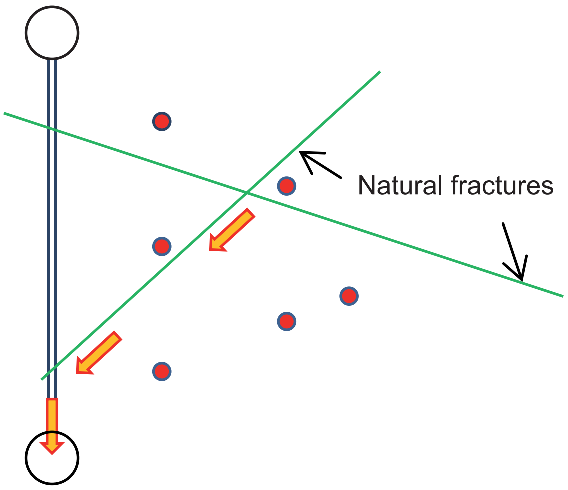


Figure 4-8. *Illustration of likely flow paths for the gases from the explosives. The red dots in the figure show loaded bore holes near natural fractures.*

5 Results

5.1 Geometry and contour posture

The tunnel contour was fine with completely smooth walls, roof and floor. No overbreak occurred and no reinforcement needed to be done. Such smoothness in the contour cannot be obtained with conventional drilling and blasting. In Figure 5-1, Figure 5-2 and Figure 5-3 pictures are shown of the finished tunnel.

The result of the experiment shows that the step between two sawing campaigns, in practice, is in the order of 125 mm in the horizontal plane and 180 mm in the vertical plane.

***Figure 5-1.** Overview photo of the finished blasted tunnel, phase 1 closest to the camera.*

***Figure 5-2.** Detail of the finished blasted tunnel. The light field is the step between phase 1 and 2.*

***Figure 5-3.** Detail of the finished blasted tunnel, the left wall at the transition between the two phases.*

After completion of the excavation, the tunnel was surveyed by laser scanning to investigate the precision of the contour and precision in the holes for wire sawing. The entire tunnel was scanned, but the results are only reported for phase 1 as phase 2, for time reasons, was done with less accuracy regarding drilling of the holes in the roof. In reality the tunnel upper part became too narrow in phase 2. Laser scanning results from phase 1 are shown in Figure 5-4.

***Figure 5-4.** Results of laser scanning, phase 1.*

The drilling of phase 1 was done with a look-out angle. The precision in drilling of these pilot holes is critical for how well the results of the sawing will be in terms of the contour posture. Figure 5-5 shows two cross sections, the first one 0.5 m into the tunnel, start phase 1, and the other 9 m into the tunnel, the finish of phase 1, where the final scanned tunnel model is compared with the theoretical square contour. In the figure the right bottom borehole line has been used as a common fixed point through the entire model. The figure shows that the final tunnel cross section was twisted to the left and both the lower and the upper holes at the left wall have been directed towards the left. This is probably mostly due to a misdirection in the set up for drilling, as the deviation in the hole straightness is within ± 2 cm.

The smoothness of the sawn surfaces was also investigated using the scanned model. A theoretical surface was determined from the edges of the drill holes and the distance to the scanned surface was calculated. The results are shown in Figure 5-6 and Figure 5-7. The deviation from the theoretical surface is maximum ± 40 mm.

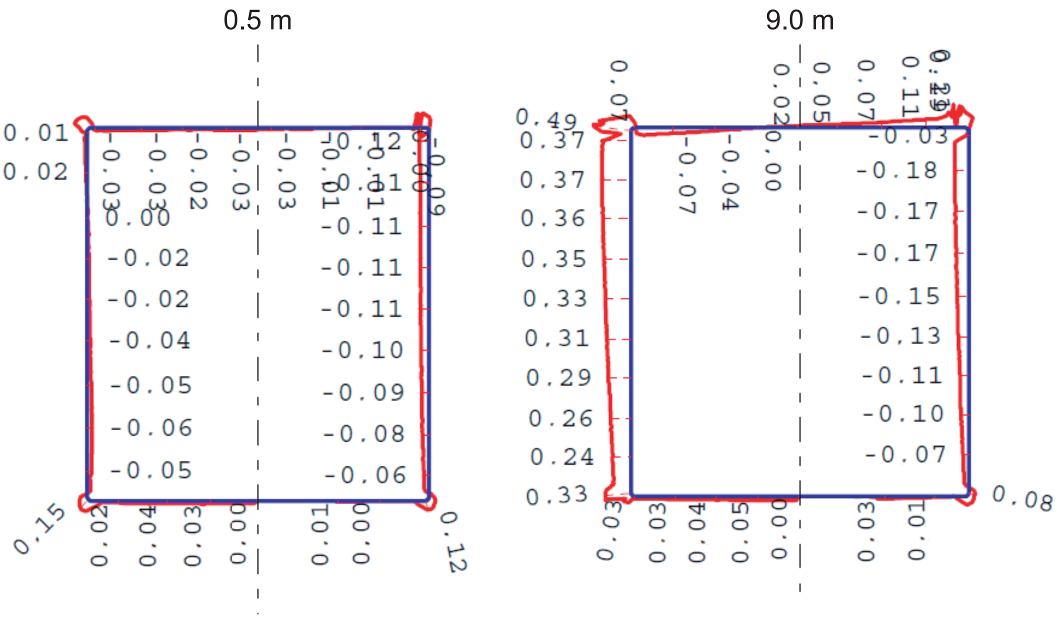


Figure 5-5. *Comparison between the surfaces of the scanned model and the theoretical square model. Dimensions are in metres.*

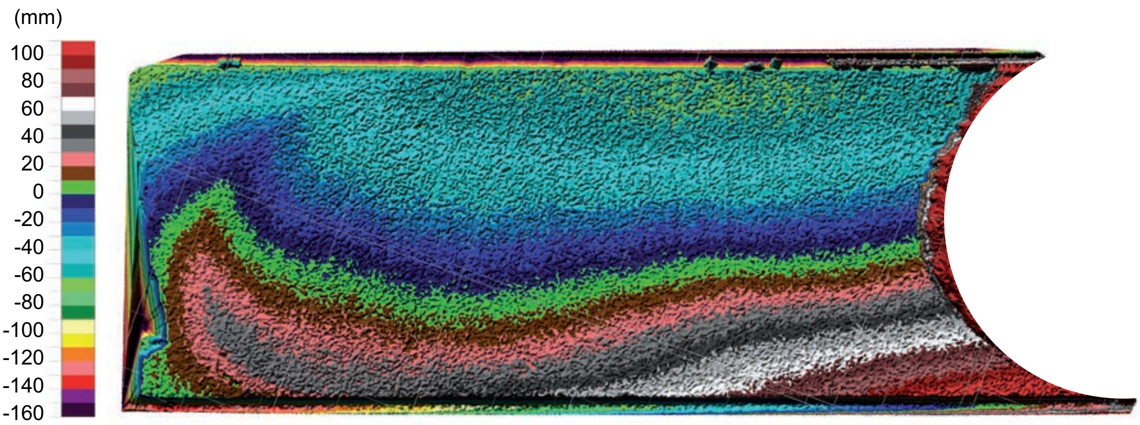


Figure 5-6. *The surface smoothness compared to the theoretical surface from drilling. This figure shows the left wall. The scale of the deviation in millimetres.*

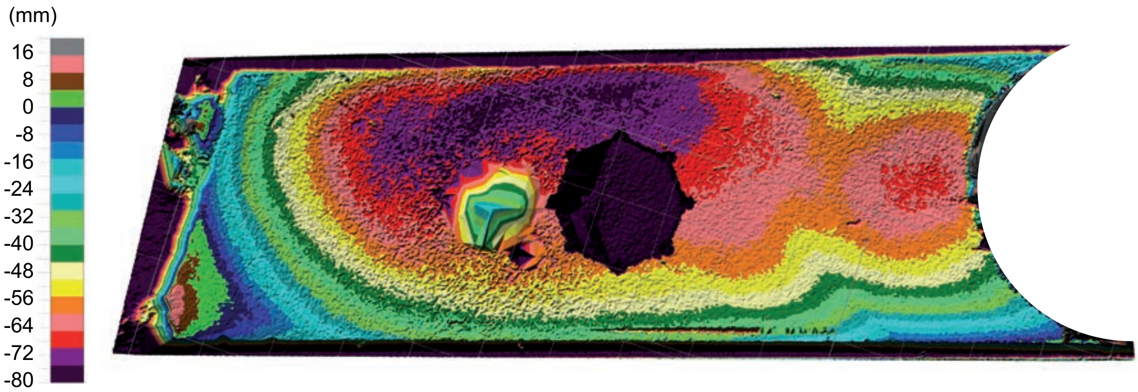


Figure 5-7. The surface smoothness compared to the theoretical surface from drilling. This figure shows the tunnel floor. The scale of the deviation in millimetres. The dark hexagon is the plug that was taken to study the EDZ, See Figure 3-9.

5.2 Follow-up on blast damage

5.2.1 Radar measurements

Ground penetrating radar measurements (GPR) have been used in drilled and blasted tunnels to investigate the blast damages (EDZ) and its propagation from the tunnel surfaces into the rock and along the tunnel. Blast damages or blasting fractures are often small and oriented in a way that makes it difficult to detect them with ground-penetrating radar as interpretable reflectors or diffractions, as in ordinary use of ground radar. Likewise, newly formed fractures can be difficult to detect because there may be no electrical contrast between the fracture and the rock. However, elevated porosity, caused by the increased frequency of fractures in comparison with the surrounding rock lowers the resistivity in areas for blast damage. At high radar frequencies the resistivity is highly dispersive, that is frequency-dependent, which means that even small differences in the resistivity can be observed by using this characteristic. In determining the presence of blasting fractures with this method a dispersion index is considered, which, after calibration against the normal resistivity in the area (electrical properties), gives an idea of the intensity and depth of blast damages.

Blasting damages in the sawed tunnel was investigated with GPR measurements in the tunnel floor; the assumption was that blast damage could occur there, because the free-sawn block has the best contact at the tunnel floor. GPR measurements on the floor of phase 2 was made in 2012-05-29, see Figure 5-8. Measurements could not be carried out on the floor for stage 1 because work with the rock plug was going on in that area. In total 40 parallel, 6.1 m long parallel scan lines, were measured with a 10 cm spacing. The result is shown in Figure 5-9 where the final outcome of the investigated area is displayed as a colour map of an interpretation of the EDZ's extent and depth.

Figure 5-8. Image of the measurement area of ground penetrating radar (left) and an image of the measuring device (right).

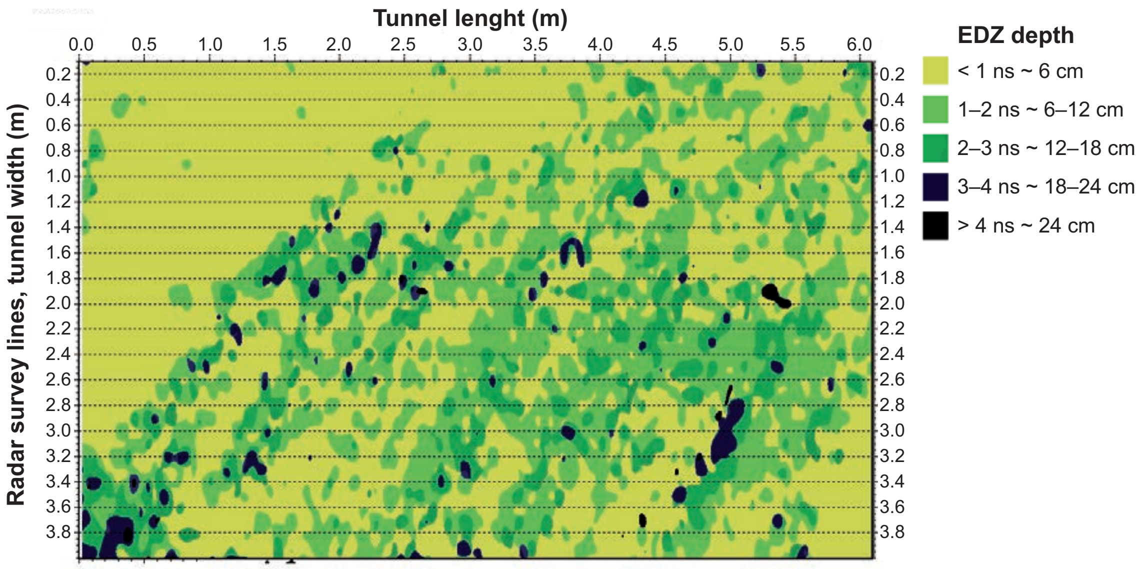


***Figure 5-9.** Interpreted EDZ deepth in the floor of the sawn tunnel. The measurements were performed on the floor of the phase 2 on a length of about 6 m and a width over the entire tunnel with a lateral shift of 10 cm between the measurements. The mica-rich portions cause anomalies with apparent larger damage zone depth in the gneissic part.*

As a calibration the resistivity of the normal granite has been used, which also can be seen in the results. Figure 5-10 is a close-up of the measuring area where the fluctuations between granite and foliated gneiss can be seen. The different lithological units and variations in intense of the foliation are clearly seen in the results. This explains the interpreted deeper EDZ areas in Figure 5-9. The gneiss and especially the mica-rich portions have different resistivity, which means that the results presented here are not reliable for those areas. Looking at the results for the granitic area it can be concluded that according to the radar, there is no blast damage there. The possible damage in the gneissic part it cannot be estimated due to lack of calibration data.

***Figure 5-10.** Detail of the tunnel floor in phase 2 where ground radar measurements were made. The rock varies from granite to foliated gneiss.*

5.2.2 The extraction of the rock plug

The extraction of the rock plug was done in the same way as for the tunnel, but as the extent of sawing was limited a smaller pulley was chosen for the wire. The holes for the rods with pulley are 160 mm. After drilling the slots between the holes were wire sawn in the same way as the tunnel and finally the floor was wire sawn under the plug. Figure 5-11 shows some photographs from this work. All sides of the plug were very smooth.

5.2.3 Mapping of fractures in the rock plug

The plug, which weighed about 4.5 tons, was transported to DWT Teknikk Halden where it was wire sawn in two parts, perpendicular to the direction of the tunnel, so as to obtain a section which is perpendicular to the tunnel blast drill holes see Figure 5-12. For mapping of damages, a technique with dye penetrant was used which exposes possible damages. The technology for this is method is described, among other things in Olsson et al. 2009.

The sawing gave a very fine surface, which after cleaning and drying was treated with penetrant. Figure 5-13 and Figure 5-14 shows a sawn surface with and without penetrant dye.

Blasting normally causes a damage zone in the remaining rock. When careful blasting is used in tunnel contours the fractures will usually normally be 15–40 cm depending on how the blast is performed. There was no formation of blast induced fractures observed in the sawed surface. This is in good agreement with the GPR measurements in the tunnel. There were no blast induced fractures in the sawn surface. The penetrant has only penetrated locally in some natural fractures.

Figure 5-11. *The left photo shows the drill holes and sawn sides (cf. Figure 3-9). The right photo shows appearance when the rock plug is removed.*

Figure 5-12. *Photo from the investigation of rock plug in Halden, Norway.*

Figure 5-13. *Photo of the sawn surface of the rock plug. The section is perpendicular to the tunnel direction.*

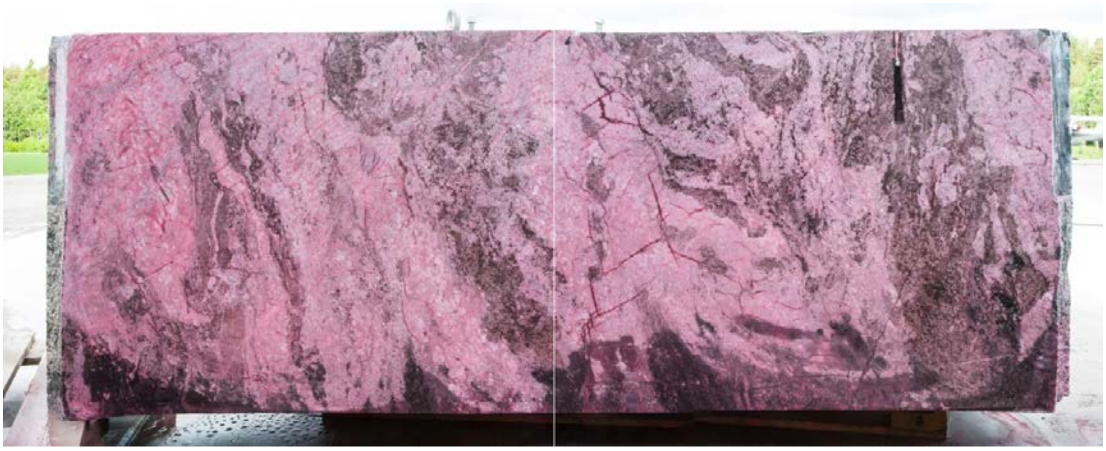


Figure 5-14. *Photo of the rock plug sawn surface with penetrant.*

5.3 Blast induced vibrations

The cooperative charge, i.e. the amount of explosive that is initiated simultaneously in the blast, varied between the blasts. The first two blasts had a co-operating loading of 3 kg, while the other blasts had an interacting charge of 1–1.6 kg. Generally relatively low vibration levels were measured in relation to the short distance that existed between the blasts and the measurement points. There was a tendency that the last two blasts gave lower vibration levels. This may be due to the fact that the rock around these two blast rounds were completely free-sawn while for blasts 1–3 it was free-sawn in the contour but not in the face to be after the blasting.

The measurement results from vibration measurements have been compared to production blasting in nearby tunnels at similar distances and with equal interacting charge. In this comparison it can be observed that the peak particle velocity (PPV) was about 20% lower during blasting of the contour wire sawn tunnels compared to normal tunnel blasts. However, there is considerable uncertainty about the prevailing rock quality which can imply variations in the vibration transmission in the rock.

The measurement results were also compared with previous data from blasts at Äspö (Nyberg et al. 2009). There vibrations were measured during blasting of a tunnel in 2003. Figure 5-15 shows the spread of the measured PPV from the 5 blasts (red rectangle) compared with data from blasting of 34 rounds at Äspö (vertical oscillation component). The cooperative charge was 0.4 to 4.3 kg at the Äspö blasts. As shown in Figure 5-15 the PPV was not extremely low compared to a conventional excavation with approximately the same size on co-operating charges, but the peak is still about 3 times lower than the maximum value measured at Äspö 2003.

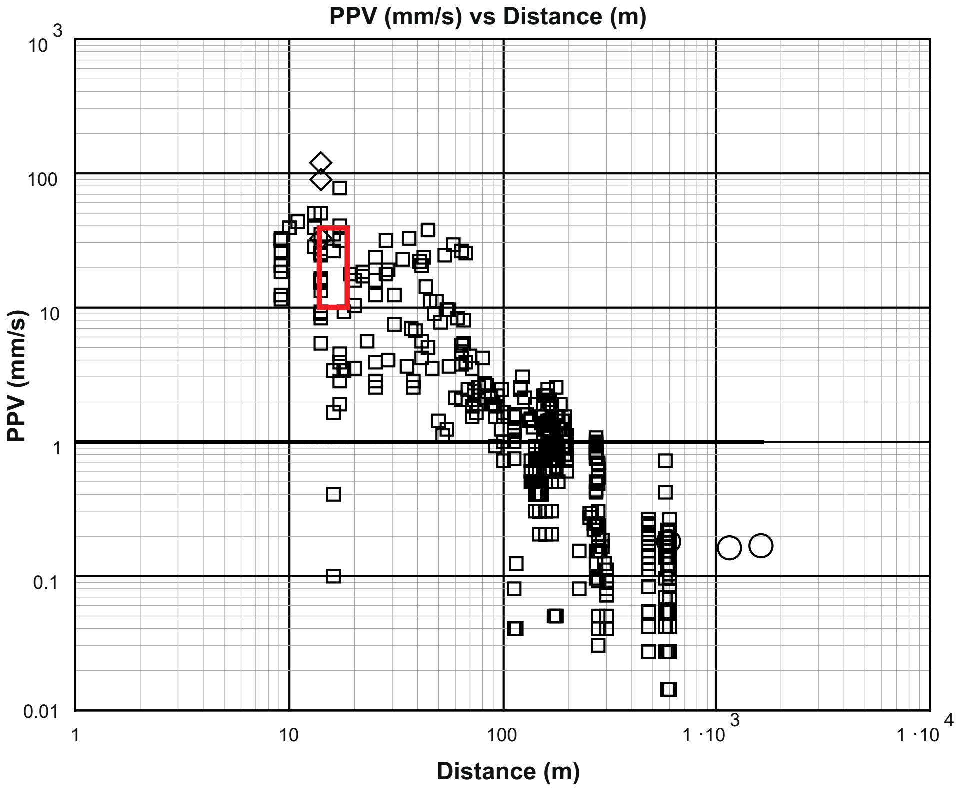


Figure 5-15. *Peak particle velocity (PPV) for each blast with blasting at Äspö 2003 (Nyberg et al. 2009). These data are indicated by the following symbols: □ Geophone 1 D data, ○ Seismograph 3 D data and ◊ Accelerometer 1D data as shown in Figure 4-2 in Nyberg et al. (2009). The red box encloses the vertically measured values during blasting for the sawn tunnel in the City Line.*

6 Costs and resources for alternative excavation methods for deposition tunnels

6.1 Prerequisites

The deposition tunnels for KBS-3V has preliminarily a length of up to 300 m and a cross section W×H = 4.2×4.8 m which gives a theoretical area of about 20 m^2.

The length and cross section through a deposition tunnel is shown in Figure 6-1.

The requirement for the Spent Fuel Repository is that it should be possible to deposit up to 200 canisters per year. Normally 150–160 canisters are estimated to be deposited annually. If we assume a total tunnel length of 300 m about 20 meter will be excluded at the beginning of the tunnel up to the first canister and about 10 m fall off at the end of the tunnel due to, among other things, the deposition machine space requirement at the deposition in the deposition hole in the back of the tunnel.

Then 270 m tunnel remain for deposition of canisters but the utilization has currently been assessed to 85% which means that about 230 m can be used and with 6 m between the canisters an average of 38 canisters can deposited in each tunnel.

In this study, the simplified assumption has been made that five deposition tunnels are required to be ready per year to be able to deposit a maximum of 200 canisters. This means that the annual production of deposition tunnels should be about 1,500 m corresponding theoretically to 30,000 solid m^3, that is fairly small rock volumes.

Below follows a comparison of the production costs for the following three options:

1. Conventional full face drilling and blasting despite SKB's reference method being blasting of a pilot and bench.
2. Rectangular tunnel with wire sawn floor, walls and roof. The excavation of the free-sawn rock is made by conventional drilling and blasting.
3. Wire sawing of tunnel floor only but conventional drilling and blasting of the entire tunnel area in the same way as for alternative 1.

The cost comparison is limited to excavation, i.e. the extraction of the rock, and it is described as the annual cost. The annual cost includes depreciation for machines with an assumed depreciation time of 10 years, regardless of equipment, consumables and labour. The labour cost includes the effective time of the excavation work, as the remaining time is considered to be used for other works in the repository, such as the preparation of transport tunnels or deposition work. All staff is assumed to be employed by SKB with a salary cost of 375 SEK/hour including shift allowances, social charges, etc. Only the production staff and the management were included. Other personnel, e.g. for surveying of tunnel works, service for machines etc have been judged to be roughly equal for the various options and are not included in the cost comparison.

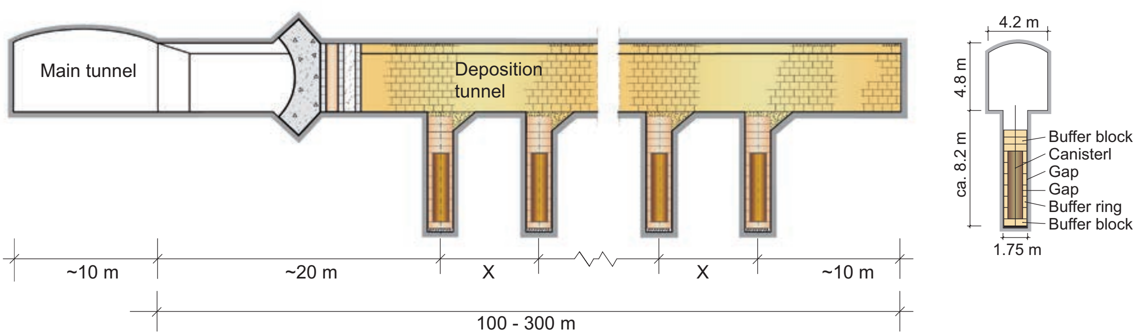


Figure 6-1. Length and cross section through a deposition tunnel. The length section shows a deposition tunnel which is backfilled and plugged while the cross section shows a deposition tunnel where buffer and canister is installed.

SKB will conduct a detailed investigation programme for the adaptation of deposition tunnels and deposition holes to appropriate geological conditions (SKB 2010). This program includes performing of core drill holes to determine appropriate locations for the deposition tunnels. The extent of hose studies is independent of the method of excavation.

The prices have been obtained from experience from projects run by SKB, and from other projects. All prices are from 2012. There is an uncertainty whether supplies always are net prices, or if they sometimes contain the contractor's overhead costs as well. Production cost for wire sawing with the rock conditions for Forsmark with quartz-rich rock is based on a calculation by DWT. Costs to blast within the sawed contour is based on the experience from this project.

Larger items that are not included in the calculation include the mucking out, grouting, rock reinforcement and mapping. The excavated rock volume differs between the different extraction methods. The average cost for mucking out of the rock at conventional excavation and the cost of increased labour and material costs of refilling has been included as it differs between the options. This cost will increase over time as the transport distance to the skip shaft when repository is being expanded gradually increases. Costs of media supplies (electricity, water, ventilation, etc), workshops etc are also not included.

6.2 Description of the different excavation methods

In conventional drilling and blasting a horseshoe-shaped tunnel is normally created. A variant may also be to first excavate a top heading and leave a trailing bench of about 0.8 m to minimize blast damage in the tunnel floor. This allows the tunnelling area for the top heading to be small and it does not permit a rational work even if it is currently SKB's reference method for the extraction of deposition tunnels.

However, if sawing of all surfaces is desired the easiest way is to make a rectangular tunnel with the same width and height as the blasted tunnel. There is also the option that only the floor is sawn which has great advantages when working with different types of transport in the deposition tunnel, drilling of deposition holes, performing the installation of the buffer and canister, and the backfilling of the deposition tunnel.

All the studied methods need to be performed with a look-out angle for providing machine space for the next collaring. Experience from SKB's contract at Äspö 2012 shows that it is possible for a "conventionel drilling and blasting" to include a look-out angle of 25–30 cm. In addition, there is a risk of local faulty drilling and larger overbreak. The experience from this project is that a look-out angle in accordance with Figure 2-3 is likely to be achieved, especially if using established technology for directional drilling. The difference in the exceeding rock between sawn and conventionally driven tunnel is expected to be about 20%, relative to the theoretical volume.

6.2.1 Basis for conventional drilling and blasting

Based on the experience of careful blasting at Äspö for a deposition tunnel the blast round has approximately 96 holes of which 33 are contour holes in walls and roof and 8 boreholes are in the tunnel floor. In addition to this there are also four larger opening holes. The drilling jumbo must be provided with modern control guiding techniques to keep tight tolerances in the drilling of the contour holes. The specific charge is assumed to be approximately 2.85 kg/m^3 rock.

The assessment of the production times for driving nuclear fuel repository deposition tunnels with conventional drilling and blasting has assumed that a drill jumbo with two booms should be able to drill charge holes for a blasting round on three fronts during a working day provided that the work is carried out during two shifts. Time will however not permit both drilling and charging of the three fronts during a working day.

In the calculation it has therefore been assumed that, on average, drilling and blasting can be made with 2.5 blasts on three fronts per working day with two shifts. The round length is assumed to be 4.5 m, but effective advance per round is 4.3 m, or approximately 96% of theoretical length.

This gives an average advance for excavation of 10.75 m/working day. With an advance of 10.75 m per working day about 140 days or 28 weeks are consumed to complete 1,500 m deposition tunnel. The production time is stated briefly in the time schedule, Appendix 2.

The requirements for high precision in drilling the contour holes may imply that the production rate decreases to 2 rounds per working day, i.e. an advance of 8.6 m. The time for completion of 1,500 m deposition tunnel will then be 35 weeks.

It is assumed that the rock workers will be occupied with other tasks when not working with the excavation for the deposition tunnels. The utilization ratio of a drill jumbo is thus more than 60%, which should allow sufficient margin for service and maintenance as well as possible injection.

Option 1, blasted tunnel, will also require an extra effort to cleanup and levelling of the floor. The levelling can be done with removing of tights with hydraulic hammers, scraping/milling and local filling with concrete. At present it is difficult to accurately quantify this cost. An extra charge has been made in the estimate for the cost of equipment for cleaning and possible levelling of the tunnel floor and the costs of additional backfill material and personnel. The uneven floor leads also to disadvantages associated with the drilling of the deposition holes and imposes additional stresses on the machines needed during the deposition and backfilling work. Time for work with backfill will be longer since more work is required on the floor bed prior to backfill blocks to be installed.

6.2.2 Basis for wire sawing of all surfaces

In the alternative with wire sawing of all surfaces it is necessary first to drill and blast an approximately 5 m deep niche to create space for the drilling and sawing equipment so that it does not prevent transport in the main tunnel. The sawing is preliminarily carried out in steps of 50 m to ensure tolerances in the drilling of the corner holes. Experience of drilling long straight holes can later permit the length of the boreholes to be increased. In such a case the number of drilling and sawing phases with associated setup time for completion of a deposition tunnel will be reduced.

This calculation includes the following equipment:

- Two complete drilling rigs for drilling and reaming of core drill holes in two steps with percussion drilling. The drilling rigs shall be designed to drill in the left respectively the right hand sides boreholes. In the calculation it is assumed that this will also be a complete spare drilling rig.
- Two wire sawing machines of next-generation equipment as shown in Figure 7-1. The calculation assumes also that there should be two complete spare sawing machines. These extra sawing machines are used in parallel for a total of six sawing phases, about 48 working days in total.
- Five carrier type mobile service tables, for the equipment for drilling and sawing which permits establishment of the equipment being safe and quick without the need for scaffolding.
- A drilling rig for drilling blast holes. The drilling jumbo need not have an equipment to manage narrow tolerances for the contour holes, which will reduce the investment and maintenance cost for the drilling jumbo. The absence of contour holes makes time for drilling for blasting being much shorter as the drilling of the contour holes requires very accurate setting of the drilling rig.

The drilling precedes the sawing and with dual drilling equipment, the four holes in the corners for a 50 m length can be drilled in about 132 hours in continuous work which is equivalent to 6 working days. Once the drilling is completed sawing can begin by sawing the floor, after which sawing of both walls takes place simultaneously and finally the roof is sawn. The sawing of all surfaces can be performed in about 168 hours as dual devices allow simultaneous sawing of both walls. The sawing then requires 8 working days. The times of the sawing is based on the capacity of the wire sawing of floor and roof being approximately 4.25 m^2 per hour and approximately 6.10 m^2 for walls. In this assessment the high quartz content of the Forsmark rock has been taken into account.

The division of the excavation in 50 meter segments implies that a total of 30 drillings and sawing phases are required for 1,500 m deposition tunnel. The production time for this option are given in Appendix 2. The drilling of holes for sawing will be time ruling if an extra sawing recourse is inserted at six times as shown in Appendix 2.

All drilling for five deposition tunnels corresponding to 1,500 m will theoretically require 30×6 = 180 working days (36 weeks) but some delays will be incurred for the drilling why drilling work is completed after about 190 working days (38 weeks). This requires that drilling and sawing is done continuously in 3 shifts, while drilling and blasting can be carried out in two shifts. When drilling work is completed there remains 12 working days to excavate the rock for the last 50-meter phase. In the schedule referred to in Appendix 2 it has been assumed that there is margin for some production problems.

This capacity assumption is based on access to five tunnel fronts. Drilling and sawing is going on at three faces and the blasting is normally done at two faces where sawing work is done. Based on experience from this project the explosive charge for the free-sawn rock only requires 48 holes and no contour holes.

As the rock is free-sawn, the specific charge is reduced significantly. The experience of this projects shows that a specific charge of approximately 1.0 kg/m^3 rock is enough. The absence of contour holes implies that there is no need to use the electronic detonators which are a significant cost item in the alternative drilling and blasting.

This option affects the backfill of deposition tunnels. With the current design and dimensions of the deposition tunnels, W×H = 4.2×4.8 m, and the current size of the backfill blocks 144 blocks per meter tunnel are required. With a rectangular tunnel with the same size of W×H somewhat more blocks are required, 154 blocks per running meter but less refilling with pellets is required to compensate for the removal of the rock due to look-out angle.

6.2.3 Basis for wire sawing of the tunnel floor only

This option maintains the deposition tunnel horseshoe shape but the floor is wire sawn. This option provides the advantage of a flat floor without EDZ and the sawing efforts are limited to only the floor. In this alternative it is assumed that work is done in parallel at five tunnel faces to achieve the desired production rate. The extent of the wire sawing is less and only drilling for sawing at floor level is performed and this is shown in Figure 6-2.

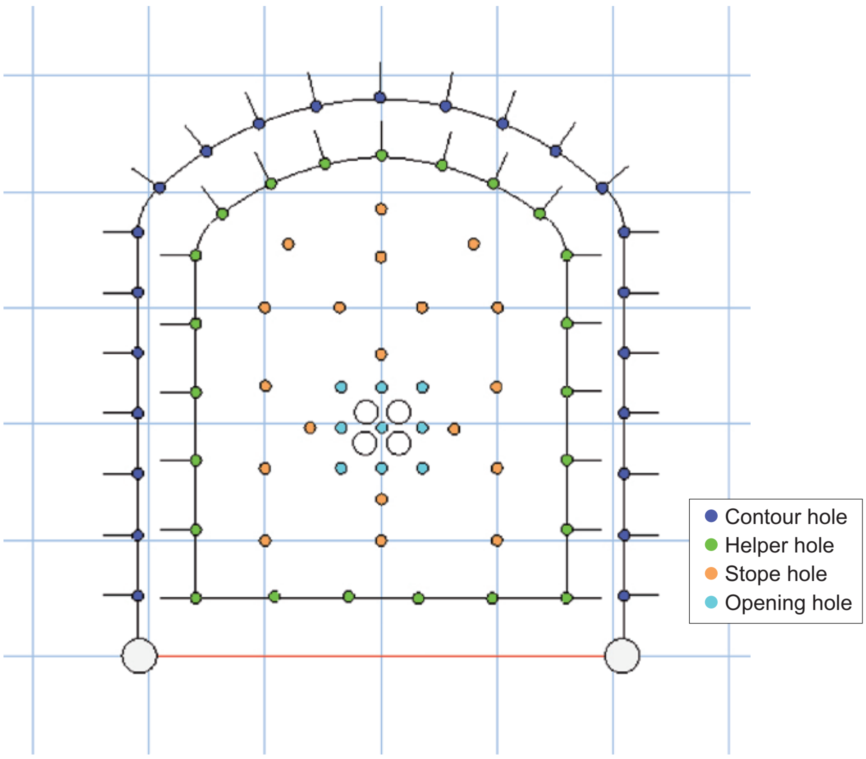


Figure 6-2. *Illustration of the alternative with the floor only sawn. The red line between the coarse holes of figure symbolizes the wire sawn floor.*

This calculation includes the following equipment:

- A modern drill jumbo.
- Two complete drilling rigs for drilling and reaming of core drill holes in two steps with percussion drilling. The drilling rigs shall be designed to drill in the left respectively the right hand boreholes. In the calculation it is assumed that there will also be a complete spare drilling rig.
- A wire sawing machine of next-generation equipment and a complete spare sawing machine.

Drilling for sawing 50 m tunnel can be performed in about 66 hours and sawing of the floor requires about 60 hours, which means that even for this option the drilling time is ruling.

Drilling and sawing of just the floor can then be carried out in much the same rate as conventional drill and blast, and with only 2 shifts during weekdays.

All drilling for five deposition tunnels corresponding to 1,500 m will require 36 weeks in two shifts and 8 hours per shift and the production schedule is shown in Appendix 2. Shift times for drilling for sawing and sawing versus excavation by drilling and blasting will need to be trimmed in order to achieve a practical and economic shift work.

The drilling for blasting decreases with only 8 holes in the tunnel floor compared to conventional drill and blast. The need for electronic detonators decreases correspondingly. Proportionally the production time is reduced with about 8% per blast.

Also this option affects the backfill of deposition tunnels. With the current design and dimensions of the deposition tunnels, W×H = 4.2×4.8 m, and the current size of the backfill blocks 144 blocks per meter tunnel are required. The amount of pellets in a tunnel with sawn floor becomes less compared to the conventional production drilling and blasting because the floor is flat and only needs a thin bed of pellets for buffering of the inflowing water.

6.2.4 Summary of the calculation basis

The production assumptions underlying the calculation of production costs in section 6.3 is reported in Table 6-1.

6.3 Production costs

Based on assumptions of production time and resources the following cost comparison per year are obtained when a total of five deposition tunnels, 1,500 m deposition tunnel will be completed annually.

Equipment for drilling of core holes and reaming requires the same investment for both sawing options but life expectancy is at least twice as long if only the floor be sawn. Furthermore, five working platforms/carriers are required to carry out work rationally at the roof.

Based on the assumptions obtained an annual cost is obtained taking into account the costs that separates the three options according to Table 6-2. This means that the investment costs of the equipment used in all options is not listed, such as charging equipment, but the cost of labour for removal of the rock and extra work for cleaning floors and backfill are included.

It should be noted that there are uncertainties in these estimates of cost so that a deeper analysis of costs is required for the choice of the direction for excavation.

Table 6-1. Summary of cost-influencing differences between the three options.

Difference between options	Conventional drill and blast	Sawn contour. Blasting for excavation	Conventional dill and blast with sawn floor
Investment in machinery.	• A modern two-boom jumbo.	• A less advanced two-boom jumbo. • Two drilling rigs for core drilling with booms for reaming of core drill holes in two steps plus one spare rig. • Five working platforms for drilling the upper corner holes and sawing. • Four wire sawing equipment, two are spare units.	• A modern two-boom jumbo. • Two drill rigs for core drilling with bars reaming for core drill holes in two steps plus one spare rig. • Two wire sawing equipment, one is spare unit.
Working time.	Two shifts, 5 days/week.	Excavation two-shifts. Drilling/sawing 3 shifts, 5 days/week.	Two shifts, 5 days/week.
Personnel per working day and shift.	1 foreman and 2 miners/shift.	1 foreman and 2 miners/shift . 1 foreman and 4 drillers/sawyers/shift. Extra sawing resources 2 man/shift, 48 working days .	1 foreman and 2 miners +1 foreman and 3 drillers/sawyers.
Time needed to produce 1,500 m deposition tunnel/year.	28 weeks (140 days).	38 weeks (190 days)	36 weeks (180 days).
Number of fronts as needed to meet production.	3 pcs	5 pcs	5 pcs
Plus items for the option.	• Requires minimal staff intervention. • Conventional mining operations; standard operating procedures can be applied.	• Smooth contour on the floor, walls and sides. • The volume of rock is minimized. • No extra work on the floor before deposition. • Ideal conditions for the works and runs required in the deposition tunnel. • Minimal damage zone.	• Smooth contour on the floor which provides ideal conditions for the works and runs of vehicles required in the deposition tunnels. • No extra work on the floor before deposition. • The amount of exceeding rock smaller than with conventional excavation..
Minus items for the option.	• Large cost for preparation and cleaning the floor in the tunnel. • High costs for over break. • Large extra costs for backfilling of the tunnels, both materials and installation.	• Requires the largest personnel resources compared with other options. • High cost of personnel and consumables. • This work will involve three shifts during weekdays over 38 weeks per year. In addition, 12 working days with rock work for the final 50-meter phase. • The risk of disturbances is greater than for the other options.	• Exceeding rock and resulting need for more backfilling. • Requires larger personnel resources compared with conventional excavation. • Higher costs for equipment compared with conventional rock excavation. • When drilling and sawing is ready after 36 weeks 12 working days are still needed for excavation of the last 50-meter of the tunnel.

Table 6-2. Compilation of annual production costs for the three options.

Annual costs in MSEK	Drilling blasting	Sawing all surfaces	Sawing only floor
Capital costs	2.5	2.4	3.5
Consumables	4.8	10.3	6.8
Labour	2.5	9.6	6.6
Additional cost for extra exceeding rock	1.4	0	1.1
Cost for tunnel floor and levelling and cleaning	3.8	0	0
Extra cost of backfilling	9.6	0	7.2
Total:	24.6	22.3	25.2

7 Analysis

7.1 Production times

In conventional drilling and blasting the advance is estimated to10.75 m per working day with two shifts. Then 1,500 m deposition tunnel can be completed in approximately 28 weeks as per Appendix 2. It may also be possible to work only long single shifts for drilling and blasting with some flexibility. There is also the option of just blasting on two fronts but then the production time increases.

The option to cut all surfaces, i.e. floors, walls and roof, is the most labour intensive method and requires virtually continuous work over about 38 weeks a year to manage drilling and sawing surfaces for all blasting operations. It also means that the work must be conducted at five tunnel fronts. Appendix 2 shows that drilling and sawing must go on for 14 days before the work on the last excavation can begin. That work is in progress for 12 working days after the sawing of the last phase is complete. At steady-state the drilling and sawing personnel will begin with next year's needs of deposition tunnels and therefore the production cost is based on 38 weeks of work.

In the option of just sawing the floor the drilling and sawing of the floor can be performed in 36 weeks with two shifts as shown in Appendix 2. Also this option will require 12 working days to carry out the excavation of the last 50 meter phase.

The blasting will need to be done at the end of each working day, and it also means that the drilling for sawing and the wire sawing then must be interrupted. In this analysis it has been assumed that one hour falls off during each shift, even if only one shift per working day will be affected by blasting and ventilation of explosive gases. The adaptation of the shift time is therefore important to minimize disturbance in connection with blasting.

7.2 Costs

The cost comparison in Table 6-2 reports only differing costs for the alternatives and does not give the total cost of excavation for five deposition tunnels, with a length of 300 m each per year. The calculations are based on actual cost data from completed construction projects but they contain nonetheless a lot of uncertainties. Accurate data and analyzes will be required in order to reduce the uncertainty in the cost estimates for the various options. The expenses which requires careful analysis is mainly the cost of levelling and cleaning the tunnel floor and extra cost for backfilling. Driving on an uneven floor will imply additional strain on the vehicles which are driving in the tunnel but this cannot be quantified at present.

The option of sawing all surfaces shows lower cost when compared to conventional excavation but there are uncertainties in the estimate for drilling and sawing versus conventional excavation and costs of levelling, cleaning and backfilling are very uncertain.

The alternative of only sawing the floor will be slightly more expensive than conventional excavation but the cost difference is negligible. The sawn floor provides great advantages in the process of drilling deposition holes, less disturbances and stress of the equipment to perform the installation of buffer, deposition of canisters and installation of the backfill material. These advantages have not been assessed in terms of cost but they should be fairly large.

In the conventional excavation by drilling and blasting, the cost of the rock from overbreak and consequently extra material of backfilling provide a significant item to charge this option. Besides involving extra work arising from the levelling of the tunnel floor before placement of backfill blocks, an uneven tunnel floor implies extra equipment wear and the levelling with concrete may be required when drilling the deposition holes. If this concrete levelling can be left or needs to be removed is not investigated. This means that certain costs and their sum are not fully understood at present.

Assumed cost for the over break and extra backfilling means that the option of sawing all surfaces has the lowest total cost. However, there are large uncertainties in these cost items. A more accurate cost estimate is required at a later phase when the cost of the mining industry, the backfill material and the positioning of the material can be substantiated accurate.

The option of sawing all surfaces is appealing and may be the overall cheapest option if all cost factors are included. Labour input and drilling of the pilot hole required for sawing and the cost of wear of the diamond wire indicate that in-depth analyzes of technology and cost must be implemented for the selection of alternatives. Given that excavation for deposition tunnels might be about 10 years into the future, it is possible to implement this type of cost analysis and follow the development of technology for both conventional drilling and blasting and for wire sawing.

7.3 Rationalisation benefits

Sawn contour minimizes excavation and materials for backfilling, and is judged to be very rational for the deposition and backfill, compared to conventional excavation. These benefits have been cost estimated, excluding any time savings for backfill in the deposition tunnels.

The sawing means that no excavation damage arises as was demonstrated in this project. Lack of EDZ is estimated to have significance for the long-term safety, but this cannot be quantified in this stage. Theoretically, it might be possible to reduce the depth of the deposition holes as the current depth is originally based on the circumstance that there should be a margin of safety for blast damage to the floor which does not occur when sawing. Furthermore, the sawn profile possibly requires less scaling and rock reinforcement efforts, but the cost of these works cannot be quantified at present.

Wire sawing implies no damaged zone in the tunnel floor and will bring about optimal conditions for the activities which will then be performed in the deposition tunnels. The main activates will be different transports as well as drilling of deposition holes, installation of buffer, deposition of canister and finally refilling of deposition tunnel and plugging.

7.4 Development trends

DWT has for many years worked to develop the equipment for sawing from the outside inwards with diamond wire and the first generation of sawing equipment was primarily designed for vertical sawing to create shafts for such as ventilation or high voltage cables. The purpose of the tests in this project was to test the sawing in horizontal surfaces. The challenge at horizontal sawing is that gravity affects the wire. It is thus difficult to maintain the wires position both laterally and longitudinally over longer distances. The wire's own weight makes the cut "falling" in the longitudinal and transverse direction. The extent of this depends on the tension of the wire. In this experiment DWT has shown with current equipment that horizontal incision < 15 m can be performed with no loss of precision. With the next generation of saw equipment, which is under development, preliminary results show that equipment can maintain accuracy up to 50 m horizontally wide sections.

The development of the next generation of sawing equipment has been going on for some years. At one stage in the development the long push rods with pulleys for the wire were mounted on the same machine that normally performs drive and tension of the wire. However, this is not possible when cutting of walls and roof unless extra pulleys for diamond wire are installed.

The new equipment will move in the same way as a TBM machine having grippers that alternately fix the rear and front part of the machine. There is also cylinders between the front and rear of the equipment that allows the movement in the hole direction.

After moving the equipment and locking of the front and rear part of the machine, sawing continues by the diamond wire being run with the selected speed and tension until the cut is aligned with the front pulleys. Then the process is repeated with the machine moving forward one step and the sawing process can continue along the drilled pilot holes in the rock. Figure 7-1 illustrates a situation where the front pulleys are moved forward and the wire is about to treat the rock until the wire becomes straight between the front pulley sheaves.

Figure 7-1 shows an indicative illustration of the design of the next generation saw equipment. With this configuration of the equipment, it is possible to saw both sides simultaneously by the use of double equipment, which shortens production time.

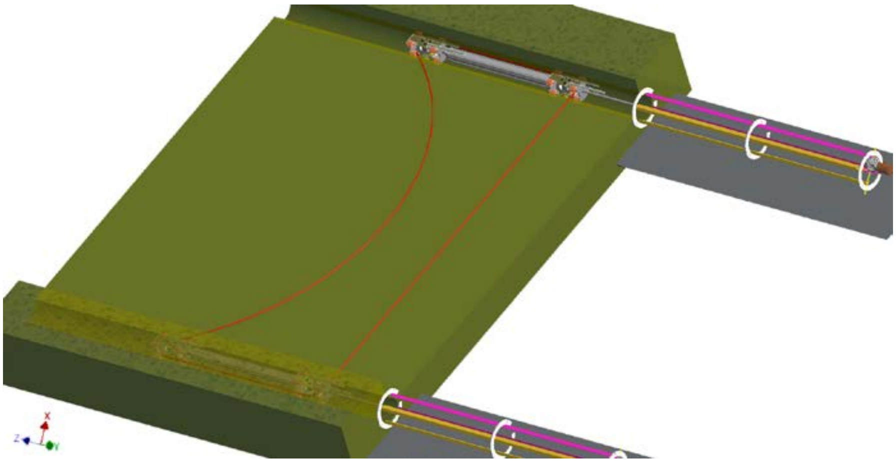


Figure 7-1. *Illustration of the next generation of sawing equipment for sawing from the outside and into the rock.*

It will require an approximately 3 m long carrier with "starting pipes" bolted to the face for this third generation equipment. Starting pipes have the same inner diameter as the bored holes and are required in order to start sawing. The diamond wire runs from the pulley wheels in the front of the self-propelled units and to their end, and the wire forms a closed loop as illustrated in Figure 7-1.

The starting pipes are slotted so that the diamond wire can saw from the front of the tunnel and inward so far as the corner holes are drilled. This design of the equipment enables better opportunity to optimize the production capacity and production costs for the wire sawing.

Also the development of technology to drill longer boreholes with desired precision would help to increase productivity as well is the critical path. Longer boreholes yield fewer phases which reduce the cost of the work.

7.5 Summary

The project of sawing all sides for a deposition tunnel was carried out to demonstrate and compare technologies and costs for excavation of the deposition tunnels through conventional drilling and blasting or by wire sawing floor, walls and roof.

The project condition within a large tunnel undertaking was not optimal, but this relatively limited demonstration experiment shows the possibility of sawing deposition tunnels. Some overall technical conclusions can be drawn:

- Established technology for directional drilling was not applied because only short boreholes were included in the project.
- Sawing provides very smooth surface, quite superior to conventional drilling/blasting technology.
- The blasting plan to break out the sawn rock called for adaptation, probably mainly due to explosive gases finding leakage paths through the cracks into the sawn gap, instead of breaking, as in conventional tunnelling
- In this experiment, the blast damaged zone was nonexistent.
- The test has provided valuable information on the production times for drilling, reaming of holes and sawing.
- From a production point of view the excavation with completely sawn contour seems slightly cheaper than conventional drive, but there are several parameters regarding cost saving in less overbreak, smoother contour and less backfill material that needs to be analyzed in more detail.
- The option to cut only the tunnel floor is more expensive than conventional drive because both drilling and sawing equipment is needed but the option is expected to provide substantial benefits to the deposition and backfilling work. This option is also expected more robust than sawing the entire contour as it mainly uses the conventional rock-driving technology.

8 Recommendations

Based on the results of the project it is advised to continue following closely the development of technology and costs for wire sawing of rock by sawing from the outside and inwards. A number of questions have been identified that must be studied:

- It is production effective to extend the pilot holes for wire sawing in order to reduce the number of sawing phases. The calculation is based on a drilling length of 50 m, but if drill length can be increased to 100 m the number of arrays of equipment can be cut in half. This would tentatively offer cost savings of approximately 400,000 SEK/year.
- The ability to drill long straight drill holes is investigated in other SKB projects. Among others the KBS-3H project is planning to drill a 300 m long pilot hole as a demonstration project.
- This study has strictly been based on tolerances according to the current reference design for the deposition tunnels. Which variations in tolerances can be tolerated? Drill holes deviations may go any way any time, to keep the floor at the right level is probably more important than whether the tunnel area locally becomes too narrow. This can be adjusted in conjunction with the blasting of the finished contour. It is recommended to investigate what it means if the look-out angle becomes too large, or how correction of any faulty drilling outside the intended contour can be made. Different scenario analyzes need to be conducted.
- The benefits and cost savings with a flat floor instead of a raw, blasted surface should be better quantified for the deposition and backfilling works.
- Uncertainty in the sawing capacity in the quartz-rich rock in Forsmark should be reduced in the future. Knowledge of the wear of the wire should be ensured by testing for obtaining a safer estimate of the capacity and cost.
- Further work on developing the next generation of sawing equipment should be followed.
- Definitive technology choice can be made when the results of planned and ongoing development projects regarding tunnelling and backfilling exist.

References

SKB's (Svensk Kärnbränslehantering AB) publications can be found at www.skb.se/publications.

Nyberg U, Harefjord L, Bergman B, Christiansson R, 2009. Äspö Hard Rock Laboratory. Monitoring of vibrations during blasting of the APSE tunnel. SKB R-05-27, Svensk Kärnbränslehantering AB.

Olsson M, Markström I, Pettersson A, Sträng M, 2009. Examination of the Excavation Damage Zone in the TASS tunnel, Äspö HRL. SKB R-09-39, Svensk Kärnbränslehantering AB.

SKB, 2010. Ramprogram för detaljundersökningar vid uppförande och drift av slutförvar för använt kärnbränsle. SKB R-10-08, Svensk Kärnbränslehantering AB. (In Swedish.)

Appendix 1

Detailed description of drill and blasting work

For the drilling of the blasts an Atlas Copco Rocket Boomer 353 unit was used. The drill rig had three booms and was slightly oversized for this tunnel, see Figure A1-1. Two of the blasts were loaded with SSE using a charge car from Orica, see Figure A1-2. Three of the blasts were charged with cartridged explosive.

The charges were distributed with a big wheel loader and after excavation the roof and walls were checked for loose stones by means of scaling. The rock in the area was worse than expected and the area was intersected by many distinct fractures.

Phase 1

This phase consisted of three rounds with a length of 3.5 m each, based on a planned drilling plan with parallel-hole aperture with four coarse opening holes, see Figure 3-6. Round No 1 and 2 were charged with SSE while round No 3 was loaded with cartridged explosives.

Round No 1

Round No 1 was drilled 2012-02-14. Figure A1-3 is a photo of the tunnel face and displays the drilled holes for round No 1. The round was charged with SSE but not really after the proposed charge plan, as shown in Table A1-1. The charge was increased in helpers and stope while the amount of charge was reduced in the opening. As a primer PentexTM 25 was used having a length of 150 mm and weighing 25 g. Specific drilling was 5.2 bm/m^3 and charging 3.3 kg/m^3 and maximum concurrent loading was 3 kg, according to NCC's charge log-book.

Figure A1-1. The drilling unit for the drilling of holes for blasting – Atlas Copco Rocket Boomer 353.

Figure A1-2. *Charge car.*

Figure A1-3. *Photo of front pre-drilled before charging the round No 1.*

Round No 2

Two rounds were loaded 2012-02-23. Figure A1-4 is a photo of the front facing round No 2 and on the right the charge of round No 2 is displayed. The round was charged with SSE, see Figure A1–4. Here it appears that the rock was sectioned of fractures that gave rise to the large blocks in the roof. A number of holes collapsed and they were difficult to load with pump emulsion. These holes were instead loaded with cartridges with 17 mm Detonex. One interesting observation was that there seemed to be no slots in the walls, which may indicate that the rock expanded at the blast due to fractures or previous blasts.

Specifications for round No 2 were the same as for the round No 1 in Table A1-1.

Round No 3

This round was drilled 2012-03-06 and loaded 2012-03-07. The round No 3 was charged only with cartridge explosives and each hole was charged according to Table A1-2. The charge free length was about 0.75 m. The specific drilling was 5.2 bm/m^3 and the specific charge 1.5 kg/m^{3,} which is significantly less than emulsion charges in round No 1 and 2.

Figure A1-4. Photo of the front facing round No 2 and to the right is the charge of round No 2

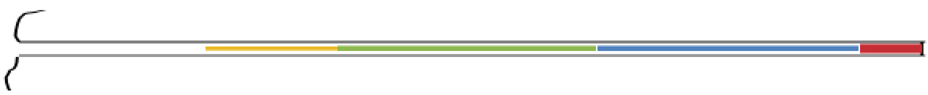
Table A1-1. Charge plan for round No 1 and 2.

Hole	Planned		Real		
	Charge (g/m)	Charge free length (m)	Charge (g/m)	Charge free length (m)	Charge amount (kg)*
Contour	350	0.35	400	0.5	1.2
Helper	800	0.55	1,000	0.5	3
Stope hole	800	0.55	1,000	0.5	3
Opening	1,600	0.3	1,000	0.5	3

*According to NCC blast records.

Table A1-2. Charge plan for Round No 3.

Type of charge	Explosive	Charge length (mm)	Charge amount/hole (kg)
Bottom charge (red)	DynoRex 32 mm	260	0,3
Pipe charge 1 (blue)	Kemix 22 mm	1,000	0,42
Pipe charge 2 (green)	Kemix 17 mm	1,000	0.22
Pipe charge 3 (yellow)	Dynotex 17 mm	460	0.095



Phase 2

This phase consisted of round No 4 and 5. The rounds were drilled without opening hole where round holes now were angled against the sawn slit in the left wall. The holes in these blasts were loaded with cartridged.

Round No 4

Round No 4, which was done 2012-05-04, consisted of 48 pieces of holes with a round length of 2.0 m, the specific drilling was 4.0 bm/m^3 and the specific charge 2.3 kg/m^3. The holes were interval partitioned according to Figure A1-5, which also shows a photo of the drilled front for round No 4. Maximum interacting loading was 1.26 kg.

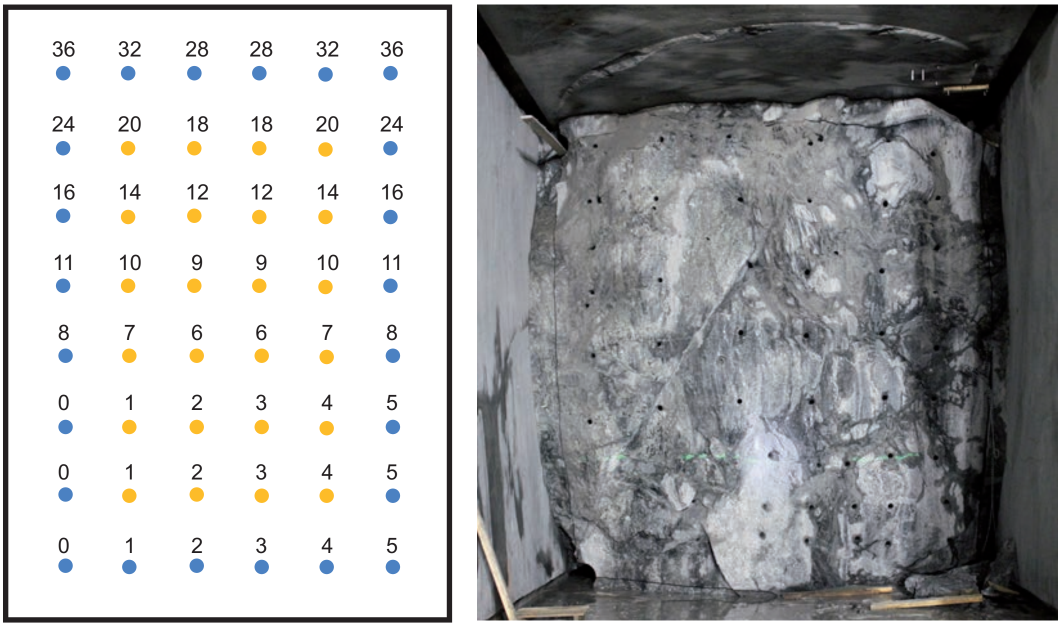


Figure A1-5. *The left image shows the drilling plan for round No 4 with 48 drill holes for the explosives with indication of the firing order. The picture to the right is a photo of the drilled front.*

Round No 4 was charged with cartridged according to Table A1-3. Charge free length about 0.35 m.

Table A1-3. Charge plan for round No 4.

Charge	Explosive	Charge length (mm)	Charge amount (kg)
Contour	DynoRex 25 mm	1,650	102
Bottom charge	DynoRex 32 mm	550	0.58
Pipe charge	DynoRex 25 mm	1,100	0.68

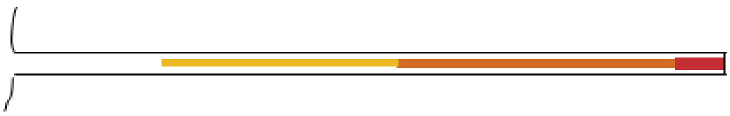
Round No 5

Round No 5, which was blasted on 2012-05-14, consisted of 42 pieces of holes and had a round length of 4.5 m. The specific drill was 2.6 bm/m^3 and the specific charge was only 0.9 kg/m^3. Maximum interacting charge was 1.6 kg. The charging plan for round No 5 is shown in Table A1–4. Round No 5 was loaded with cartridge explosives and the charge-free length was about 1.2 m.

Figure A1-6 shows a drill-and charge plan for round No 5. Round No 5 resulted in an opening to the track tunnel.

Table A1-4. Specific charge for round No 5.

Type of charge	Explosive	Charge length (mm)	Charge amount/hole (kg)
Bottom charge (red)	DynoRex 32 mm	260	0,3
Pipe charge 1 (brown)	DynoRex 25 mm	1,620	1
Pipe charge 2 (yellow)	Dynotex 17 mm	1,400	0.29



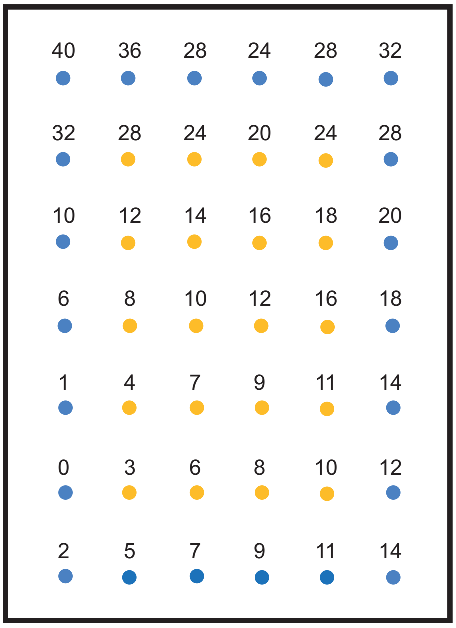


Figure A1-6. *Drilling plan for round No 5 indicating the firing of explosive charge. Round No 5 meant that there became an opening into the track tunnel.*

Production time for the different alternatives

Appendix 2