

Choice of rock excavation methods for the Swedish deep repository for spent nuclear fuel

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Preface

This report is an account of the SKB project “Choice of rock excavation methods for the repository for spent fuel” to compare methods and prepare recommendations and design justification statements for the principal choices.

SKB will make use of the study in the development of the reference design and the site-specific engineering of the repository.

Göran Bäckblom (Conrox) has acted as project manager and lead author of the report and been involved in repository engineering, and evaluation of technologies. Åsa Sundqvist (LKAB), Bengt Niklasson (Skanska Teknik), Ingemar Marklund (GHRR) and Mats Olsson (SweBrec) has prepared the information on Drill & Blast technology. The Robbins Company in the USA commissioned Odd G Askilsrud (Tunnel Engineering and Applications Inc.) and Peter Dowden (Mechanical Tunneling Consulting Service) for the information on tunnel boring machines. Märthen Elgenklöw, Jan Forsberg, Gunnar Nord and Stig Brännström (Atlas Copco Rock Drill AB) has been involved in the work on raise-boring machines and Magnus Hörman (Wassara AB) concerning the cluster drill technology. Hannes Kaalsen (Grinaker-LTA Mining Construction, South Africa) furnished information on shaft sinking. Christer Andersson (SKB) has provided details on excavation of deposition holes at Äspö Hard Rock Laboratory. Information in their respective reports has been extracted for the benefit of this report.

Rolf Christiansson (SKB) has in co-operation with Allan Hedin (SKB) been involved in work on assessment of Excavation Damaged/Disturbed Zone and long term assessment respectively and Leif Lagerstedt (SwedPower) in repository engineering, construction scheduling and cost estimates. Elin Svedberg (Atrax Energi) has contributed to work on environment and sustainable management of natural resources in co-operation with Caroline Setterwall and Petra Sarkozi (SwedPower).

Stig Pettersson (SKB) was the Owner’s representative with Tommy Hedman, Olle Olsson and Fred Karlsson (SKB) acting as members of the Steering Committee. These as well as all other involved are acknowledged for their constructive contributions.

Summary

Choice of rock excavation methods will or may have implications for a number of issues like repository layout, long term and operational safety, environmental impact, design of and operation of transport vehicles and methodology for backfilling the repository before closure as well as effects on costs and schedules. To fully analyse the issues at hand related to selection of excavation methods, SKB organized a project with the objectives:

- To investigate and compare principal technical solutions for rock excavation, both methods that are used at present but also methods that may be feasible 10 years from now.
- To assess how the selection of excavation method influences the design and operation of the deep repository.
- To present a definition of the Excavation Damaged/Disturbed Zone (EDZ) and practical methods for measurements of EDZ.
- To present advantages and disadvantages with different excavation methods for the various tunnels and underground openings as a basis for selection of preferred excavation methods.
- To present the Design Justification Statement for the selection of particular excavation methods for the different tunnels and openings in the deep repository to underpin a decision on excavation method.
- To present background data that may be required for the evaluation of the long term safety of the deep repository.

Alternative excavation methods for the different portions of the repository tunnels and underground openings have been investigated and evaluated against a set number of objectives with focus on excavation of the deposition tunnels. Main alternatives studied are very smooth blasting, excavation with a tunnel-boring machine (TBM) and excavation with horizontal pull-reaming using more or less conventional raise-boring equipment. The detailed studies were carried through in co-operation with major suppliers and end-users of the technology. An observation in this study is that all excavation technologies are mature; no major breakthroughs are foreseen within a 10 year period but it is likely that for any technology selected, SKB would specifically fine-tune the design of the equipment and work procedures in view of requirements and site specific conditions.

Any selection made on present premises and understanding would not prevent change of technology in the future. The excavation of the deposition tunnels is made over several decades and consequences of any future changes in technology would then be scrutinized.

Excavation methods have been compared with respect to a set of factors – long term safety, occupational safety during construction, operation and closure of the facility, environmental impact and sustainable management of natural resources, schedules and costs and finally flexibility, risks and opportunities. The evaluations have been made in Best Available Technology (BAT) perspective. While data are lacking for many instances, advantages and disadvantages are discussed in a qualitative rather than quantitative way.

The main advantage with Drill & Blast is flexibility and cost. The method can easily adapt to a range of rock conditions where tunnel shape and blasting design is adjusted to meet particular requirements. The technology is mature and efficient with resulting good overall economy both with respect to the excavation itself but also with respect to downstream costs of the repository.

The main advantage with mechanical excavation is that the operation is more or less continuous with a very constant and high excavation quality as the human factor cannot impact the quality to the extent possible with Drill & Blast. A disadvantage is that cost is higher, not necessarily due to excavation costs itself, but rather to downstream costs as the circular shape creates voids of no use but that anyhow need expensive backfilling.

Based on reasoning it is suggested that long term safety would not be impacted by choice of excavation method pending that Drill & Blast is executed to minimize the excavation damaged zone and would not create connected flow paths along the perimeter of the deposition tunnels. Also differences in backfill quality or volume or types of remaining stray materials are not significant between different methods.

There is no data to corroborate that Drill & Blast is safer/more hazardous than mechanical excavation. It is rather suggested that Drill & Blast and mechanical excavation present different risks, which once understood can be mitigated by design, regulations, and education.

The overall repository impact on environment and sustainable management of natural resources is to a smaller degree dependent on the selection of excavation methods.

The preliminary Design Justification Statements are as follows:

Access ramp

Drill & Blast is the reference design with TBM as a viable option. The TBM alternative however, can only be used for certain site conditions. The rock should be of good quality and the layout of the ramp feasible to construct with the TBM. In such circumstances the TBM is the preferred option, the main reason being reduction of cost and overall increase of occupational safety as shafts due to faster advance of the ramp can be constructed by raise-boring instead of by shaft sinking. The TBM-alternative assumes that procurement of the TBM is finalised around 12 months before receiving the permit for construction in order to start excavation in accordance with the current master schedule for the deep repository.

Shafts

With respect to schedules, it is assumed that the skip shaft is constructed as a shaft from the ground surface and downwards. Other shafts (hoist- and ventilation shafts) can be excavated by Drill & Blast and by raise-boring. The latter method is preferred in due consideration of costs and the decreased risks for accidents.

Central area

Drill & Blast is the only doable method in consideration of the large underground openings and the irregular shapes.

Pilot- transport and main tunnels

The reference design assumes conventional smooth blasting and this is still the recommended method. Construction of transport and main tunnels by TBM is less favourable as the layout flexibility is low. Pilot tunnels have smaller diameter than the ramp, which provides the opportunity to use a smaller TBM that more easily can construct curves with smaller radii. In case the ramp is constructed with a TBM, the possibility to use TBM for the pilot tunnels should be studied later.

Deposition tunnels for the KBS-3V alternative

All excavation methods studied (very smooth blasting, TBM and horizontal pull-reaming) would be technically feasible and possible to adapt to the requirements and preferences for the repository.

Drill & Blast can still be used in the reference design, but SKB will further study the integrated function tunnel/backfill, where the possibility of hydraulically connected flow paths along the tunnel floor is one of many parameters to consider.

In case mechanical excavation is needed, the TBM methodology would be selected before horizontal pull-reaming in consideration of the overall efficiency and economy. TBM is however a more complicated method for excavation than Drill & Blast.

Deposition holes

Drill & Blast is not a possible method due to requirements on final geometry like surface roughness etc. Two different types of mechanical excavation (down-reaming and shaft boring machine) are viable as both can fulfil the geometrical requirements, but neither of the methods are efficient and further studies are required before selection of method. It is assumed that down-reaming would be a more favourable method than using a shaft boring machine, but further studies are necessary.

Horizontal deposition drifts for the KBS-3H alternative

Cluster drilling technology and horizontal reaming are deemed to be viable methods. Horizontal push-reaming is preferred to pull-reaming as the latter requires an extra service tunnel. Also TBM using button bits gear cutters may be feasible.

SKB now has initiated practical field tests also with horizontal push-reaming to provide a firm basis for later decisions in case the alternative of horizontal emplacement is pursued.

Sammanfattning

Valet av bergbrytningsmetoder har, eller kan ha påverkan på ett flertal faktorer som förvarsutformning, långsiktig säkerhet, driftssäkerhet, miljöpåverkan, utformning och drift av transport och deponeringsfordon, metodik för återfyllnad före förslutning, liksom påverkan på kostnader och tidsplaner.

För att grundligt analysera frågorna som hänger ihop med valet av bergbrytningsmetoder, planerade SKB ett projekt med följande mål:

- Ta fram underlag för, och jämföra principiella lösningar för bergguttar, både metoder som är beprövade, men även metoder som bedöms vara användbara inom en 10-års period.
- Att bedöma hur metodval påverkar förvarets utformning och drift.
- Redovisa definition för sprängskadegren och praktisk metodik för hur denna mäts upp.
- Redovisa för- och nackdelar för olika bergguttmetoder i olika anläggningsdelar som underlag för en rekommenderad utformning och val av metod.
- Redovisa motiv för det specifika valet av metod för bergguttet för de olika anläggningsdelarna i djupförvaret för att underbygga ett beslut om val av metod.
- Ta fram det underlag som kan behövas för det säkerhetsanalytiska arbetet

Alternativa bergguttmetoder för förvarets olika tunnlar och berggrum har undersökts och jämförts mot ett antal mål. Deponeringstunnlarna har varit i fokus där mycket skonsam borrhning-sprängning, tunnelborrningsmaskin (TBM) och horisontell upprymning med mer eller mindre konventionella raise-borrningsmaskiner varit huvudalternativen. Detaljerade studier har genomförts i samarbete med dominerande leverantörer och slutanvändare. En observation i sammanhanget är att samtliga metoder är mogen teknologi; inga större teknikenombrott förutses de närmaste 10 åren; det är sannolikt att SKB oavsett teknik som väljs, behöver påverka detaljutformningen av utrustningar och arbetsformer med hänsyn till satta krav och platsspecifika förhållanden.

Ett val av metoder på basis av nuvarande förutsättningar och förståelse hindrar inte förändringar av tekniken i framtiden. Uttaget av deponeringstunnlar sker över flera decennier och konsekvenser av framtida förändringar av tekniken kommer att utredas grundligt vid ett eventuellt tekniskifte.

Bergbrytningsmetoderna har inbördes jämförts med hänsyn till ett antal faktorer som långsiktig säkerhet, arbetarskydd under anläggande, drift och förslutning av förvaret, miljöpåverkan och hållbar hantering av naturresurser, tidsplaner och kostnader och slutligen flexibilitet, risker och möjligheter. Utvärderingar har skett i ett Bästa Tillgängliga Teknik (BAT)-perspektiv. Då kvantitativ information saknas i många fall, diskuteras fördelar och nackdelar kvalitativt.

Den väsentligaste fördelen med borrhning-sprängning är flexibilitet och kostnad. Metoden kan lätt justeras till olika bergförhållanden där tunneform och sprängdesign anpassas till rådande krav. Tekniken är mogen och effektiv och resulterar i låga kostnader inte bara för själva bergguttet men också för att efterföljande kostnader blir låga.

Den väsentligaste fördelen med mekanisk bergavverkning är att bergguttet är "kontinuerligt" med hög och jämn kvalitet och där mänskliga faktorn har mindre betydelse för slutproduktens kvalitet än vid borrhning-sprängning. En nackdel är högre

kostnader, inte nödvändigtvis för själva berguttaget, men för högre följdkostnader; den runda formen är kräver ökad mängd dyrbar återfyllning.

Baserat på resonemang antas det att långsiktig säkerhet inte påverkas av val av bergbrytningsmetod, förutsatt att borrar-sprängning utförs för att minimera den sprängskadade zonen och att denna inte skapar konnekterade flödesvägar längs deponeringstunnels periferi. Skilda berguttagsmetoder ger heller inte upphov till betydelsefulla skillnader i återfyllningens kvalitet eller i volym eller typ av förekomster av främmande material.

Det finns inga data som underbygger att borrar-sprängning är en säkrare/farligare metod än mekanisk avverkning. Borrar-sprängning och mekanisk avverkning medför olika risker med hänsyn till arbetarskydd, som när de förstås kan begränsas genom utformning, föreskrifter och utbildning.

Förvarets allmänna miljöpåverkan och hållbar hantering av naturresurser är endast till en mindre grad beroende på val av bergbrytningsmetod.

De preliminära motiven för val av berguttagsmetoder följer nedan:

Tillfartsrampen

Borrar-sprängning är referensutformning med TBM som ett möjligt alternativ. TBM kan dock endast genomföras för vissa platsförhållanden, men vid goda bergförhållanden och om en god ramputformning, torde TBM vara att föredra. TBM ger lägre totalkostnader och minskar överlag olycksfallsrisken eftersom den snabbare rampdrivningen medför att skipschakt kan byggas med raise-borrar istället för genom schaktsänkning. Alternativet med TBM förutsätter att upphandling sker cirka 12 månader före tillstånd erhålls om TBM-borrar ska påbörjas i enlighet med nuvarande huvudtidsplaner för djupförvarsanläggningen.

Schakt

Med hänsyn till tidsplaner förutsätts att skipschaktet byggs som ett sänkschakt från markytan och nedåt. De övriga schakten (hiss- och ventilationsschakt) kan byggas med borrar-sprängning eller med raise-borrar. Den senare metoden är att föredra, med beaktande av kostnader och med hänsyn till den minskade risken för olycksfall.

Centralområde

Borrar-sprängning är den enda genomförbara metoden med hänsyn till de stora berggrummen och dess oregelbundna former.

Undersöknings-, transport- och huvudtunnlar

Referensutformningen förutsätter konventionell skonsam sprängning och detta fortblir referensutformningen. Anläggande av transport- och huvudtunnlar med TBM är mindre fördelaktigt eftersom flexibiliteten för anpassning av layouten är låg. Undersökningstunnlar har mindre diameter än rampen, vilket ger möjlighet att använda en mindre TBM-maskin som kan driva tunnel med snävare kurvradier. Om TBM används för tillfartsrampen bör möjligheten att använda TBM även för till exempel undersökningstunnlar utredas senare.

Deponeringstunnlar för KBS-3V alternativet

Samtliga de metoder som studerats (mycket skonsam sprängning, TBM och horisontell dragande upprymning) är tekniskt genomförbara och möjliga att anpassa till de rådande krav och önskemål som ställs på förvaret.

Borring-sprängning utgör fortfarande referensutförning men SKB kommer ytterligare att studera den integrerade funktionen tunnel/återfyll, där möjligheten till hydrauliskt konnekterade flödesvägar längs tunnelgolvet är en av många parametrar att beakta.

Om mekanisk avverkning behövs, bör TBM-tekniken väljas före horisontell upprymning med hänsyn till övergripande effektivitet och kostnad; TBM är dock en krångligare metod för berguttag än borring-sprängning.

Deponeringshål

Borring-sprängning är inte en möjlig metod med hänsyn till de geometriska kraven på ytjämnhet med mera. Två skilda metoder för mekanisk avverkning (nedåtgående upprymning och schaktborrningsmaskin) är möjliga, eftersom båda metoderna uppfyller de geometriska kraven. Ingen av metoderna är effektiv och ytterligare metodstudier är nödvändiga innan val av metod sker. Det bedöms att nedåtgående upprymning kan vara en enklare och robustare metod än en schaktborrningsmaskin, men ytterligare studier krävs.

Horisontella deponeringsorter för KBS-3H alternativet

Klusterborring och horisontell upprymning bedöms vara möjliga metoder. Tryckande upprymning föredras före dragande upprymning, eftersom den senare metoden kräver en extra servicetunnel. Det är också möjligt att TBM, där ytterkuttern är försedd med stift, är en genomförbar metod.

SKB har nu initierat praktiska fältförsök med horisontell upprymning, för att få en fast grund för kommande beslut i det fall att alternativet med horisontell deponering fullföljs.

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1 Introduction

The plan to construct a geological repository for spent nuclear fuel in Sweden has reached the phase of site investigations at the two candidate sites at Forsmark and Oskarshamn. A general description of the overall programme is found in the latest Research, Development and Demonstration Programme /SKB, 2004/. An outline description of the geological repository is conveniently found at SKB's website www.skb.se. Basic engineering of the repository is developed in parallel with the site investigations with the overall objective that the repository is a safe and effective facility that fully complies with international guidelines and standards, national regulations and the general design requirements for the facility /SKB, 2002a/. One of the many issues studied are selection of suitable excavation methods for the repository. Selection of excavation methods will or may have implications for a number of issues like repository layout, long term and operational safety, environmental impact, design of and operation of transport vehicles and methodology for backfilling the repository before closure as well as effects on costs and schedules. To fully analyse the issues at hand related to selection of excavation methods, SKB organized a project with the objectives:

- To investigate and compare principal technical solutions for rock excavation, both methods that are used at present but also methods that may be feasible 10 years from now.
- To assess how the selection of excavation method influences the design and operation of the deep repository.
- To present a definition of the Excavation Damaged/Disturbed Zone (EDZ) and practical methods for measurements of EDZ.
- To present advantages and disadvantages with different excavation methods for the various tunnels and underground openings as a basis for selection of preferred excavation methods.
- To present the Design Justification Statement for the selection of particular excavation methods for the different tunnels and openings in the deep repository to underpin a decision on excavation method.
- To present background data that may be required for the evaluation of the long term safety of the deep repository.

The methodology for optimization (i.e. balancing of factors) was previously used in a project to select access (shaft or access tunnel) to the repository /Bäckblom et al. 2003/ and the same principles have been applied to this project. The principles for optimisation are deemed to be in line with the regulations that Best Available Technology (BAT) is to be used. Compared to any underground facility for civil engineering or for mining additional issues are of importance for the overall optimization:

- **Long term safety:** Will selection of excavation methods impact the long term safety? Issues at hand are for example the creation of an excavation damaged zone. Different methods will also create different tunnel shapes. Diverse stray materials will be produced. Selection of excavation method may influence the quality of the backfilling work. Excavation methods have dissimilar flexibility to allow layout-changes for adaptation of the repository to rock conditions.

- **Operation of the repository:** Selection of methods may influence the layout of the repository that would affect the logistics of the operation. Excavation technology will have an effect on occupational safety during construction, disturbances to the operation of the repository, handling of adverse conditions like fires and flooding etc. One aspect is also the management of detailed characterisation of the bedrock in conjunction with the construction of the repository.

As for any other industrial endeavour, we also have to compare environmental impact, sustainability, costs, schedules, flexibility and project risks involved for the suite of alternatives in consideration.

This report is aimed at providing the factual base and transparent record of the reason for selection of rock excavation methods. Construction and mining companies, machine suppliers and experts has provided the technological know-how and also provided information needed to compare the options for a range of factors of relevance for implementation of the geological repository.

2 Construction of the repository and options for excavation

This chapter is intended to provide a context for the rest of the report starting with an outline of present reference design of the repository and how it will be constructed in steps. Alternative technical options for excavation are described and the main alternatives for this study presented.

2.1 Description of reference design

The reference design for the Swedish deep geological repository, the KBS-3 method, is primarily designed to isolate the waste within the engineered barriers. If the isolation function should for any reason fail in any respect, a secondary purpose of the repository is to retard the release of radionuclides. This safety is achieved with a system of barriers Figure 2-1.

The fuel is placed in corrosion-resistant copper canisters. Inside the five-meter-long canisters, a cast iron insert provides the necessary mechanical strength. A layer of bentonite clay, surrounding the canisters, protects the canister mechanically in the event of small rock movements and prevents groundwater and corrosive substances from reaching the canister. The clay also effectively adsorbs many radionuclides that could be released should the canisters be damaged. The canisters with surrounding bentonite clay are emplaced at a depth of about 400–700 m below surface in crystalline bedrock, where mechanical and chemical conditions are stable in a long term perspective. Should any canister be damaged, the chemical properties of the fuel and the radioactive materials, for example their poor solubility in water, put severe limitations on the transport of radionuclides from

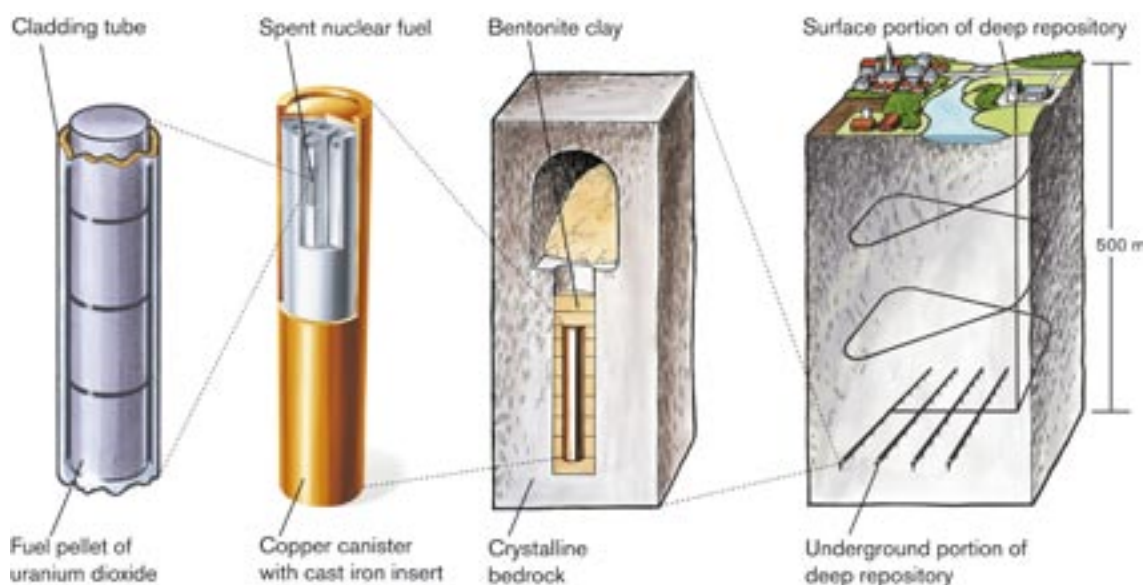


Figure 2-1. The KBS-3 system to safely dispose of spent nuclear fuel. The picture shows the KBS3-alternative when the canister is deposited vertically. SKB is also studying the feasibility of horizontal deposition of the canisters.

the repository to the ground surface. This is particularly true of those elements with the highest long term radiotoxicity, such as americium and plutonium. The repository is thus built up of several barriers, which support and complement each other. The safety of the repository must be adequate even if one barrier should be defective or fail to perform as intended. This is the essence of the multiple barrier principle.

The generic layout for the repository is shown in Figure 2-2 and a description of main underground openings for the reference design at an assumed depth of 500 m below the surface follows. However it should be kept in mind that the dimensions of the opening are preliminary and will be revised as the detailed engineering design continues.

- **Access ramp:** Excavated from the surface at the start of construction. The length of the ramp is around 5,000 m (1:10 decline) with a section area of around 46 m². The access ramp is used for transportation of the shielded canisters (Figure 2-3, Figure 2-4), for transport of the buffer material (bentonite blocks), transports of consumables etc.
- **Shafts:** A number of shafts are to be constructed for transportation and communication. The skip shaft is constructed as a blind shaft from surface, length 570 m, area 24 m², diameter 5.5 m. This shaft will be used for transportation of rock muck to the surface and for transportation of backfilling material to close the underground openings. The hoist shaft is constructed for the personnel hoist (Length 500 m, area 24 m², diameter 5.5 m) and also contains technical systems for drainage. Ventilation shafts are constructed with length 500 m and diameters 2.5–3.5 m, area 5–10 m². Two are for inlet and outlet air to the central area. Power is fed through the outlet air shaft in the central area. Outlet air shaft(s) will be needed in the deposition area as well, but the number of shafts is a site specific issue.

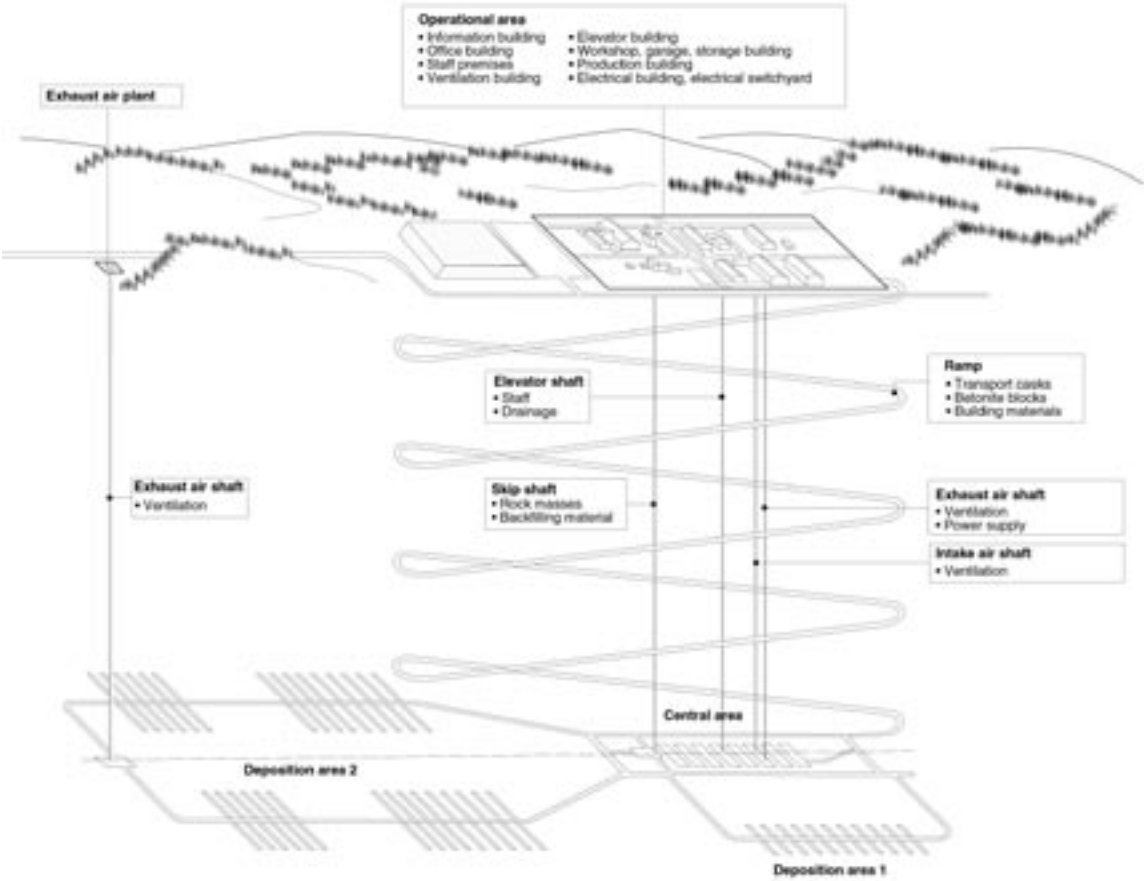


Figure 2-2. Outline of the deep repository and the nomenclature for the portions of the repository.

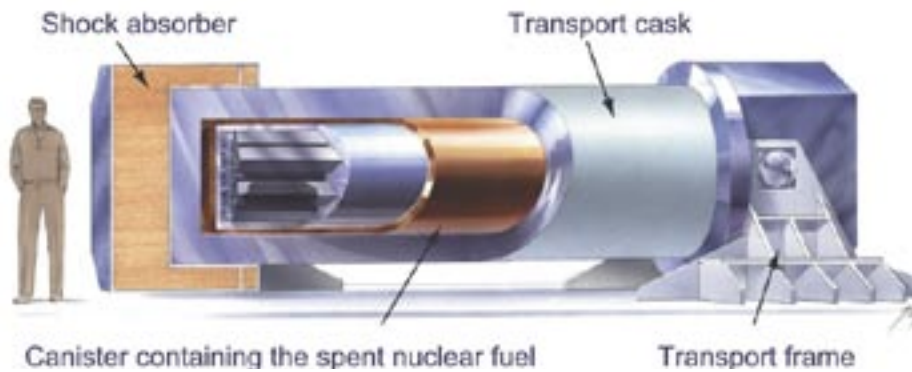


Figure 2-3. Canister shielded within the transportation cask /Pettersson, 2002/.

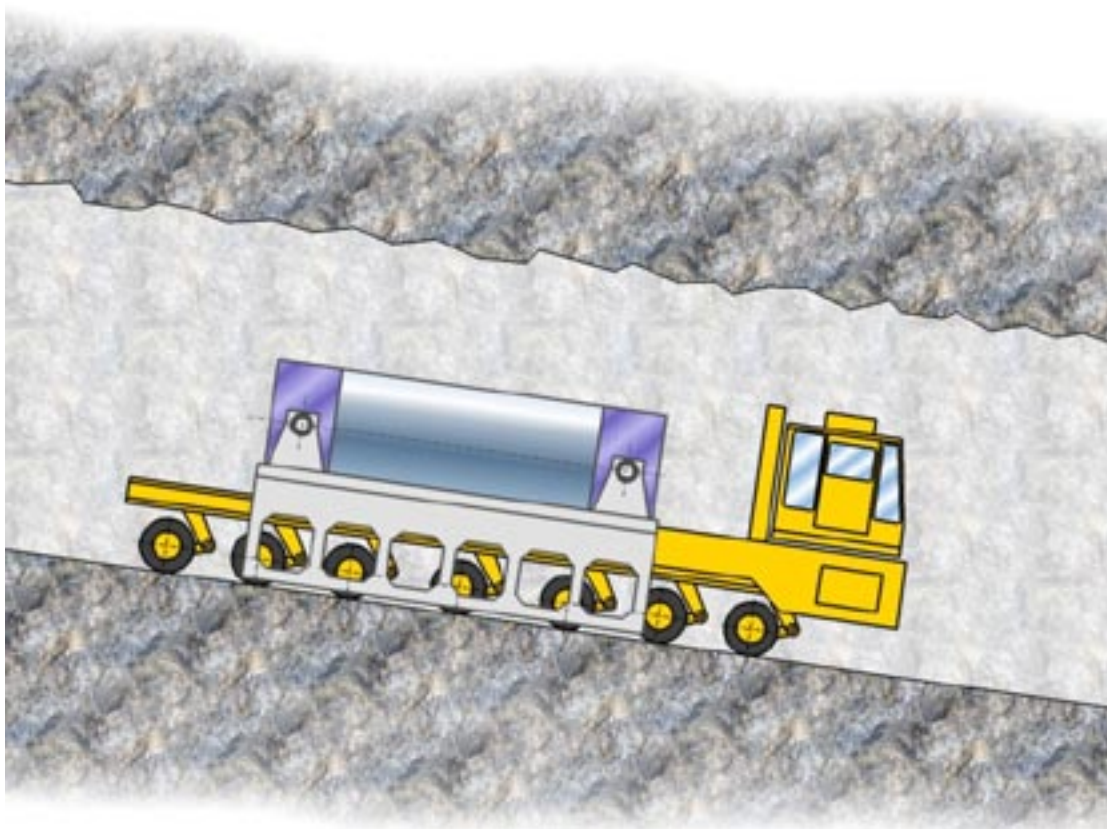


Figure 2-4. Transportation vehicle in the access ramp with the canister shielded within the transportation cask.

- **Central area:** A number of caverns are constructed to contain the technical systems needed to operate the repository, Figure 2-5, Figure 2-6.
- **Tunnels:** Several types of tunnels are defined in the reference design, Figure 2-6. The most important are the *deposition tunnels*, (“Type D”), where deposition holes are excavated before the buffer and canisters are deposited. The length of the deposition tunnels are in total around 32,000 m, with an area of 25 m². The cross section area is determined by the size of the deposition machine, Figure 2-7. *Main tunnels* (“Type A”, in yellow) will be constructed perpendicular to and prior to construction of the deposition tunnels. The cross section area of the tunnel is 64 m². The area is decided so the deposition machine can be moved from one deposition tunnel to the next.

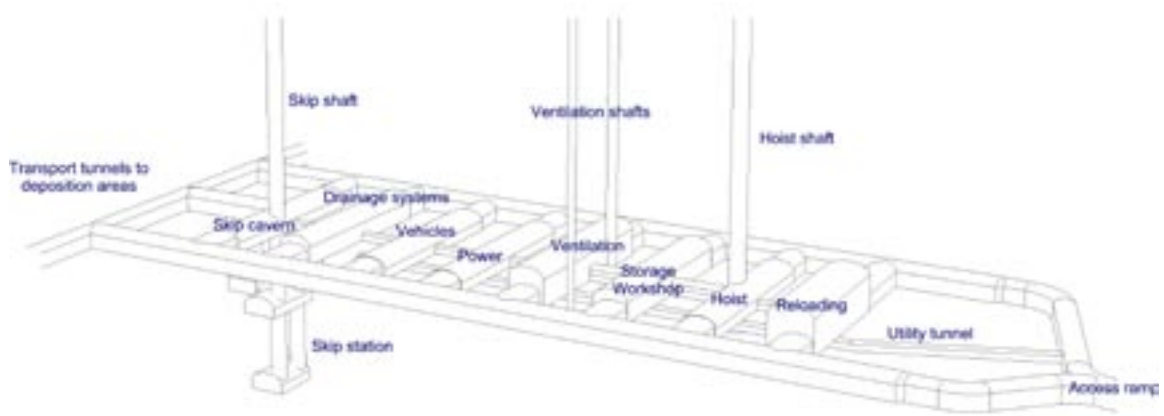


Figure 2-5. Overview of the Central Area.

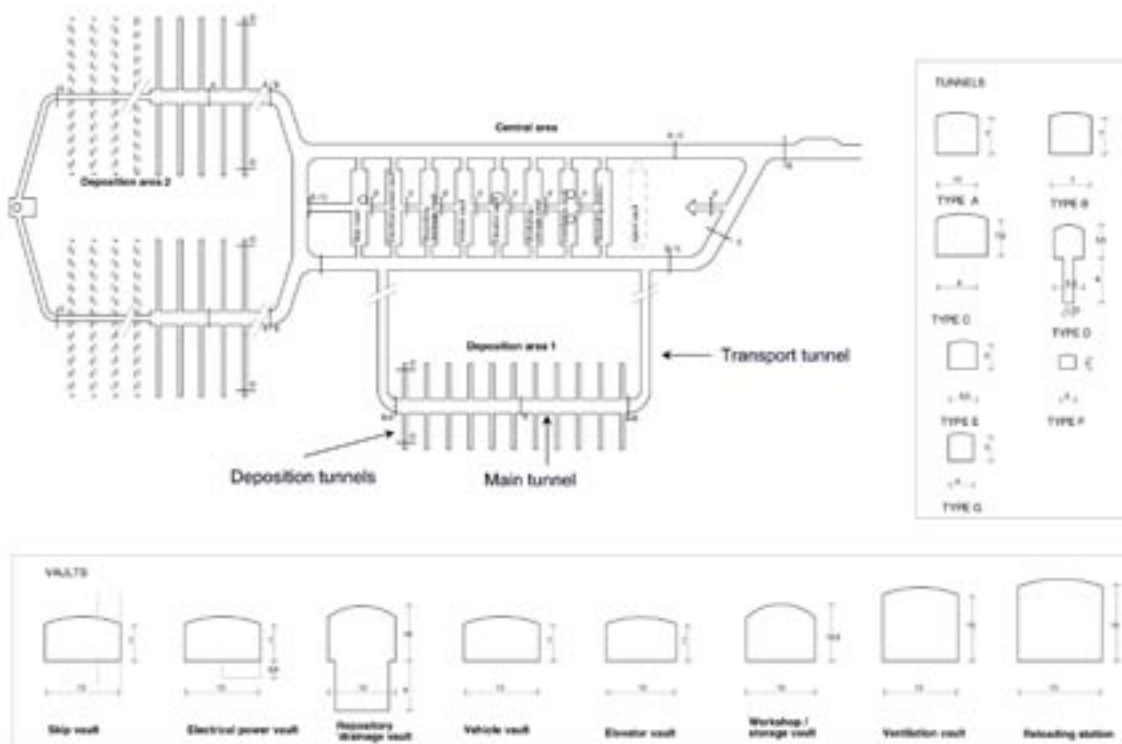


Figure 2-6. Overview of the different types of tunnels.

The overall length of main tunnels is around 2,500 m. *Transport tunnels* are tunnels outside deposition areas (“Type B” and “Type C”, blue) and typically with the same cross section area as the access ramp (46 m²). Total length of transportation tunnels is at least 2,200 m. *Pilot tunnels* (“*investigation tunnels*”) will often be constructed prior to the main tunnels. These are smaller (“Type G”), around 20 m² and will later be increased in cross section area to the area of the main tunnels.

- **Deposition holes:** These are drilled from the deposition tunnel floor. Each of the 4,500 holes is 1.75 m in diameter and 8 m in depth.

Backfilling of the deposition tunnels are following the deposition of the canisters, Figure 2-8.

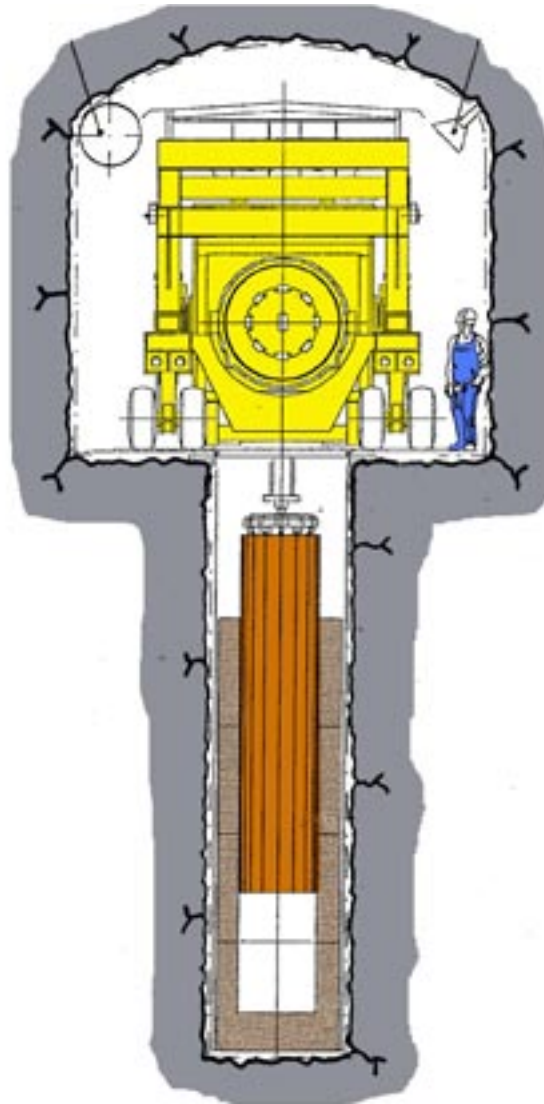


Figure 2-7. Sketch of the deposition machine, deposition tunnel and the deposition hole. The buffer (bentonite blocks) are pre-placed before the canister is deposited /Pettersson, 2002/.

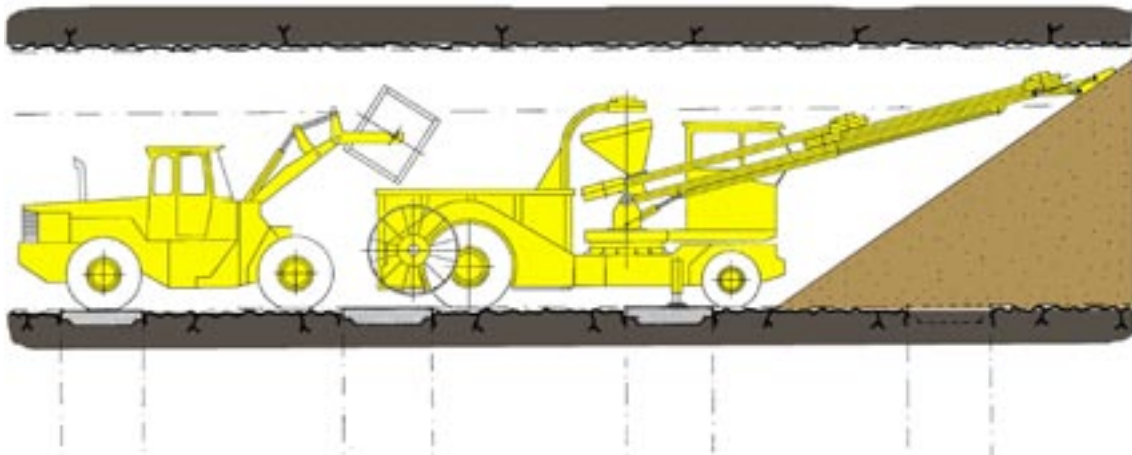


Figure 2-8. After deposition of the canisters, the deposition tunnels are backfilled as soon as possible using a mixture of rock and clay.

2.2 The implementation plan

The implementation of the deep repository is executed in phases, see Figure 2-9. The site investigations provide the general setting that is used to make the site-adapted layout for the repository. During the construction phase detailed site characterisation continues in conjunction with construction of the access to the deposition area, construction of parts of the deposition area and the central service area, Figure 2-2. During the initial operation phase, around 200–400 canisters of spent fuel are emplaced and deposition tunnels backfilled with a clay/crushed rock mixture. The initial operation phase is followed by the phase of regular operation phase where detailed characterisation, construction of the repository and waste emplacement is concurrent activities. The closure of the repository will take place when all spent fuel has been emplaced, i.e. in the latter part of this century.

The development sequence for the repository is shown in Figure 2-10. The numbering below refers to the picture:

Construction phase

- a) After receiving permits for the repository, temporary infrastructure is arranged at the surface. The skip shaft is sunk in parallel with the excavation of the access ramp. The excavation of the central area will start when the skip shaft is completed so the muck is hoisted in the skip. The reason for the shaft sinking of the skip shaft is elaborated in /Bäckblom et al. 2003/. At a ramp depth of around 200 m, preparations are under way to excavate ventilation shafts. Detailed investigations of the rock and monitoring, /see Bäckblom and Almén, 2004/, are integrated activities in this and subsequent phases.
- b) After around four years of construction, the skip is operating so the muck from excavation of the central area is transported to the surface through the skip shaft while excavation of the ramp still is ongoing. The upper ventilation shafts are already excavated. The maximum excavated volume of rock during this stage is up to 225,000 m³ of solid rock per year.
- c) The access ramp has reached the deposition level. Transport and main tunnels are under development and installations of technical systems in the central area are under way.

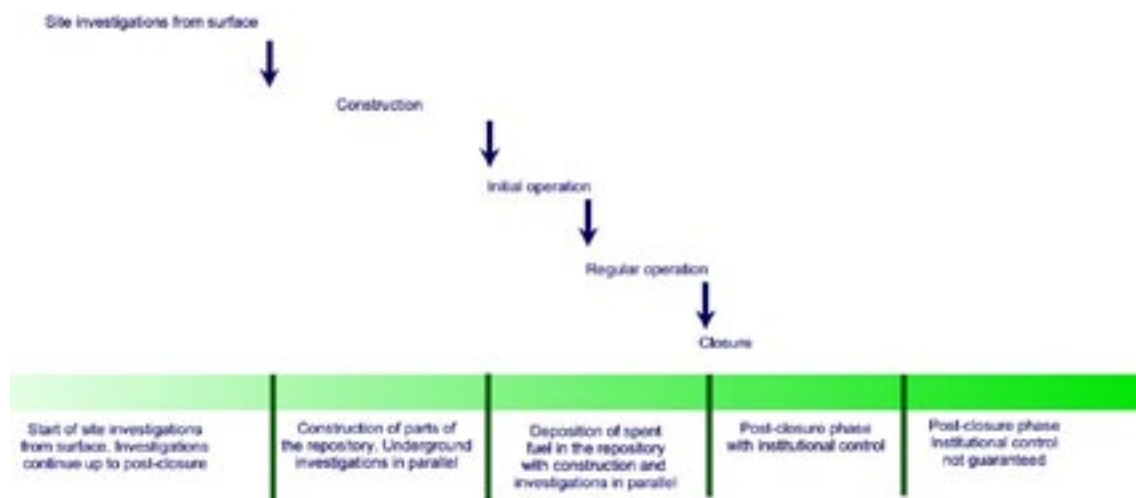


Figure 2-9. Outline of phases in the step-wise implementation of the Swedish deep geological repository for spent fuel.

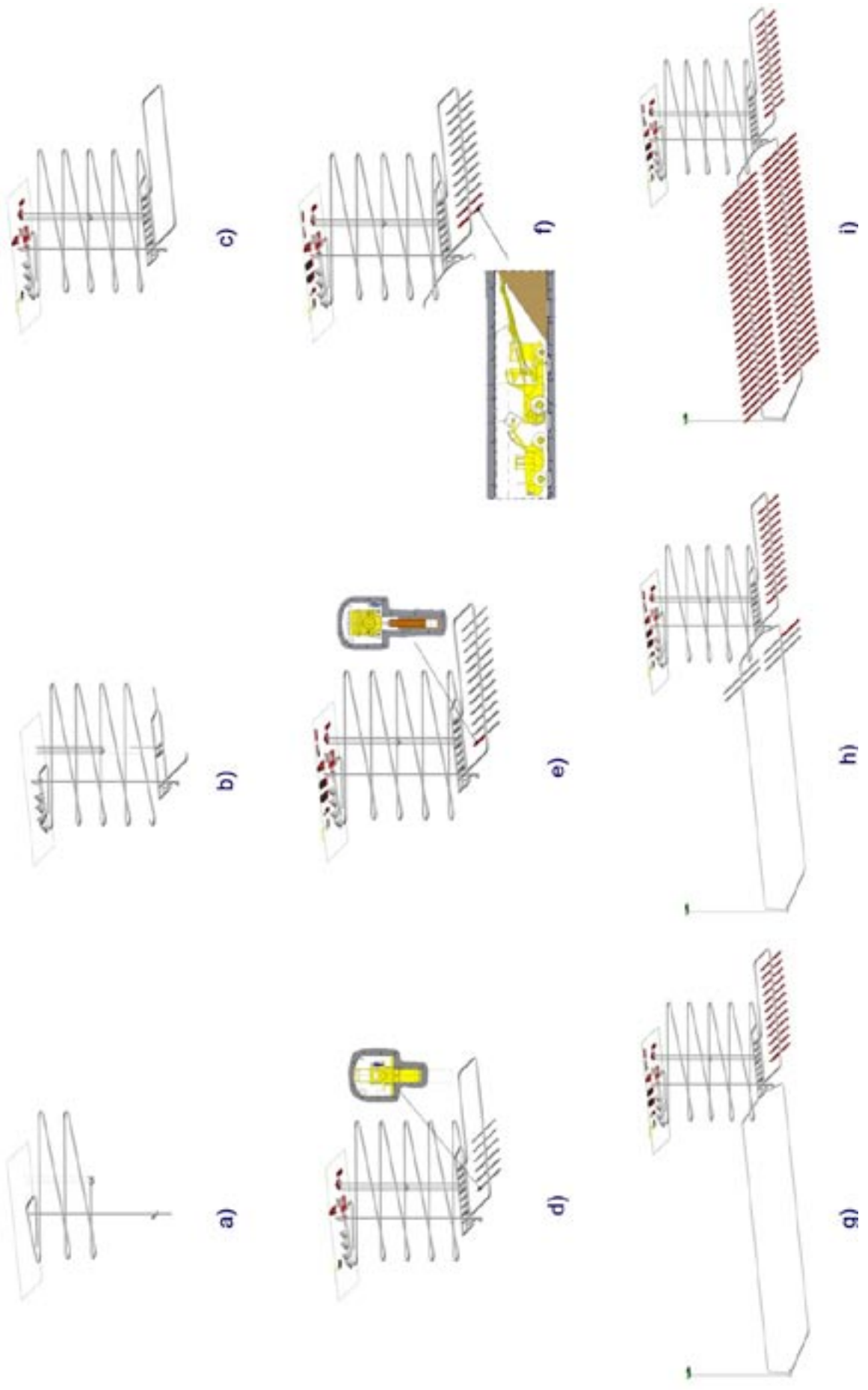


Figure 2-10. Development plan for the repository. For detailed descriptions of each inset, see the text.

- d) A number of deposition tunnels and deposition holes are excavated for the initial operation of the repository. Provisions are in progress to prepare for the initial operation when 200–400 canisters are to be deposited.

Initial operation phase

- e) After receiving the permit to operate the repository, deposition of canisters is initiated. Detailed characterisation of the site and the deposition area are integrated activities with the deposition work and construction work for the deposition areas for regular operation.
- f) Backfilling of deposition tunnels and construction of concrete plugs in between the deposition tunnels and the main tunnels are sequenced in parallel with the deposition works.
- g) During the initial operation phase pilot tunnels are constructed in the area for the regular operation. Additional ventilation shaft(s) are constructed for outlet air.

Regular operation

- h) After receiving the permit for regular operation, deposition of canisters of spent fuel commences with stepwise construction and continued detailed investigations of the site. Operation is split in a “Construction Area” (upper part of the area for regular operation, see Figure 2-10) and a “Deposition Area” (lower part of the area for regular operation) separated at least by 80 m in distance between already backfilled tunnels and tunnels to be excavated. The construction and deposition areas are swapped at regular intervals.

Closure

- i) When all spent fuel has been deposited and after receiving the permit to close the repository all underground installations are removed. The underground openings being backfilled in a similar fashion as the backfilling of the deposition tunnels.

2.3 Overview of technical options for excavation

This report mainly deals with three main principles for rock excavation that are outlined in the following:

- Drill & Blast
- Mechanical excavation
- Percussion drilling

2.3.1 Drill & Blast

Conventional blasting operations include (1) drilling holes in a converging pattern, (2) placing an explosive and detonator in each hole, (3) igniting or detonating the charge, and (4) clearing away the broken material, the muck.

The most common explosives used in standard blasting are e.g. dynamite (nitroglycerine mixed with kieselguhr) or ANFO (around 94% ammonium nitrate and 6% fuel-oil mixture) or emulsion explosives. The detonator is a device that initiates the detonation of a charge of a high explosive by subjecting it to percussion by a shock wave. Holes are so placed as to require a minimum quantity of explosive per volume of rock broken. Most blast-hole patterns are based on the fact that fragmentation is most uniform if the exploding charge

is within a particular distance from an exposed free face of the rock. To break up a large body of rock, charges are placed in series of holes drilled so that, as the holes nearest the exposed surface are fired, the blasts create new exposed faces at the proper distances from the next set of holes, in which firing of the charges is slightly delayed. The holes are fired in a predetermined order, at intervals of only thousandths of a second to tenths of seconds. In a tunnel round, the firing sequence starts with blasting of the cut area. The cut area consists of a number of parallel larger holes that are not to be charged, together with holes that are to be charged or holes drilled to form a wedge. Around the cut area there are slashing holes that are fired later in a sequence. The last holes in the sequence are the contour holes. Normally, a stronger charge is first applied in the bottom of all holes followed by a weaker, the pipe charge. The weaker charges are used to obtain smooth walls and roofs, although the floor holes often carry a higher charge compared to the wall and roof holes.

In the first stage the detonating explosive crushes the rock in the proximity of the hole-wall due to high detonation pressure. In the second stage, compressive stress waves created by the blasting propagate in all directions with a velocity equal to the sonic wave velocity of the rock-material. When the compressive stress waves are reflected towards a free rock face, they return as tensile stress waves. This causes tensile stresses in the rock that will fail if the energy in the shock wave is large enough. The energy that is released from the detonating charge along with the distance between the hole/row and the free face has to satisfy a defined relation in order to generate the failure of the rock. In the third stage, the high pressure gas (around 1 m³/kg of explosives) from the detonation penetrates the cracks that are the consequence of the failure in the previous stage and widens them. The rock mass between the hole/row and the free face will then yield and be thrown forward by the gas pressure.

An example is shown in Figure 2-11 where around 80 parallel drill holes with depth 5 m are used. Blasting starts at the “cut” where here three open boreholes are used as the initial opening, which is successively enlarged when explosives are detonated in the sequence (1–22) as shown in the picture.

Conventional blasting is carried out in a cycle of drilling, charging, blasting, ventilating fumes, removing muck, scaling to remove loose rock before rock support and surveying for the next round of blasting, Figure 2-12. Since only one of these five operations can be conducted at a time in the confined space at the heading, concentrated efforts to improve each have resulted in rate of advance to a range of 5–15 m per day.

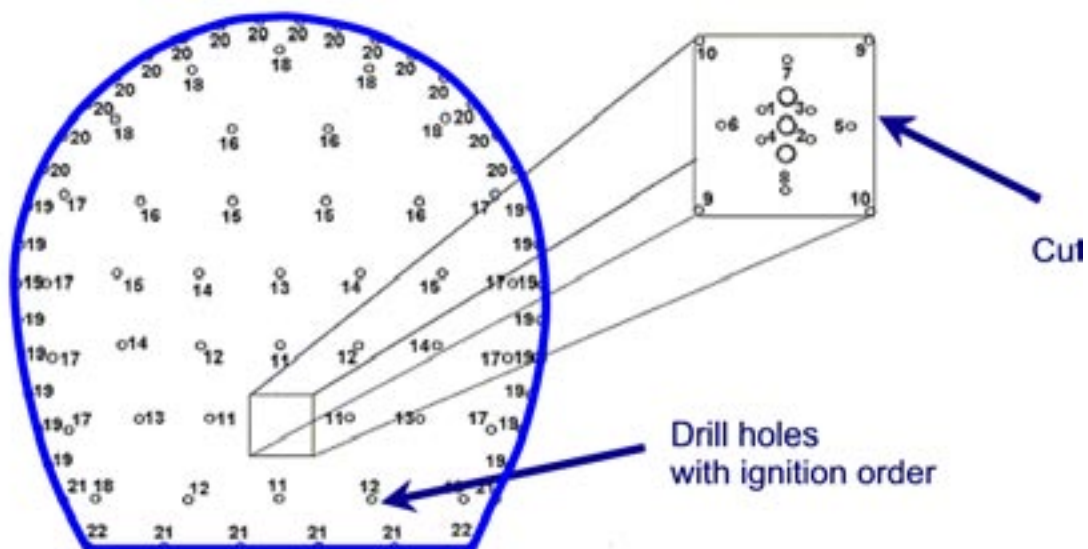


Figure 2-11. Example of blast design for a tunnel.

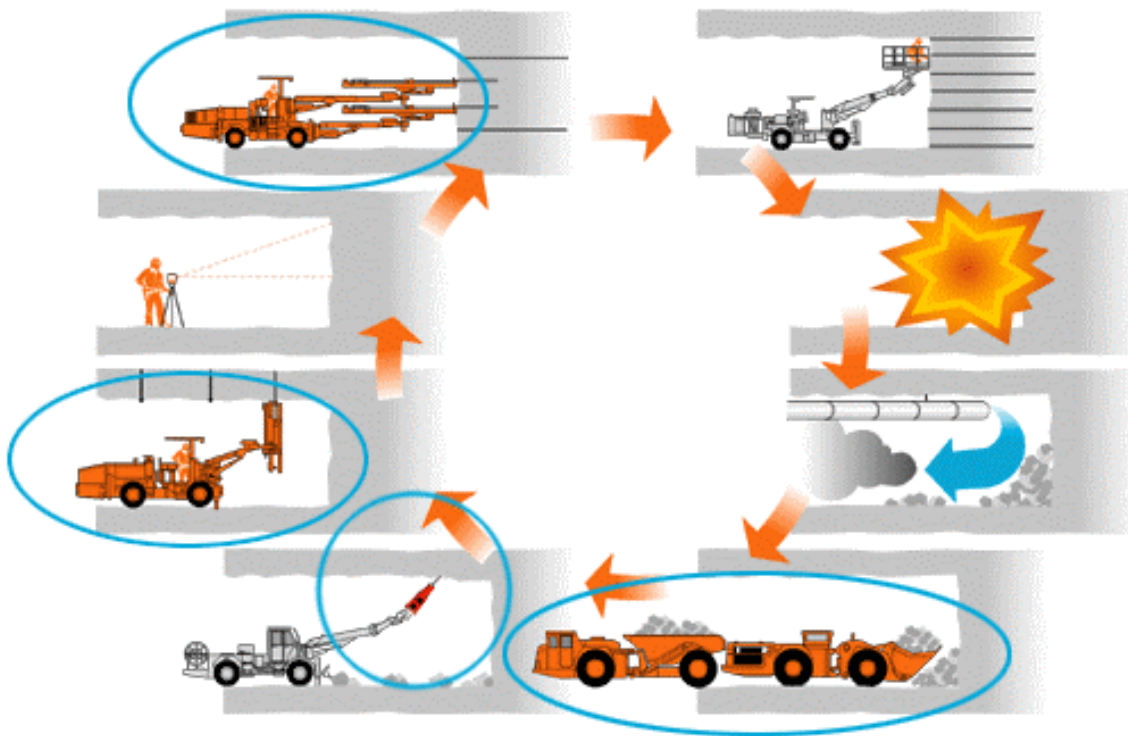


Figure 2-12. Drill & Blast operating cycle. Courtesy Sandvik Tamrock.

2.3.2 Mechanical excavation

The principle for mechanical excavation for the alternatives studied in this report is explained in Figure 2-13.

High forces are applied to tools (cutters) that are rolling over and over the rock surface to produce a kerf (groove) some cm deep. The forces and cracks developed interact with the natural fractures to produce rock chips. Two types of cutters for two different applications are shown in Figure 2-14 and Figure 2-16 respectively. The former figure shows a picture from horizontal tunnelling using a Tunnel Boring Machine (more details are found in Section 4.3). A thrust force of around 25–30 tonnes per cutter is applied to the tunnel face while the full face cutter head is rotating. The distance between the kerfs is around 75 to 90 mm for cutters 17 to 18 inches. Advance rate is 1–3 m/h dependent on machine and rock conditions.

A special case of the TBM is the Mobile Miner, where only one row of cutters is mounted on a beam that may move in two directions, Figure 2-15.

A Mobile Miner was for example used to construct a 70 m² tunnel in granitic rock in Japan /Tamura et al. 1997/. A standard TBM will obviously produce a circular opening, but the Mobile Miner in principle would produce any tunnel shape. The main disadvantage is the low advance rate as only very few cutters simultaneously engage the rock with full penetration only at the “springline” compared to the TBM that will have many cutters at the face at the same time. The actual advance rate for the Japanese tunnel (Uniaxial Compressive Strength around 150 MPa) was around 0.2–0.3 m per machine hour, only 1/10 of the advance rate for a TBM.

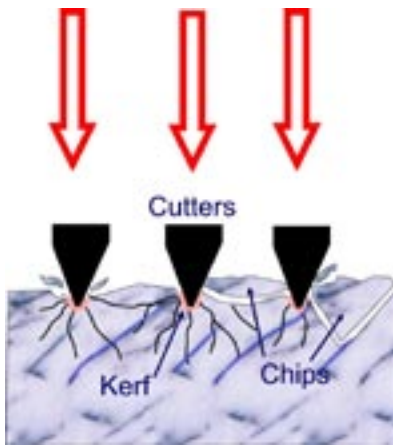


Figure 2-13. Principles for mechanical excavation.



Figure 2-14. Kerfs (grooves) produced at a tunnel face by using disc cutters in a Tunnel Boring Machine (TBM) operation. Photos: Courtesy The Robbins Company.



Figure 2-15. Picture of Mobile Miner. Courtesy Atlas Copco.

An application for vertical mechanical excavation is shown in Figure 2-16. Here the cutter is not a steel disc but hard metal buttons. The Figure 2-17 shows how vertical shafts (raises) can be constructed; a pilot hole is drilled from the surface down to an underground opening (1). A reamer is attached to the drill string (2) and the equipment at the surface generates the thrust and torque needed to produce the grooves and the chips for excavation of the vertical raise (3).

Similar technology can be used for horizontal tunnels as well. The horizontal reaming is similar to vertical reaming. A variant of horizontal reaming is push-reaming, where the reamer is pushed instead and the drill string is supported by stabilizers, Figure 2-18. The advantage is that there is no need for an extra service tunnel behind to mount the reamer on the drill string. Advance rate is however lower than for a TBM as less thrust and torque can be applied at the cutters through the relatively small diameter drill string.

So far we have discussed mechanical excavation where kerfs are manufactured. There are also other principles for mechanical excavation used for example in a roadheader, where “pick cutting” is used. The rotating head contains hard metal bits, Figure 2-19, that rip rock pieces from the tunnel face. Materials technology is steadily improving, but excavation of hard, granitic rock is still not feasible. The roadheader technology is used for softer rock, for sandstone etc. where restrictions are posed by the picking tool resistance against shock load at extremely high rock strength (resulting in breakage of the tungsten-carbide tips) or their resistance against abrasive wear when encountering rock with high content of hard minerals.

2.3.3 Percussion drilling

The percussion drill bit is literally hammered into the rock, forcing it to smash to pieces, Figure 2-20. The rock fragments are removed up the side of the drill string by high pressure compressed air or by water. Percussion drilling is the conventional technique for shallow water wells and also for the boring the drill holes in the Drill & Blast operation. The Figure 2-20 shows Down-The-Hole equipment where the hammer (piston) also is down in the hole. A variant of percussion drilling technique would be possible for horizontal deposition drifts, the cluster drilling (see Section 4.6.4). Several percussion drills are put together in a frame that slowly rotates, Figure 2-21.

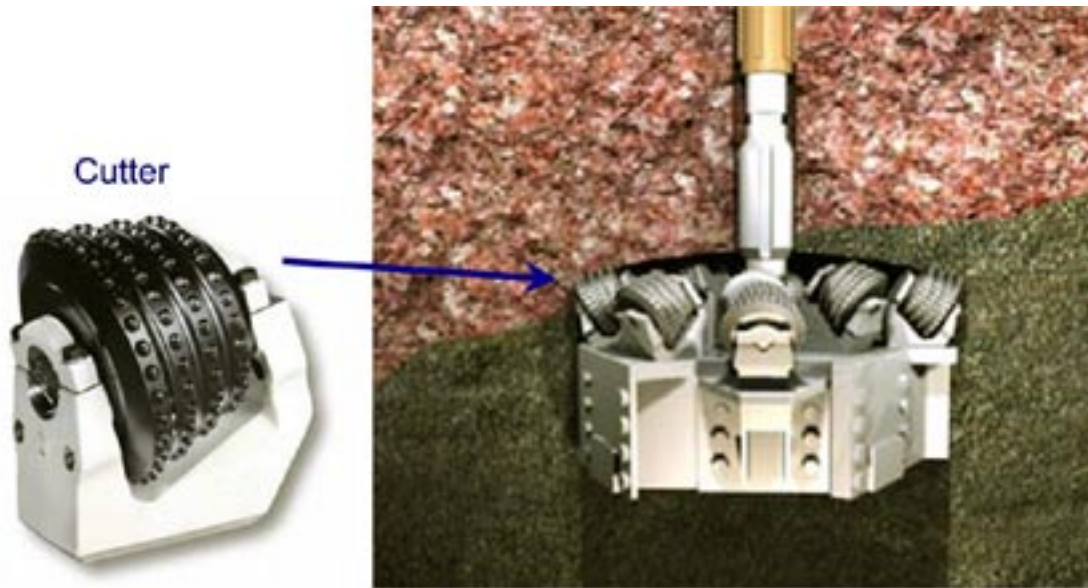


Figure 2-16. Cutter with hard metal bits (left). Sketch of a raise-drilling operation. Photos: Courtesy Atlas Copco Construction & Mining.

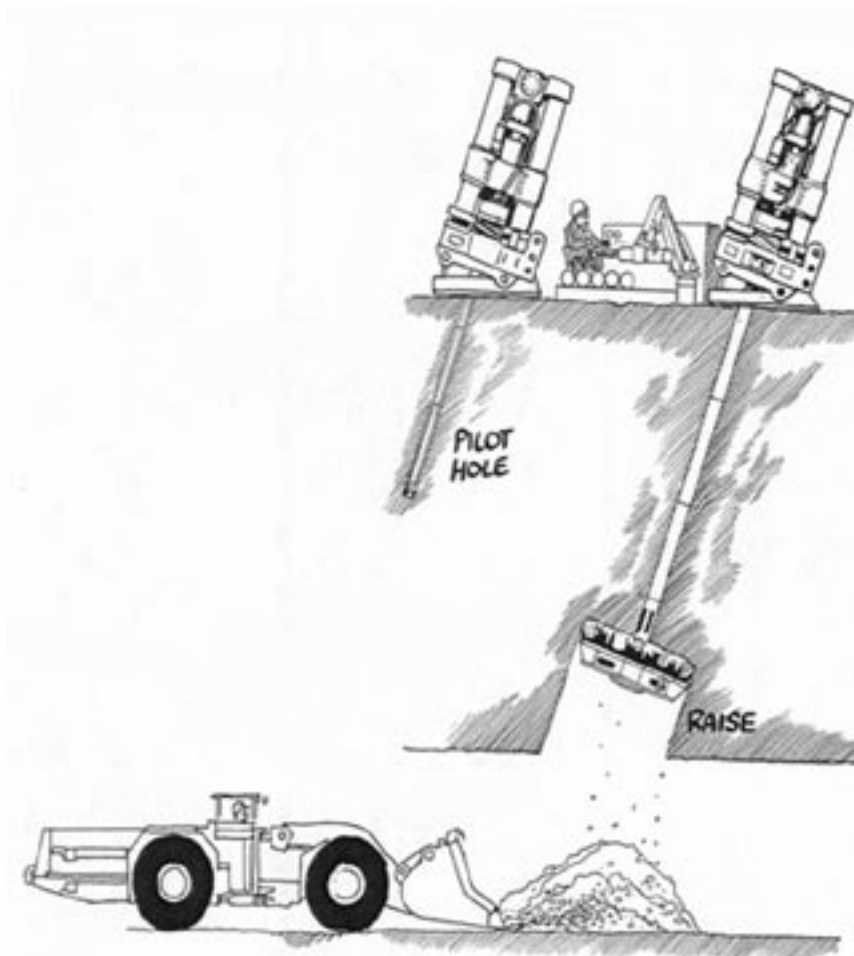


Figure 2-17. Principle of mechanical excavation for vertical shafts. Courtesy Atlas Copco Construction & Mining.

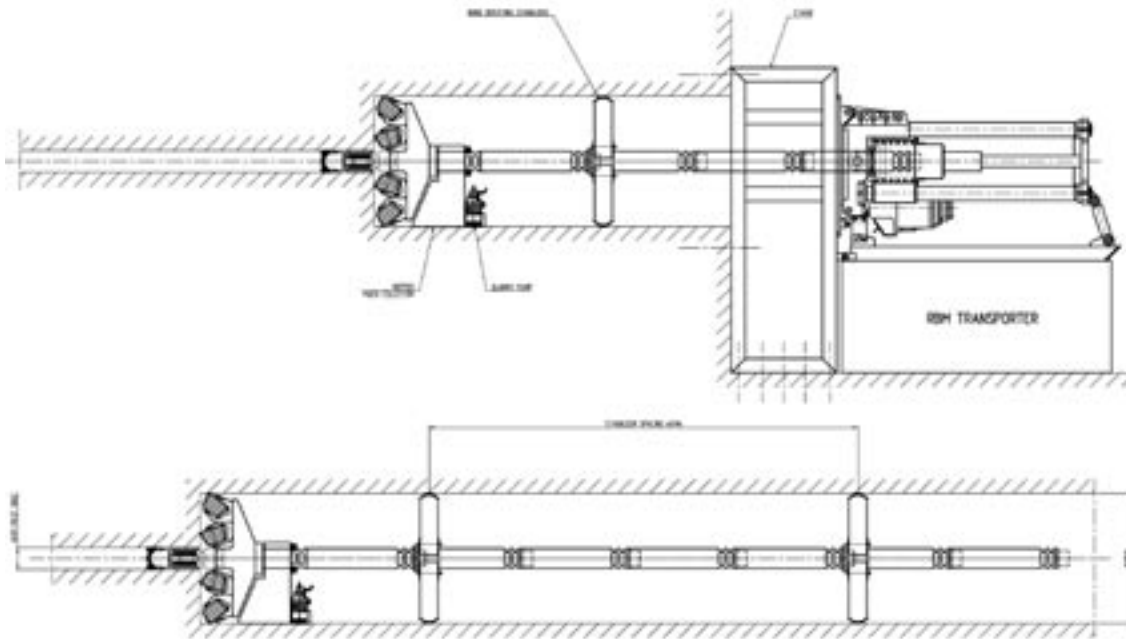


Figure 2-18. Sketch of Raise-Boring Machine for horizontal push-reaming. Courtesy Atlas Copco Construction & Mining.



Figure 2-19. Example on equipment for mechanical excavation – roadheader. Courtesy Sandvik Voest Alpine.

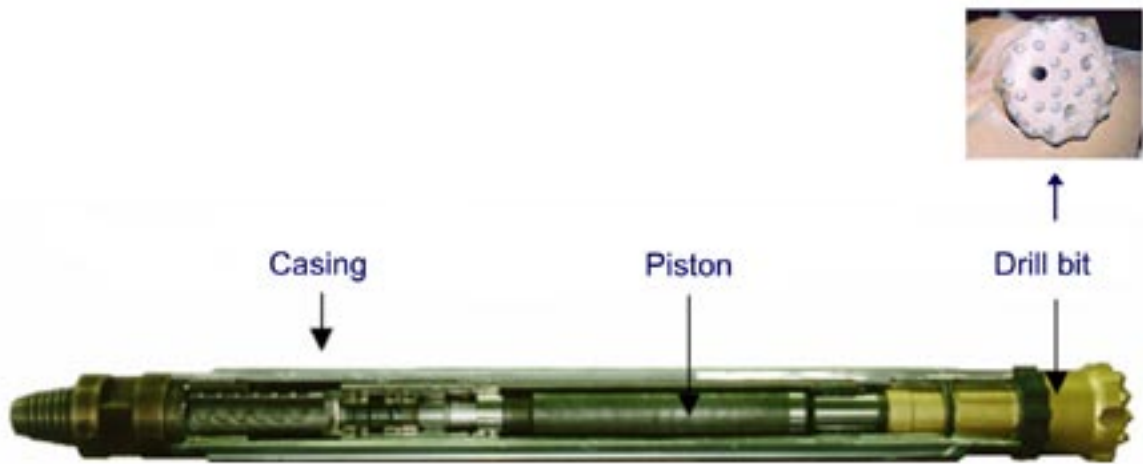


Figure 2-20. Example on design of percussion drill. Photo: Courtesy Wassara AB.

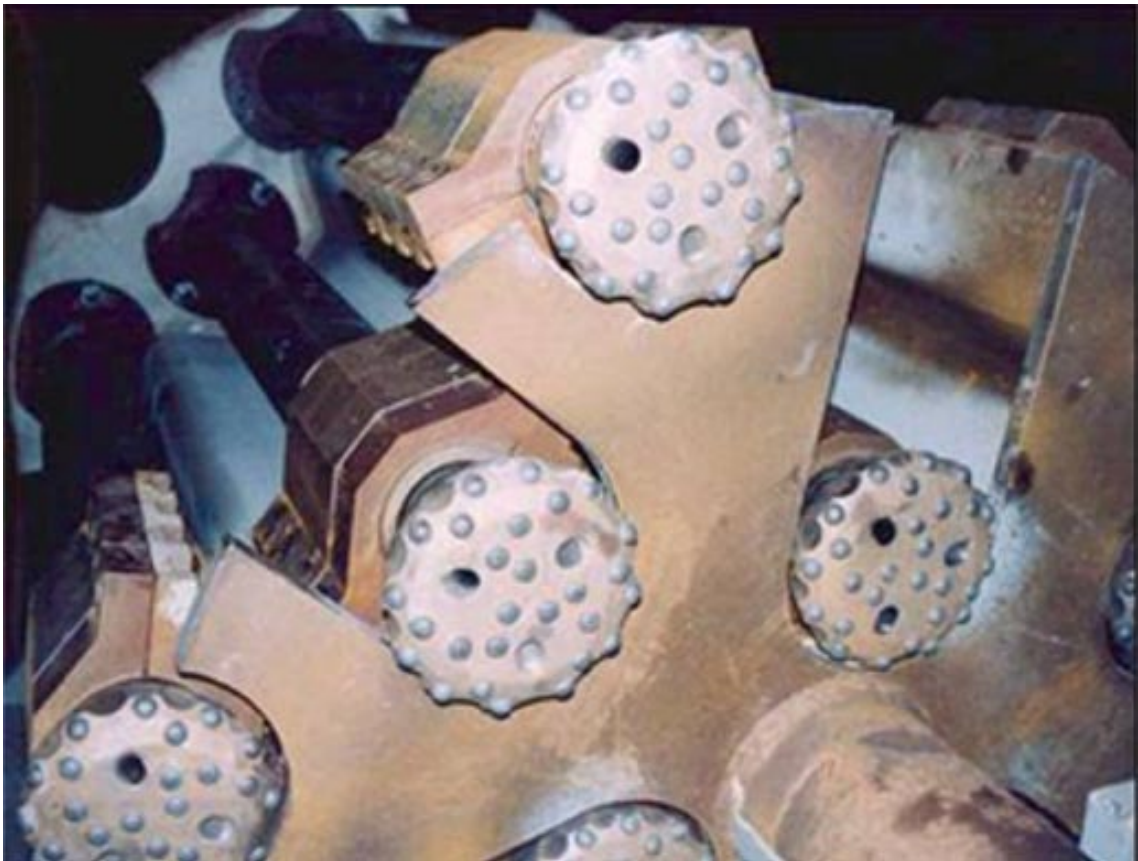


Figure 2-21. Several drills are combined in a rotating framework for excavation of horizontal drifts. Photo: Courtesy Wassara AB.

2.4 Selection of main alternatives for this study

From the previous description we understand that the repository consists of openings from around 3 m² up to over 200 m² and it is evident that all excavation technologies will not cover all applications from a feasibility point of view. An outline of type of underground openings and excavation methods is shown in Table 2-1 with rationales following for selection of main alternatives for this study.

Access ramp

The main alternative in the reference design is Drill & Blast, but the TBM may be a feasible alternative in special conditions where there are good rock conditions, the curve radius not too small (> 200 m) and the more expensive TBM alternative can pay off by shorter construction time. Both alternatives are explored in this study.

Central area

The reference design is Drill & Blast. Mobile Miners could be used in theory, but due to the low efficiency, costs would be very high without any special benefit. Multiple-entry TBMs of design similar as for deposition tunnels could also in theory be used, but Drill & Blast would anyhow be used to create the shape needed for the installations

Shafts

Shafts can be constructed from surface by Drill & Blast and this is the method chosen for the skip shaft in the SKB reference design. The shaft option is selected to shorten the overall duration of construction. Shaft Boring Machines of different types have been built and used, but very often failed due to problems to lift the muck to the surface. The most common way to excavate vertical shafts is by raise-boring where the reamer is mounted from an underground opening and this is the preferred method by SKB when applicable.

Main, transport and pilot tunnels

The reference design is by Drill & Blast. We here also study an alternative using TBM for pilot tunnels where a small machine (diameter 5 m) is used for rapid tunnelling. The pilot tunnels are later slashed into the size of main tunnel. In case the access ramp is excavated by TBM, some main tunnels (“Type B”) may be excavated by the same TBM (diameter 7.1 m) or by the pilot tunnel TBM.

Deposition tunnels

The deposition tunnels are the main concern for this study. Drill & Blast is the reference design, but we have also studied alternatives. Robbins Company was commissioned to prepare a conceptual design of a multiple-entry TBM and compile information for comparison. Atlas Copco has prepared design of a Raise-Boring Machine (RBM) and compiled information of relevance for excavation of deposition tunnels using RBM.

Table 2-1. Alternative excavation methods for the facility units • = alternatives evaluated. – = alternatives not deemed to be feasible. N.A. Not applicable.

Excavation methods	Access ramp	Central service area	Shafts	Pilot-, transport- and main tunnels	KBS-3V Deposition tunnels	Deposition holes	KBS-3H Deposition drifts
Drill & Blast	•	•	•	•	•	–	–
Tunnel Boring Machine (TBM)	•	–	N.A.	•	•	N.A.	Likely to be feasible, but not evaluated specifically in this report ¹
Vertical raise-boring (RBM)	N.A.	N.A.	•	N.A.	N.A.	N.A.	N.A.
Horizontal pull-reaming (RBM)	N.A.	N.A.	N.A.	N.A.	•	N.A.	•
Horizontal push-reaming (RBM)	N.A.	N.A.	N.A.	–	–	N.A.	•
Cluster drilling (Wassara)	N.A.	N.A.	–	–	–	–	•
Down-reaming (RBM)	N.A.	N.A.	–	N.A.	N.A.	•	N.A.
Shaft Boring Machine (SBM)	N.A.	N.A.	–	N.A.	N.A.	•	N.A.
Mobile Miner	–	–	N.A.	–	–	N.A.	–

¹ SKB in a previous study concluded that TBM using disc cutters is of less interest; wear of the cutter would mean frequent cutter changes to meet the geometrical specifications for diameter and surface roughness. However, a TBM with button bits gage cutters may be an attractive alternative for excavating horizontal deposition drifts, but this alternative was not explicitly studied.

Deposition holes

The reference design is based on mechanical excavation where certain geometrical tolerances are defined. Two techniques for excavation are described, either using a Shaft Boring Machine or a down-reaming RBM.

Deposition drifts – KBS-3H

KBS-3H with horizontal deposition of canisters is an alternative design under evaluation. Three excavation techniques are compared, the cluster technology using water-hammer percussion technology, (see Section 4.6.4) horizontal reaming using pull and horizontal reaming using push, the latter two using an RBM, see Section 4.6.2 and Section 4.6.3 respectively. Conclusions from a previous TBM study for deposition drifts it would be very difficult to obtain maximum ± 5 mm allowed deviation of diameter and 10 mm coneness of the entire length of the drift, based on using disc cutters. It might however be that the tolerances are complied with using button bits gage cutters. The overall system solution for the TBM would also benefit from a slightly larger diameter, 2.0 m or more.

2.5 Industrial references

Drill & Blast is a mature technology proven for a number of excavation applications and requirements. The specific challenge here is to excavate the deposition tunnels with minimum excavation damage; technology exists, but there is need to test the interaction human factor – organisation-technology to prove that excavation damage can be minimized in practice.

TBM's have been in extensive use for tunnelling in both soft rocks and hard rocks. The technology is mature; there are however very few, if any industrial references to multiple-entry short tunnels in hard rocks. The specific challenge for deposition tunnels is the practical handling of the set-up and move of the heavy and bulky equipment.

Mobil Miners are not so common and in hard rocks there are very few industrial references. Roadheader is a mature technology and in common use for soft rock excavation.

Excavation of vertical deposition holes has very few industrial references, besides the work at SKB and Posiva in Finland. To some extent the technology benefit from equipment developed for Shaft Boring Machines.

Construction of vertical shafts from ground surface using Drill & Blast (downwards) or raise-drilling (upwards) from below are both mature technologies.

Construction of horizontal drifts using horizontal pull-reaming is an emerging market. There is also more and more interest for horizontal push-reaming. There are no any industrial references for horizontal drifting for the very strict geometrical requirements imposed by SKB.

Cluster technology is used as standard practice in several mines, but often for sub-vertical holes with diameters less than 1 m.

3 Premises and methodology for the study

This chapter provides an overview of premises used for this study. SKB has in a previous report /SKB, 2002a/, compiled general design requirements for the repository encompassing international guidelines, legal requirements and stakeholder requirements etc. Particular assumptions for this study follows. Methodology for comparison and judgement of alternatives are similar as for a previous study to decide access routes for the repository /Bäckblom et al. 2003/. It is deemed that the methodology used for evaluation is compliant with the regulations to assess alternatives in a Best Available Technology (BAT) perspective.

3.1 General requirements

Essential information in /SKB, 2002a/ is here extracted for convenience and transparency. Other aspects are as well included to shed light on the following factors that influence this project:

- Long term safety.
- Repository design, repository construction and repository operation.
- Environmental impact.
- Sustainable management of natural resources.
- Costs.
- Schedules.
- Flexibility.
- Project risks.
- Research and Development.

3.1.1 Long term safety

Several laws and guidelines apply:

- Construction and operation of the repository should only provide limited effects on the safety functions of the repository.
- The amount of construction material (concrete, steel etc.) and other stray materials that will be left in the repository shall be estimated and be shown to have limited importance for the long time safety of the repository.

3.1.2 Repository layout, repository construction and repository operation

In general, *Best available technology* is to be used, /SSI, 1998/ i.e. “the most effective measure available to limit the release of radioactive substances and the harmful effects of the releases on human health and the environment, which does not entail unreasonable costs.”

Layout:

- Inclination of access ramps and tunnels shall be sufficient to allow for easy management of drainage water ($> 1:100$), but not more than ensuring safe and efficient transports (1:8–1:10).
- Radius of curves shall be large enough to allow for the vehicles to be used.
- The repository shall be designed considering that vertical or horizontal deposition of canisters may be selected.
- The rock facility shall be designed for the alternative designs of equipment for construction and operation that may be in place.
- The size of the deposition tunnels shall be adapted to the equipment and installations needed for ventilation, transport of excavated rock, investigations of the rock, excavation and preparation of the deposition hole, deposition of buffer and canisters and backfilling and closure.
- The underground rock facility shall be designed so that the host rock after closure provides a suitable and stable mechanical and chemical environment for the engineered barriers and so the rock retards transport of radionuclides to the biosphere.
- Design of the facility, equipment and preparation of routines shall be made so investigations, construction and operation can be executed in parallel with low probability for interruption in construction and operation.
- The respect distance from a canister position to a fracture zone shall sequentially be determined by rock mechanical analyses and based on available information of the host rock properties.
- Need for grouting and pumping of drainage water in deposition holes and deposition tunnels for safe deposition and backfilling shall be estimated in due consideration of requirements on salinity and impurities of the drainage water and need to emplace the backfill material in accordance with specifications. It shall be possible to open backfilled deposition tunnels both before as well as after saturation.

Construction and operation:

- Radiation doses in connection with the operation of the facility shall be limited as far as possible.
- Construction and operation of the repository should only provide limited effects on the safety functions of the repository.
- In connection with the construction, parameters having an effect on constructability and the long term safety functions of the repository should be measured and shown to comply with requirements on the deep repository and the functions of the rock.
- Excavation work or other work for the deposition tunnels and the deposition holes must be conducted so that areas where deposition is completed should neither compromise the canister, the buffer, the backfill nor the functions of the near-field rock.
- Fracture apertures in the deposition holes must not allow for erosion of the buffer or for colloid release. Acceptable channel widths are 0.1–0.5 mm.
- Deviations in the dimensions of the deposition holes must not cause density variations in the buffer creating uneven swelling pressure and subsequent uneven loading of the canister.

3.1.3 Environmental impact

General laws and regulations are applicable. In addition:

- The repository shall be designed so that it provides adequate radiation protection and so that present and future environmental impact is minimised.
- Lowering of the groundwater table during construction and operation of the rock facility must not cause unacceptable consequences for the ecosystem or the local groundwater supply.
- The rock facility shall be designed so that future generations' use of the repository site not is unnecessarily restricted.

3.1.4 Sustainable management of natural resources

General laws and regulation are applicable. A specific requirement is that:

- Consumption of material, raw material and energy shall be as low as possible with regards to what can be deemed necessary for an adequate radiation protection of humans and the environment.

3.1.5 Costs

- The repository shall be safely and efficiently designed, constructed and operated.
- The rock facility shall be designed, constructed, operated and closed in accordance with specifications, in the pace that is required and to a reasonable cost.

3.1.6 Schedules

- The repository shall be designed in consideration that the construction phase or operational phase may be longer than the present planning.
- Design, construction and operation of the facility shall be carried through so that one canister per working day can be deposited in the repository.
- Deposition tunnels and deposition holes should be kept open so short time as possible in consideration of construction and operation. With present knowledge, it should be less than 5 years. Other underground openings may be open 50–100 years.

3.1.7 Flexibility

- The rock facility shall be designed for the alternative designs of equipment for construction and operation that may be in place.
- The rock facility shall be designed in consideration of that deposition areas may be extended and that area for deposition of low- and medium level long-lived waste may be needed.

3.1.8 Project risks

- Events or conditions that may have effects on barriers shall be identified.

3.1.9 Research and development

- Design principles and design solutions shall be tested under conditions corresponding to those that can occur during the intended application in a facility. If this is not possible or reasonable, they must have been subjected to the necessary testing or evaluation with reference to safety.
- The repository shall be designed considering that presently known, but untried and alternative technology maybe used in the future.

3.2 Specific conditions and assumptions for this study

This chapter describes some specific, detailed assumptions and requirements that are used as input for technical descriptions of rock excavation methods and an optimised selection of excavation methods.

3.2.1 Repository layout (deposition area)

Drill and blast

Layout has already been prepared for the reference design using Drill & Blast, see Figure 3-1.

TBM

For the TBM it is assumed that a tunnel radius of 150 m is needed for the pilot tunnels (Figure 3-3) and 200 m radius is needed for the access ramp. An example on a TBM layout for an access ramp with TBM is shown in Figure 3-2.

Tunnel dimension are shown in Table 3-1. The area of the deposition tunnels is derived from the deposition machine, width 4.1 m and height 4.6 m. For a circular TBM-tunnel, it is supposed that a free width of 4.3 m is needed so the minimum diameter of the circular tunnel is $(4.3^2 + 4.6^2)^{1/2} = 6.3$ m. It is here assumed that the 5.0 m or 6.3 m diameter machine may be used for the main tunnel with subsequent slashing to full area. The area stated for access ramp and transport tunnel is smaller than in the reference design as it is assumed that standard trucks are used rather than electric trucks.

Horizontal reaming

The layout when using horizontal pull-reaming is shown in Figure 3-4. It is assumed that the method can drill and ream up to around 400 m. An extra service tunnel is needed so the raise-boring machine can be moved from one deposition tunnel to the next deposition tunnel. The reamer is mounted on the drill string in the main tunnel and pulled to the service tunnel. It is also planned that a rock bulkhead is left at the service tunnel effectively eliminating the need for a second concrete bulkhead; only the pilot hole needs to be plugged. As shown in the Figure 3-4, the two main tunnels are needed to separate the construction area from the deposition area.

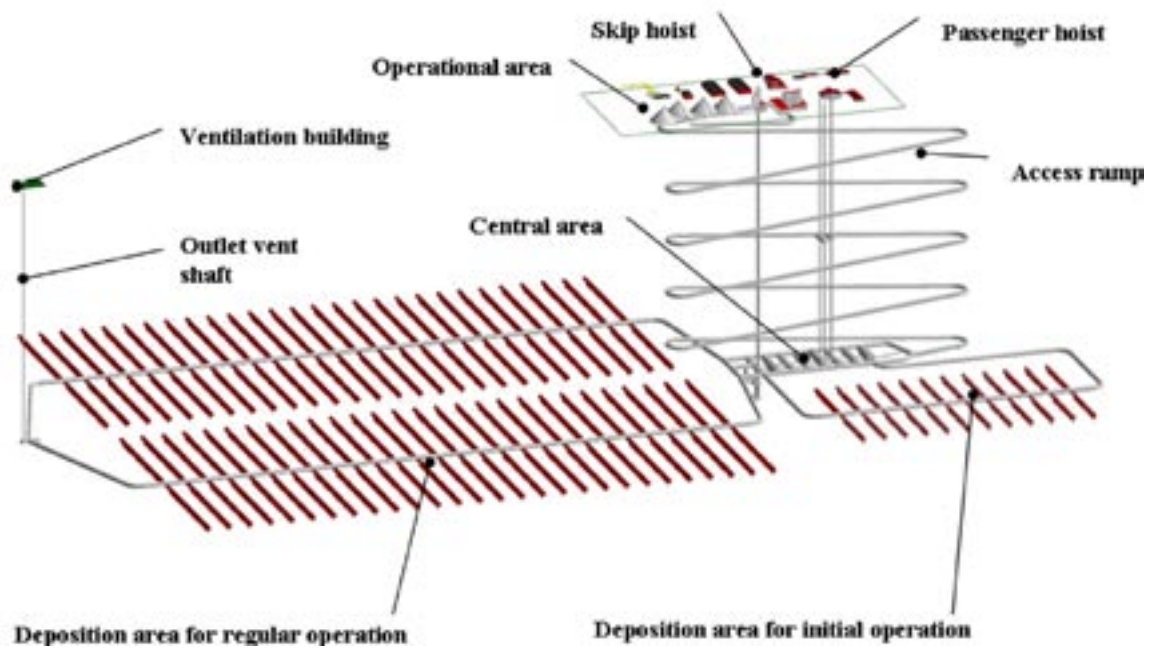


Figure 3-1. Outline of the KBS-3V layout.

Table 3-1. Tunnel dimensions for TBM-tunnels (in m and m²).

Tunnel	Needed free height	Needed free width	Need for Diameter	Area m ²
Access ramp	4.5	5.5	7.1	40
Pilot tunnel	5	4	5.0	20
Transport tunnel	4.5	5.5	7.1	40
Main tunnel	10	7	–	–
Deposition tunnel	4.6	4.3	6.3	31

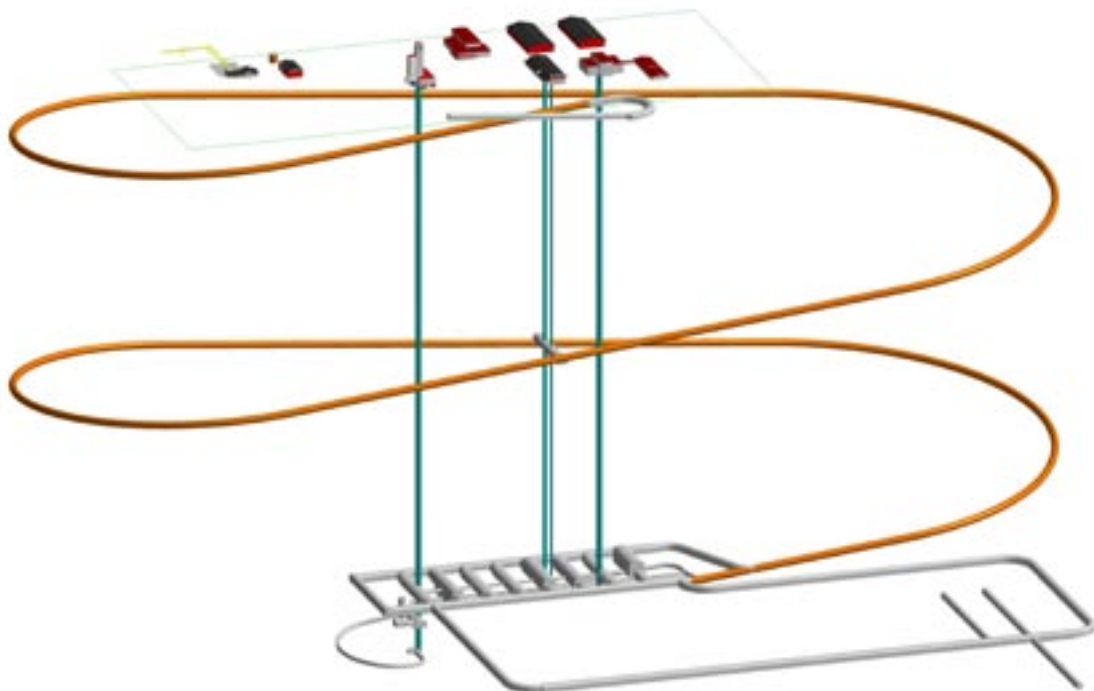


Figure 3-2. Generic layout of access ramp for a TBM option.

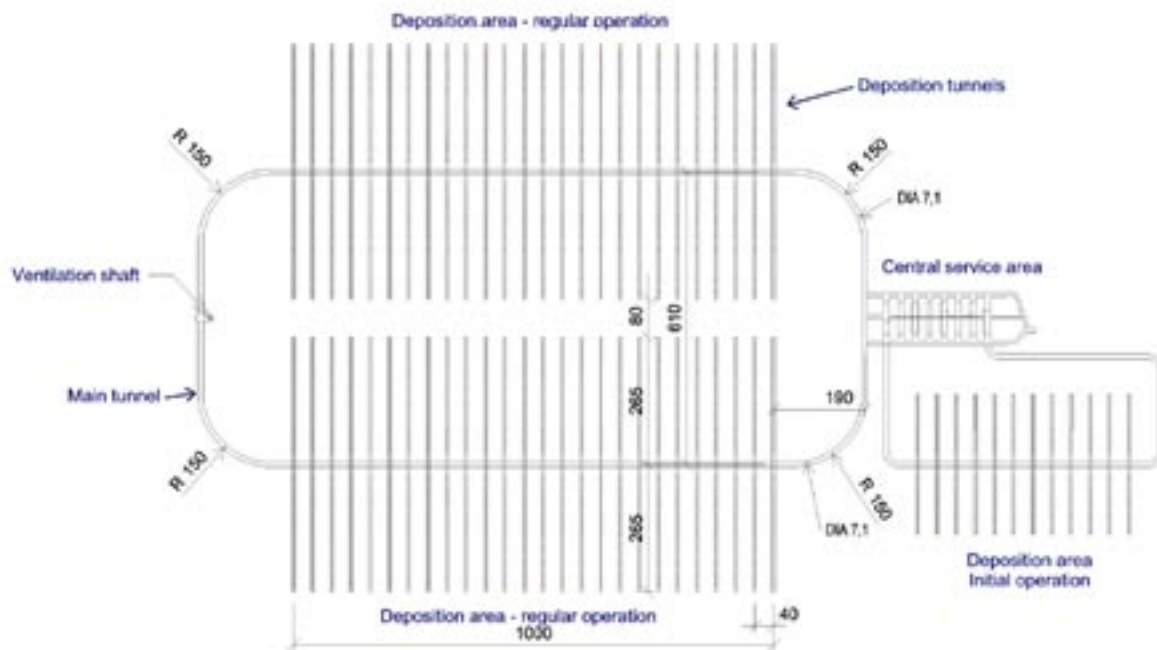


Figure 3-3. Layout for a R=150 m TBM.

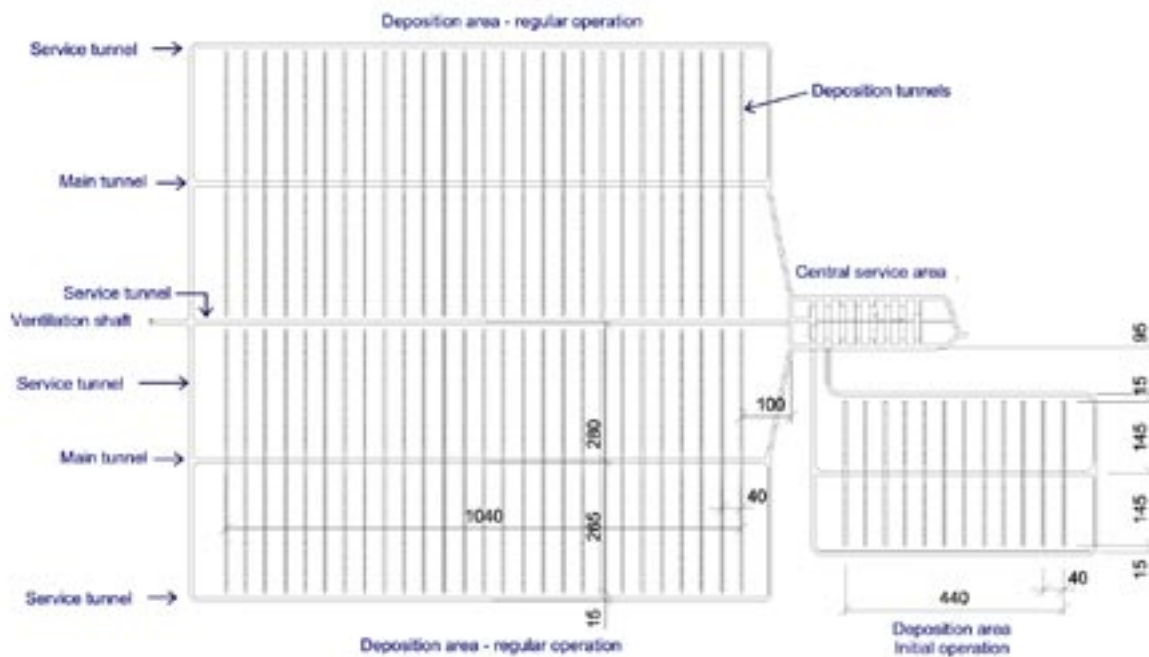


Figure 3-4. Layout for a repository with horizontal reaming.

3.2.2 Layout of the KBS-3H

The KBS-3 horizontal emplacement concept, KBS-3H, is a variant of the SKB's disposal method with the spent fuel positioned into a horizontal deposition drift rather than in vertical disposal borehole in the floor of the deposition tunnel. To differentiate the two disposal concepts with vertical or horizontal disposal they are called the KBS-3V or KBS-3H respectively. The layout for the KBS-3H reference case is similar to the KBS-3V, but deposition drift is smaller than the deposition tunnel and there are no deposition holes, Figure 3-5.

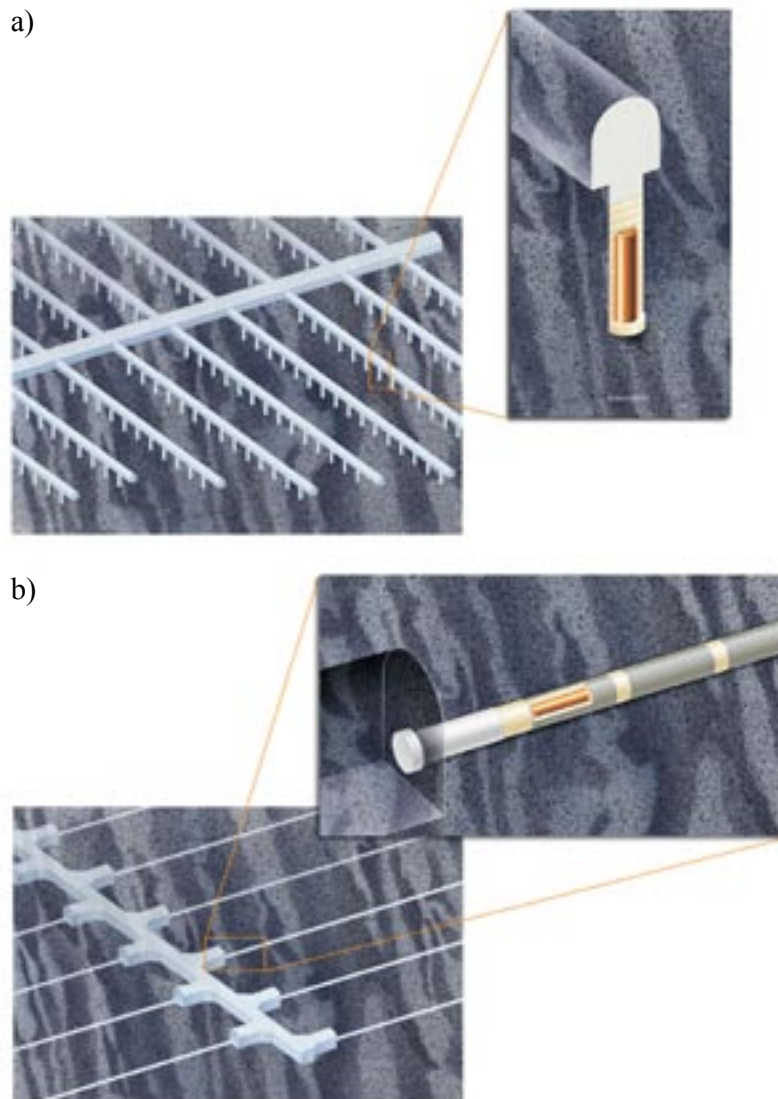


Figure 3-5. a: KBS-3V layout. b: KBS-3H layout.

Each deposition drift is up to 300 m in length and with the diameter 1.85 m. A total of around 45,000 m of deposition drifts are needed. We use two different layouts. In the case of horizontal pull-reaming (see Section 4.6.2) an extra service tunnel is needed, see Figure 3-4. In the case of cluster drilling (see Section 4.6.4) or horizontal push-reaming (see Section 4.6.3) no need for these tunnels exist.

3.2.3 Excavation volumes

The preliminary theoretical excavation volumes for the reference design are found in /Pettersson, 2002/:

- Access ramp: 5,000 m, area 46 m² (270,000 m³).
- Shaft for rock and bulk material transport, skip station: 570 m, area 24 m², diameter 5.5 m (13,500 + 8,500 = 22,000 m³).
- Hoist shaft for staff etc: 500 m, area 24 m², diameter 5.5 m (13,500 m³).
- Ventilation shafts: 3 shafts á 500 m, area 5–10 m², diameter 2.5–3.5 m (11,000 m³).
- Central service area: Miscellaneous openings etc. (130,000 m³).

- Main tunnels.
 - Type A: 2,460 m, 64 m², (171,000 m³).
- Transportation tunnels.
 - Type B: 2,200 m, 46 m², (101,000 m³).
 - Type C: 620 m, 20 m², (12,500 m³).
- Ventilation tunnels, inspection tunnels etc.: Included in the central service area.
- Deposition tunnels: 109 tunnels, around 265 m in length and 22 tunnels around 140 m in length, 25 m² (802,000 m³).
- Deposition holes: 4,500 each at 20 m³ (90,000 m³).

Total excavation volume is around 1.6 million m³ for the reference design KBS-3V with the preliminary annual excavation volumes as in Figure 3-6.

Total volume for the horizontal reaming alternative in Figure 3-4 is around 1.9 million m³

The total volume of the repository using the KBS-3H is around 800,000 m³, with the volume for deposition drifts being around 120,000 m³. In the case of extra service tunnel needed to excavate the drifts using horizontal pull-reaming, the total volume is 950,000 m³.

The deposition rate is described in Table 3-2. The initial rate is 25 canisters per year up to around 160 canisters per year. The design capacity is decided to 1 canister per day.

3.2.4 Geometrical tolerances

The deposition of the engineered barriers necessitates certain gaps between deposition equipment and the rock and between the engineered barriers and the rock. Geometrical tolerances follow from Table 3-3 and Figure 3-7.

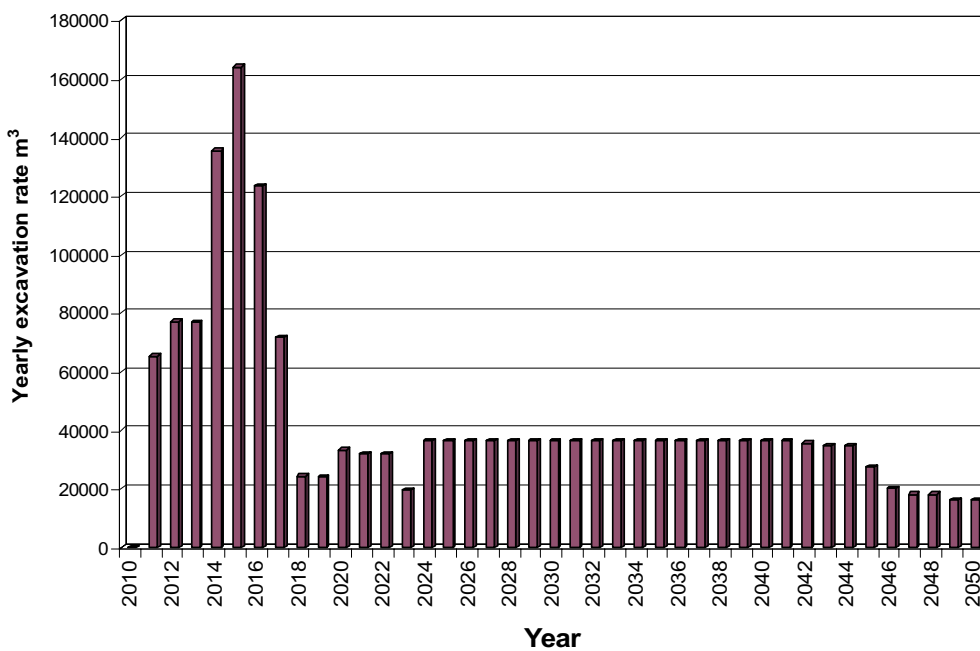


Figure 3-6. Rock excavation volumes (solid m³ per year including overbreaks) for the reference design and implementation plan.

Table 3-2. Assumed deposition rate (# of canisters per year) in the deep repository.

Period	Year	Canisters per year
Initial operation	2018	25
	2019	50
	2020	75
	2021	100
	2022	150
SUM	2018–2022	400 canisters
Regular operation	2023	100
	2024–2044	160
	2045–2052	80
SUM	2023–2052	4,100 canisters
GRAND TOTAL	2018–2052	4,500 canisters

Table 3-3. Preliminary geometrical tolerances for the deposition tunnel (KBS-3V, deposition hole (KBS-3V) and the deposition drift (KBS-3H).

Component	Parameter	Tolerances
Deposition tunnel	Length	Length 285 m ±100 m
	Height, Width	H: < 0.1 m, W: < 0.5 m, Diameter: < 0.2 m (TBM)
	Gradient	> 1%
	Size	Size > than theoretical profile
Deposition hole	c/c distance	6 m ± 1 m (dependent on rock conditions)
	Diameter	1,750 mm –5 < D < 50 mm
	Length	≥ 7,900 mm
	Starting point	< 25 mm from theoretical point
	Alignment	Centre point in the bottom of the hole shall not divert more than 25 mm from a vertical projection of the starting centre point.
	Straightness $\delta D/\delta L$	≤ 0.002 (= < 16 mm). A measured centre point at any depth shall not divert more than 16 mm from a theoretical line between the starting point and ending point.
	Max deviation from center line at the end of the hole	≤ 50 mm
	Surface roughness	± 2–10 mm
	Steps due to re-gripping	Instant horizontal displacement of the centre point < 10 mm
	Deposition drift (KBS-3H)	Gradient (Inclination)
Break-through position of pilot hole		< 2 m from its nominal position
Diameter (Coneness)		≤ 10 mm
Length		< 300 m
Straightness (Waviness, deviation from centerline)		± 2.5 mm over 6,000 mm
Steps		≤ 5 mm
Roughness		≤ 5 mm
Max deviation from the centerline at the end of the tunnel		≤ 220 mm

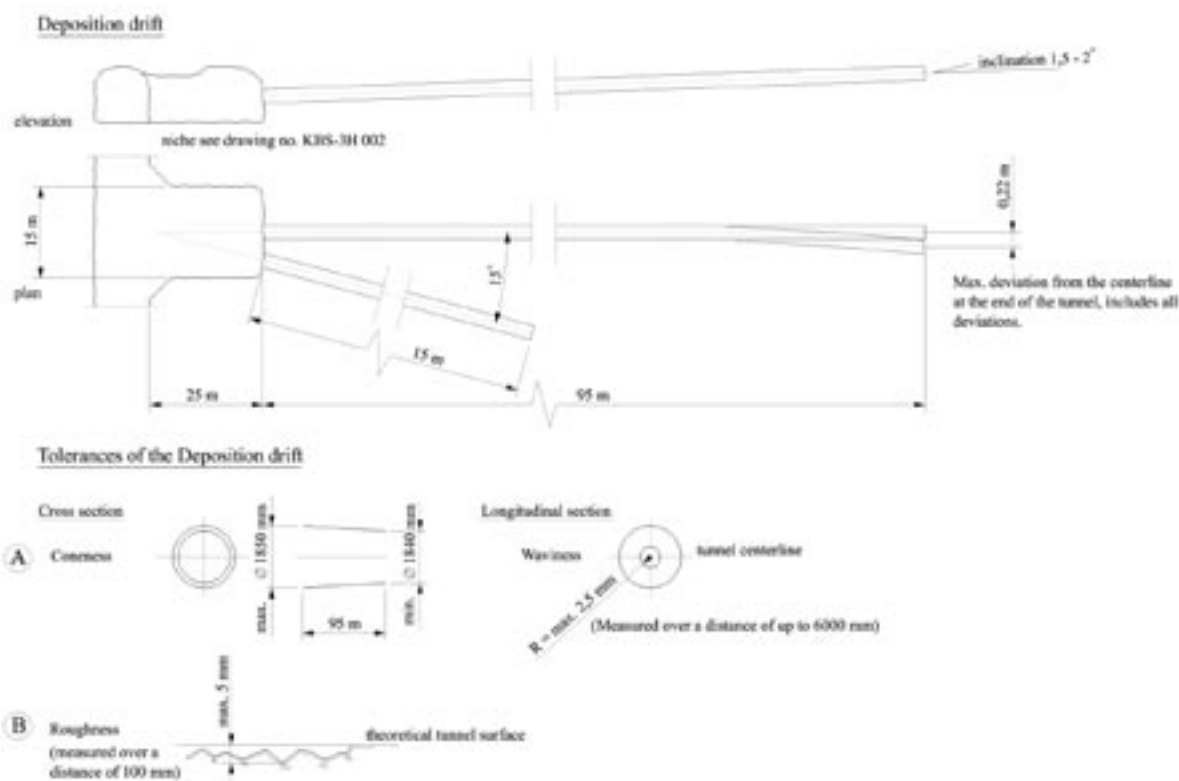


Figure 3-7. Geometrical tolerances for the KBS-3H drift.

3.2.5 Rock conditions

The preliminary site descriptive models of Forsmark and Simpevarp /SKB, 2002b,c/ are used as first estimates with the following assumptions:

- The rock is granitoid and in general self-supporting and in general “good rock”, surpassing rock quality at SFR (Forsmark) and Äspö HRL (Simpevarp). As a conservative approach, the SFR and Äspö HRL conditions are used as baseline conditions.
- The access ramp and main tunnels will sporadically be excavated through minor fracture zones (width < 5 m) that will require rock support.
- Occasional minor fracture zones will also cross-cut the deposition area.
- Minor fracture zones and other fractures are water-bearing.

For estimation of TBM performance we assume average rock strength as measured by Uniaxial Compressive Strength to average 230 MPa. 3–5 fracture sets exist with 0.8–1.6 m between the weakness planes.

3.2.6 Rock support and grouting

Temporary and permanent support is occasional short, grouted rock bolts and shotcrete. For simplicity, it is here assumed that shotcreting in deposition tunnels only occurs at the location of minor fracture zones and that shotcreting is standard practice all along hoist shafts and access ramp and for the central area and main tunnels. The rock support is assumed to be installed during non-excavating shifts.

Independent of rock excavation method it is assumed that rock grouting is by pre-grouting only (grouting ahead of the tunnel face prior to excavation). For raise-boring and for horizontal reaming, two alternatives are used for costing and scheduling: (1) grouting through the pilot hole is sufficient. (2) grouting is through the pilot hole and through four cored holes drilled along the tunnel periphery. The frequency of grouting is assumed to one grouting fan of every 100 m tunnel in the repository area. We assume that a standard grouting fan (Drill & Blast, TBM, mobile miner etc.) consumes in total 3 shifts (24 h) for drilling of grouting holes, testing, grouting and curing. Due to the number of work faces, grouting is not on the critical path on the overall excavation cycle. For horizontal reaming, it is assumed that grouting and curing is made in two shifts.

3.2.7 Detailed site characterisation

SKB has not detailed the plans for the detailed site characterisation underground during construction and operation of the repository. For simplicity here, it is assumed that the major part of investigations of the deposition areas are conducted prior to excavation of the deposition area and that the excavation work is not on a critical path. While it is supposed that a cored hole is drilled along the deposition tunnel, there is less need for probe holes from the drilling rigs. Due to the number of work faces it is also assumed that supplementary investigations of the deposition areas, like mapping etc. are not on any critical path. The basic documentation programme is executed to plan for rock support, excavation (blast) design, etc. The investigations and tests for planning grouting are included in the grouting programme and included in the time estimated for grouting.

3.2.8 Concurrent construction and operation

To maximise utilisation of equipment etc. for Drill & Blast, it is assumed that around 10 work faces are available for excavation see /Pettersson, 2002/. During the regular operation, deposition work and construction work are concurrent activities. At the same time deposition work is ongoing at other areas of the repository when in regular operation. The minimum distance between excavation work and already backfilled and closed deposition tunnels is estimated to 80 m in the reference design.

For the initial construction phase and for the preparation of initial operation, it is assumed that all tunnels in deposition area for the initial operations are excavated before any deposition gets underway. Deposition holes will be drilled in a timely manner, but well ahead of the deposition work.

3.2.9 Scheduling

Due to the continuity of operation for some decades, the basic assumption is that excavation and deposition work is conducted at a steady pace, day-by-day, rather than working in batches of more intensive activities, e.g. to excavate a full deposition area in minimum time. Nine construction shifts a week (8 h/shift) is assumed with 44 working weeks per year, in total 3,168 shift hours per year.

Excavation capacity is dependent on method and follows later in the report.

3.2.10 Costing

For calculation of capital costs, it is assumed that depreciation is made according to expected technical length of life for equipment. Differences in cash flow are discounted by the real interest rate 4% up to year 2020 and 2.5% for the capital costs thereafter in accordance with SKB's yearly fee calculation that every year is submitted to the government for decision, /SKB, 2003/. If not otherwise stated, it is assumed that costs are calculated for the level January 2004 with exchange rates 9 SEK = 1€ and 1€ is 1.2 USD.

3.3 Methodology for this study

The methodology for this study is similar to the methodology used for the Design Justification Statement for selection of access routes to the deposition areas /Bäckblom et al, 2003/ where the main steps are found in Table 3-4.

The decision to be made (Step 1) on excavation methods is of course not to be set in stone for unlimited time in the future, but should serve as a basis for the basic and detailed design of the repository, planning of site characterisation and assessment of long term and operational safety and for costing and scheduling. After licensing, changes can be made in due consideration of the change management procedures that will be applicable. In case the changes will have implications for safety, it is mandatory that an independent safety review is conducted in addition to the safety review. In addition, before the modifications may be introduced, the Swedish Nuclear Power Inspectorate shall be notified and the Inspectorate can decide that additional or other requirements or conditions shall apply with respect to the modifications /SKI, 1998/.

Table 3-4. Methodology applied in this study.

Step	Activity	Comment
1	Write down the decision to be made.	The objective here is to make a principal choice of excavation method for the different underground openings but without consideration of truly site-specific conditions.
2	Establish the objectives to be met.	The objectives are described below. In principle it would be possible to put different weights on different objectives, but such an option is neither thought to be necessary nor prudent.
3	Identify alternatives.	Several options are presented but the main alternatives are Drill & Blast or mechanical excavation see Table 2-1.
4	Collect information so the alternatives can be compared.	This report is an account of compiled and applied information.
5	Evaluate the alternatives in comparison to the stated objectives.	Simplistic scoring is used in this study: "+" (the alternative is more likely to meet the objective, "0" (the alternative is neither better nor worse to meet the objective) and "-" (the alternative is less likely to meet the objective).
6	Total assessment.	Evaluation based on total assessment on all factors in due consideration of Step 5.
7	In case several alternatives score similar in Step 6.	In case two alternatives are equally good, they are re-evaluated, especially in consideration of long term safety, operational safety and environmental impact.
8	Preparation of a Design Justification Statement.	This is an internal SKB document for traceability and transparency of decisions.

The objectives to met by the engineering and used in comparison of alternatives (Step 2) are simplified as:

1. Low radiation dose after closure (ALARA²).
2. No accidents for employees and contractors during construction and operation (incl. ALARA).
3. Small environmental impact during construction and operation.
4. Sustainable management of natural resources.
5. Low Net Present Value cost of construction, operation and closure.
6. Short duration of construction.
7. High flexibility.
8. Low project risks.

The background and interpretations to the objectives stated are described in Section 6.2 in /Bäckblom et al. 2003/ and are not repeated here.

Step 3 is selection of alternatives and these are presented in Table 2-1 in this report. Alternatives deferred are discussed in Section 2.4 and also in Chapter 4.

Collection of information to compare the alternatives (Step 4) utilised SKB internal resources where appropriate. Industrial know-how and experience was utilized to prepare a feasible technical design and to forecast major development of technology within a 10-year period so decisions at hand also utilise information on possible future developments (see the PREFACE). With respect to site-specific conditions, we use available information and also assess how a range of rock conditions would affect the repository dependent on choice of excavation methods.

The evaluation of alternatives and rationale for judgements (Step 5) are described in Chapter 6 in this report and Step 6 in Section 6.6 and Chapter 7. Life Cycle Inventory was used to evaluate environmental impact and sustainable management of natural resources see Section 6.3.3.

² ALARA is a common principle phrased as “keeping the radiation doses to humans As Low As Reasonably Achievable, economic and social factors taken into account” /IAEA, 2002/.

4 Description of excavation methods

This chapter describes excavation methods of particular interest for the Swedish geological repository. We start with a short discussion on methods for shaft sinking from surface before different methods for tunnelling and excavating vertical raises and excavation of deposition holes are reviewed. While KBS-3H is a study in progress we have also included methods to excavate horizontal deposition drifts with diameter 1.85 m. The chapter ends with discussion on possible future technologies for excavation, implications of technology on the repository layout and operation and excavation methods and the human factor.

4.1 Shaft sinking

Mining downward, generally from the surface, although occasionally from an underground chamber, is called shaft sinking. Shaft sinking is really an old technology. Already in the Bible (Job 28: 3–4) *“Man puts an end to the darkness; he searches the farthest recesses for the ore in the deepest darkness. Far from where people live he sinks a shaft, in places travellers have long forgotten, far from other people he dangles and sways.”* The methods used were surely not as efficient as present methods for shaft sinking. Most shafts are excavated by Drill & Blast. By utilizing a sinking stage of multiple platforms which permits concurrent excavation and concrete linings advance rate up to 10 m a day is achieved.

Shaft sinking by mechanical excavation has also been developed, but the feasibility for deep shafts is still uncertain.

The Nordic countries Finland, Norway and Sweden have been traditional mining countries, but modern experience from shaft sinking is almost non-existing due to the popularity of ramping and use of raise-drilling rather than using shaft sinking. One of the latest shaft sinking operations in Sweden was the deepening of a previous shaft in the Åmmeberg Mine in the early 80's. No shaft has been sunk in Finland since the mid 70's, the Hitura mine.

Within the nuclear community several shafts have been sunk in Gorleben (Germany) and for the WIPP-facility (USA). Shaft sinking is ongoing, see /Bäckblom et al. 2003/ at the Bure site in France and the excavation depth now (July 2004) is around 450 m. For Underground Research Laboratories, shaft was constructed at Mol in Belgium, URL in Pinawa, Canada and shaft sinking is in progress in Mizunami, Japan.

SKB has included a skip shaft to be constructed from surface in the reference design instead of using raise-drilling from beneath due to scheduling reason; as construction time can be shortened and this would offset the extra costs for the shaft sinking. It is assumed the shaft will have a circular form and with diameter of 5.5 m down to the depth ~570 m.

4.1.1 Excavation by Drill & Blast

Excavation is by drilling and blasting with muck loaded into large buckets, with larger shafts operating several buckets alternately in hoisting wells extending through the platforms. Grouting is carried out 50–100 m ahead of the shaft face to seal out water. Best progress is achieved when the rock is pre-grouted from a number of holes drilled from the surface before the shaft is started.

Shaft sinking can be divided in two basic steps:

Step one: Mobilisation at site and pre-sinking to 70 m below surface

This step takes around 7 months. During this period all equipment will be purchased, transported to the site and erected. Founding for headgear and winders are concreted. Shaft sinking is made by temporary equipment, using mobile cranes and using provisional platforms to the depth of 70 m to permit a safety distance to the shaft sinking platform of around 50 m when step 2 starts.

Step two: Sinking of the shaft from depth 70 m to depth 570 m below surface

Typical equipment for shaft sinking is shown in Figure 4-1 and in Figure 4-2. By using a platform with three to five decks several tasks, like drillings for rock support and rock characterisation are simultaneous activities. The drilling of the blast holes for the round is done with a drill jumbo with four to eight booms depending on the diameter of the shaft.

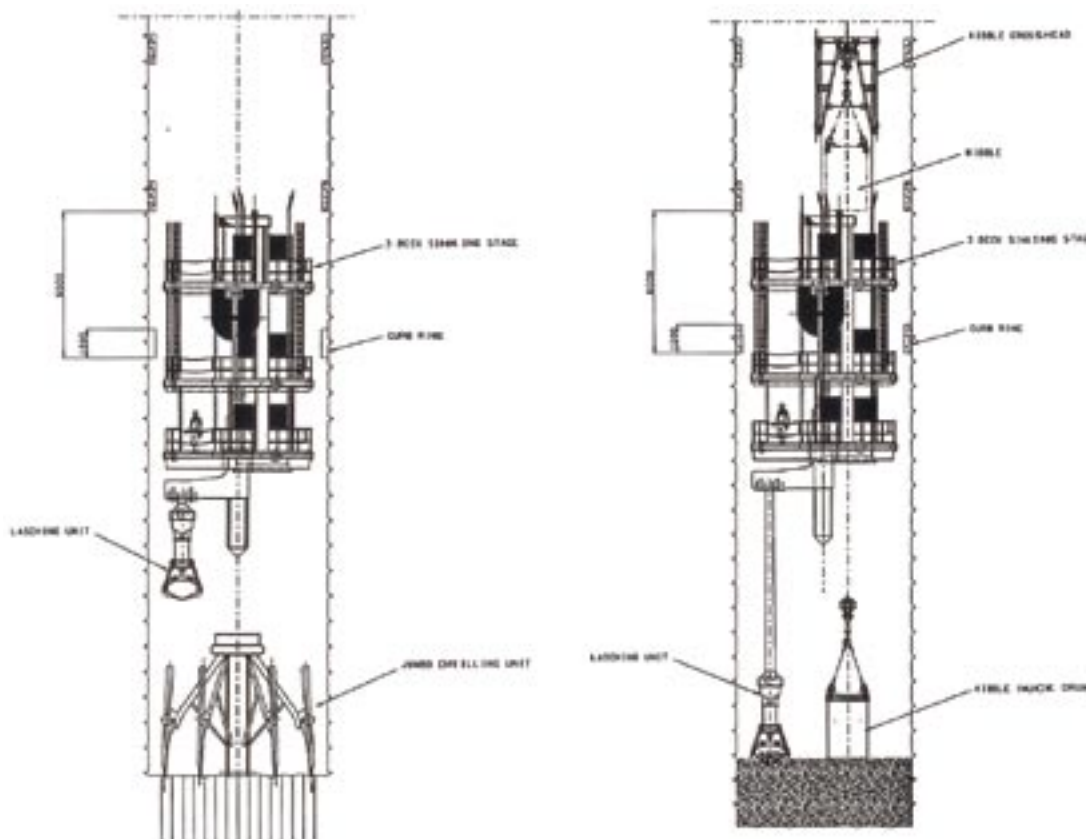


Figure 4-1. Sketch of a shaft sinking operation. Drilling (left). Mucking with grippers and bucket (right).



Figure 4-2. Pictures from equipment at shaft sinking in Sedrun, Switzerland. Drilling rig with 8 booms (left). View up from the shaft floor (top, right). Mucking equipment with grippers and bucket (down, right). Courtesy: Electrowatt Infra.

Following activities are included in the work cycle:

1. Lowering of the platform.
2. Form-work for the curb ring (for rock support and depressurizing/water collection) is dismantled.
3. Additional lowering of the platform and form-work for a new curb ring is assembled and prepared for casting (Distance between rings is around 6 m).
4. Curb ring is cast and left for curing.
5. Partly in parallel with 1–4, mucking is done, but leaving around 1 m of broken rock.
6. Drilling and installing rock bolts.
7. Shotcreting (spraying of concrete) of the wall.
8. Final mucking including rebound shotcrete.
9. Cleaning of the shaft bottom.
10. Drill jumbo is lowered, drilling of holes for the next round. The jumbo is retracted to a safe level or to the ground level for maintenance.
11. Charging of explosives and lifting of the platform around 60 m.
12. The round is fired.
13. Mucking.
14. Installation of air, water and ventilation pipes.
15. Repeat of 6–14.

For the Drill & Blast operation, some 85 holes of 51 mm and 1 hole of 127 mm will be used per round, length 3.5 m. Specific drilling is around 3.3 drill meter/m³ with specific charge or powder factor (amount of explosives) of around 3 kg/m³. The powder factor needs to be quite high to lift the rock and also ensure good fragmentation for efficient mucking. Probing and grouting is systematically performed ahead of the shaft bottom.

The advance per day can be up to 4 m. We here assume that 40 m is achievable per month, including grouting work.

4.1.2 Excavation by mechanical excavation

Mechanical equipment for shaft boring has been developed and used, but has not yet been successful in hard rocks, the main problem being the removal of cuttings from the shaft bottom.

Three different techniques have been used /Nirex, 1992/:

- Blind Shaft borer – essentially a full face tunnel boring machine, boring downwards with its own means of head rotation and forward propulsion by grippers. Spoil is removed by skip or hydraulic means. The machine of 7.5 m diameter bored to 189 m depth although depth 370 m was planned as a demonstration test. Spoil was removed from the head by bucket elevator to hoisting skips and considerable difficulty was experienced with spoil pick-up. A technique using slurry pumps made it to a depth of 200 m.
- Partial Face borer – a cutting wheel with its axis horizontal which articulates to sweep the cross section of the shaft. As the area cut at one time is less the thrust requirement is much less. As a result advance rate will be less but spoil pick-up, a draw-back on the full face borer, is improved. Spoil removal may be similar to the full face borer. A concept using a Mobile Miner was developed in the 80's but never came into production.
- Blind drill – a cutting head which is driven downwards through a drill string from the surface and which is thrust forward by a stack of weights placed on the back of the cutter head. Spoil is removed by hydraulic lift within the drill string. A 4 m diameter hole has been sunk to 800 m in the Agnew nickel mine in NW Australia. The equipment used, even at 893 tonnes of lift, still did not have enough capacity. In harder rock formations it could not carry enough weight to penetrate about 15 cm/hr.

Due to the hard rocks at Forsmark and Oskarshamn and the very few if any, successful industrial references for mechanical shaft excavation in such conditions, these alternatives are not further studied.

4.2 Use of Drill & Blast in tunneling and excavation of central area

The reference design assumes Drill & Blast for excavation as the preferred method for excavation of deposition tunnels, access ramp main and pilot tunnels and the central area. The specific design of the excavation work will be developed within the general engineering work. Some details are here extracted to allow for comparison of other excavation methods. The main focus here is on the deposition tunnels where a main issue is to limit excavation damage.

4.2.1 General description of the Drill & Blast work cycle

We here provide general description of Drill & Blast work cycle (drilling, charging/blasting/scaling) before discussing the specific applications for deposition tunnels etc.

Drilling

We here review present status and some recent trends in development of equipment for drilling with the notion that the basis for smooth blasting with low excavation damage is precision in drilling.

Drilling steel/Drill bit

The drill bit diameter used in tunnelling is between 45 and 57 mm, most commonly 48 and 51 mm. In the early 1990's, both 57 mm and 64 mm diameter drill bits were tested in tunnel rounds. The number of holes in a round could be reduced, but at the same time the weight of the maximum cooperating charge increased heavily, resulting in increased magnitudes of air shock waves and ground vibrations. Another reason for using 64 mm drill bits was to achieve a stiffer drill pipe. Rounds with varying blast hole diameters have also been tested. In the contour holes, for example 45 mm drill bits could be used, while 57 mm or 64 mm holes were used in the remaining round. Today, the empty large hole in the parallel cut is commonly drilled using a diameter of 102 mm. During the past years, the development of drifter rods has shifted towards a higher stiffness of the contact between the drill pipe and drill bit, and smaller borehole diameters may be drilled with better straightness than before.

Drilling rigs

The main components of a drilling rig are shown in Figure 4-3. The drill bit is screwed on a drill rod that is attached to the drill machines and mounted on manoeuvrable booms. The machine is on feeder that slides as the drill holes are bored.

Drilling rigs of today have developed relatively fast over the past five years with regard to production technology, compactness, use of information technology and also to some extent the drilling equipment itself, primarily the impact frequency. Rigs are equipped with standard systems for control and computer communication on board. The cabin has over the past 10 years become vertically adjustable, and therefore has improved the possibilities for the operator to actually see the collaring (positioning) of the drill holes. The construction of the drilling rig in modules makes it possible to upgrade single components, and modules may be put together following the requirements of the Owner.

Booms

Drilling rigs commonly used today are equipped with 2 or 3 booms. Trials with 4 booms have taken place. Two-boom rigs are commonly used for tunnel sections of up to 80–85 m². A large modern 3-boom rig can handle tunnel sections of up to 150 m². A rig with four booms has recently been tested. With the use of improved control and some automation use of 3-boom rigs are more common.

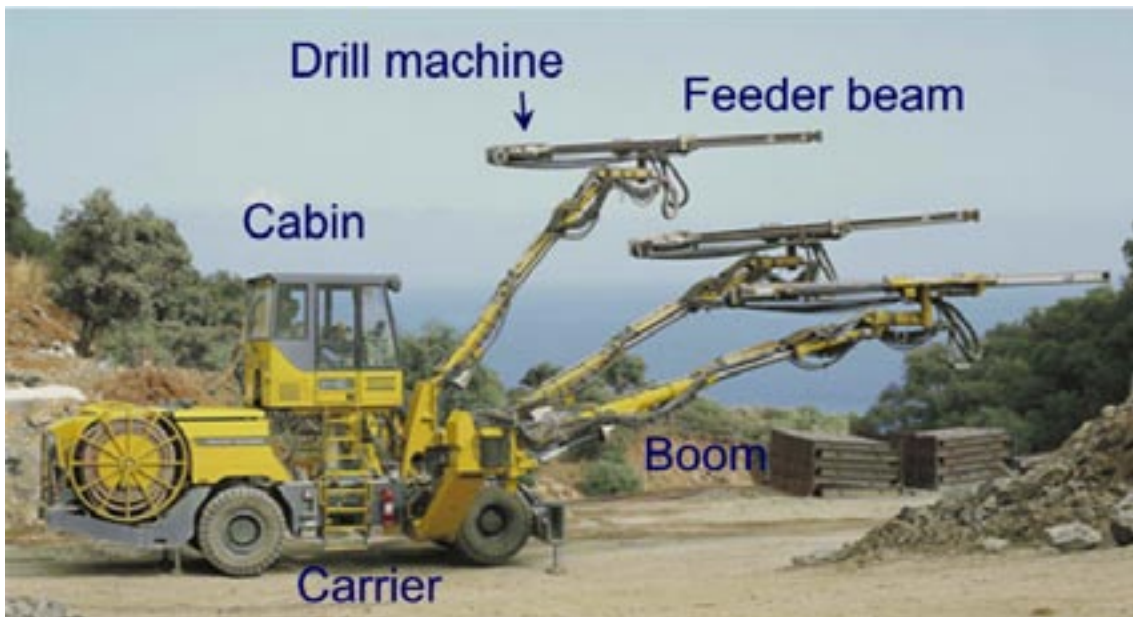


Figure 4-3. Main components on a drilling rig for tunnelling.

Drill machine

Drill machine power has increased to some extent over the past few years, and 18–24 kW machines are commonly in use. Trials with altered impact frequencies have been carried out. The increase in rate of penetration (ROP) over the past 10 years amounts to some 20–40% and is presently around 2.2–2.3 m/min in normal hard rock conditions.

Feeder length/Length of round

Feeder lengths (drilling steel lengths) have increased one size from 4.8 m to 5.4 m with the corresponding drill hole lengths 4.5 and 5.2 m. In the early 1990's lengths of rounds of up to 7.5 m were tested at LKAB's iron ore mine at Malmberget. Tests with similar lengths of rounds have also been made in Canada and Australia.

Automation

The availability of various levels of automation has increased. Today, rigs range from being almost totally manually operated to being fully automated. However, the accuracy of navigation today is not particularly good; collaring is currently 10–20 cm in practice. Tests with fully automated drilling have been performed in Canada by the mining company INCO among others. Drilling rigs without operators were used in these tests. The rig moved to the appropriate face from a service drift, drilled a round and then withdrew – without any operator.

Auxiliary equipment

Computerized directional and positioning systems of various types and makes are today available as standard equipment on drilling rigs. The systems present the drilling performance in terms of collaring location, the direction of drill holes, rate of penetration, feeding and rotational pressure etc. As options, the rigs may be equipped with additional instrumentation, for example probe drilling using Measurement While Drilling (MWD) and measurements of tunnel profile (a so called profiler).

Drilling precision

The customer usually defines his demands in terms of drilling accuracy for blast holes and grouting holes in the technical specification of the particular project. Swedish construction industry in general uses standardised specifications /Anläggnings AMA 98, 1999/ but requirements are also defined in other ways. The drilling tolerances are defined in terms of requirements on the blasted tunnel section in comparison to the theoretical tunnel section, Table 4-1 and Figure 4-4.

Table 4-1. Geometrical tolerances for tunnel and underground caverns etc. From AMA 98 Table CBC/4/.

Tolerance class	Maximum permissible measure – expressed as mean ¹ of c and d for distance between excavated contour and theoretical contour	Maximum permissible deviation for an individual drill hole in tunnel wall or roof compared to theoretical contour	Maximum permissible deviation for an individual drill hole in tunnel floor compared to theoretical contour
1	0.30 m	0.70 m	0.80 m
2	0.35 m	0.80 m	0.90 m
3	0.40 m	0.90 m	1.00 m

¹ Mean for measure c and d (in Figure 4-4) is calculated as $(c + d)/2$.

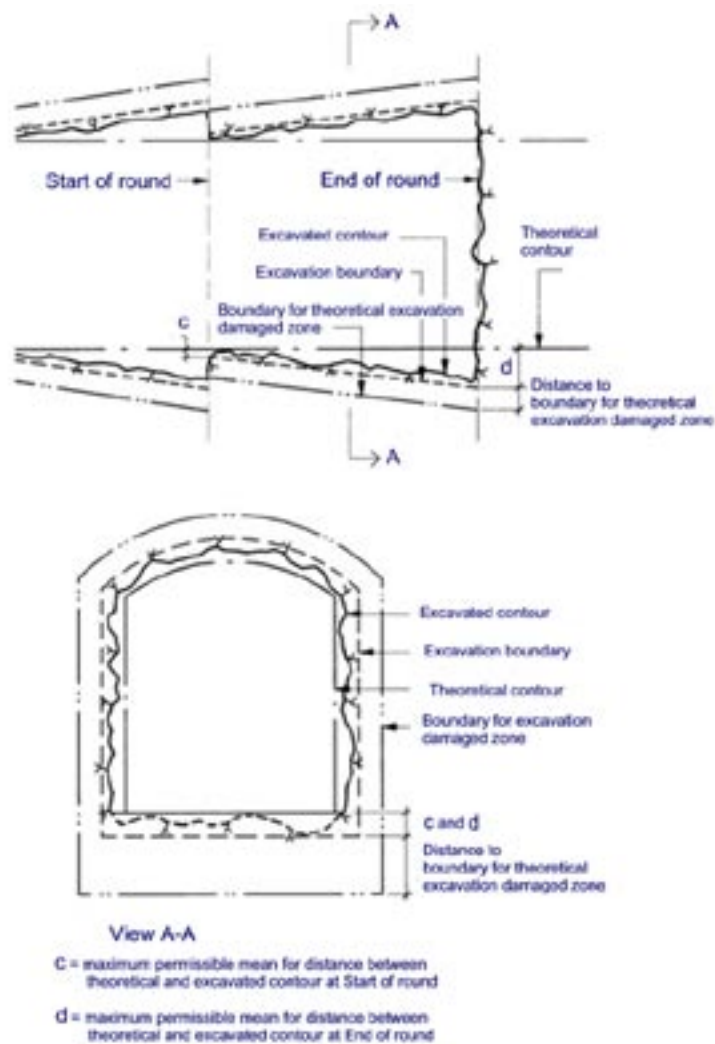


Figure 4-4. Explanation of tolerances as defined in Table 4-1. From AMA 98 CBC/2/. See also Table 4-3 for definition of extent of excavation damaged zone.

Drill pattern

The hole pattern and overall blast design with regard to openings, contour, support holes and stope holes are made in the same way as 10 years ago as the drive for improvements in drilling and charging plans has been rather weak. The demand for a reduced Excavation Damaged Zone in the tunnel floor has resulted in an increased number of holes in the bottom row and charges with lower charge concentration.

Summary and conclusions – present drilling technology

The development of equipment and user technology during drilling has progressed slowly, without any major breakthroughs. The drilling rigs are better built and possibly easier to operate, but the rounds look the same as earlier, and the blasting results are similar. Some development projects have been carried out in order to try new lengths of rounds, drilling diameters, openings etc. However, these tests have not resulted in any major breakthroughs. The accuracy in directional and positioning systems have improved to some extent, thereby improving also the tunnel geometries. The equipment is better at repeating movements as well as registering movements, directions and positions. The direction and contour smoothness of the tunnel has improved. The amount of overbreak has decreased. This has also resulted in a possibility to create drilling patterns with better precision. The drilling rigs are better at registering production data than 10 years ago. The working environment in the cabins has also improved over the past decade.

Should the drilling equipment be adapted to customer demands, or vice versa? Surely, requirements should be reviewed with regard to their relevance for a defined end result. However, the time is ripe for strengthened requirements on drilling as a part of the tunnelling process.

The equipment to be used in the deep repository must be designed and constructed according to the requirements put on the repository design. If the requirements regarding EDZ, rock surface smoothness and tunnel section tolerances deviate from the standard tunnel, drilling rigs should be adapted accordingly, for example that the different booms of the drilling rig are equipped with beams and drilling machines adapted for contour drilling or the drilling of openings. Better precision in positioning and directional systems is also required.

Blasting

Explosives

There are several types of explosives in use and one possibility to divide the alternatives is the way they are delivered:

- Cartridge explosives: Dynamite, Emulsions.
- Bulk explosives: ANFO, Water gel, Emulsions.
- Boosters and detonating cords.

The most common explosives used for tunnelling in Sweden are emulsions, ANFO, nitro glycerine and nitro glycol based explosives (dynamite). Around 70–80% of the explosives used for tunnels are emulsions. A common technique is site sensitized emulsion where the raw materials are an emulsion matrix and a gassing agent that is mixed during the charging. It is possible to vary the strength and amount of explosive for each hole and therefore all holes in a round could be charged by the same explosive.

Table 4-2 is an overview of the characteristics of common explosives.

Table 4-2. Characteristics of explosives.

Explosives	Components	Charge dimensions (mm)	Density (kg/m ³)	VOD ¹ (m/s)	Water resistance	Application
Dynamites	Nitro glycol, Ammonium Nitrate	22–85	1,400	2,500–6,500	Good	Cut, stoping, floor
Small diameter pipe charges	Nitro glycol	17–32	1,000–1,200	1,400–2,400	Less good	Contour
Emulsions	Ammonium nitrate, oil, wax, gassing comp, micro-balloons	17–90	750–1,250	5,000	Good	Cut, slashing, floor, contour
ANFO	Ammonium nitrate, fuel oil	Bulk	700–900	2,500–3,000	Poor	Cut, slashing
Detonating cord	PETN	Bobbin	1,100	6,500–7,000	Less good	Contour
Primer	Mixture	15–65	1,500	> 6,000	Very good	Initiating

¹ VOD: Velocity of Detonation

Cartridge explosives may be found in many dimensions and packaged in paper, plastic or pipes. The explosive is delivered in cardboard boxes. The producers supply data on the explosive and how it is delivered. The explosives are charged by dropping or pushing the desired number of cartridges into the bore hole. Advantages are that dynamites and emulsions are water resistant with no need for special charging equipment and being available in many dimensions. Decoupling is available (charge diameter < hole diameter) and we have a controlled amount of explosive in each hole. Disadvantages are that Nitro glycerine or nitro glycol based explosives may cause a health inconvenience (headache). Nitro glycol is also one of the chemical substances included in the Swedish National Chemicals Inspectorate list recommends to be avoided.

Bulk explosives are delivered in bags or trucks and the explosives are charged pneumatically by air pressure (ANFO) or by hydraulically by pumping (emulsions) or gravity charged. Advantages are rational and effective charging with automation of the charging operation being possible. Density and weight can be varied and decoupling is possible in horizontal holes. The disadvantage is that special charging equipment is required, difficulties to reduce strength. Spillage (for ANFO) is also quite high due to the charging technology.

Detonating cords are delivered in bobbins. The explosives have a very high velocity of detonation (VOD) and are sensitive to friction and impact. The cord exists in many different diameters and charge weights. It used to be a common technique to charge the contour holes with 40 g/m or 80 g/m cord. Nowadays it is not permitted to cut such sizes of detonating cords underground in conjunction with charging work in Sweden.

Primers are a special form of explosives used to initiate the main explosive. These primers are cartridge and characterised by a very high velocity of detonation.

Charging equipment for tunnels

Cartridge explosives are manually charged. Bulk explosives are charged pneumatically (ANFO) or by pumping (Emulsions). When ANFO is used it is poured into a steel vessel on a truck and this vessel is then pressurized by air. The explosive is transported into the boreholes by a hose connected to the vessel. The system with emulsion consists of a charging truck, tanks and pumps. The truck itself consists of different tanks for the emulsion matrix and the gassing component. The emulsion matrix and the gassing component are mixed during the charging process and become an explosive after being pumped by a hose into the boreholes. This means that transportation and storage of explosives becomes unnecessary. It is possible to vary the strength and amount of explosive for each hole and therefore all holes in a round could be charged by the same explosive.

Firing methods

The explosive must be initiated in order to start the explosive process. This is done by the detonator. The detonator consists of a small PETN (PentaErithrytol TetraNitrate) charge and a delay unit. The delay unit consists of a pyrotechnical element (ordinary caps) or an electronic device (electronic detonators). Three main types of firing methods are used, electric detonators, non-electric detonators and electronic detonators. The electric detonators consist of the cap and a wire. The caps are triggered by an electric current. Both half-second delays and millisecond delays exist. Electric detonators are not presently used underground in Sweden. In the non-electric detonators the electric wire has been replaced by a plastic tube through which a shock wave is transmitted. The shock wave from the plastic tube initiates the delay element in the detonator. The plastic tube is lined on the inside with a thin layer of reactive material which transmits the shock wave with a velocity of approximately 2,000 m/s. A system with special delay times exists for tunnelling. The use of electronic detonators may open up new possibilities in rock blasting. The pyrotechnical device is replaced by electronics, resulting in very precise timing and a small scatter. The cap is initiated by a special coded electric signal through the connecting wires.

Smooth blasting

Smooth blasting is carried out by drilling the contour holes with a short hole spacing 0.3–0.6 m and using weaker explosives. A common way to reduce the strength of the charge is to use decoupled charges (charging diameter < hole diameter). Normally the contour holes are charged with small diameter plastic tube charges like “Gurit” (now called Dynotex) or Kimulux. Charging a thin string of emulsion in the bottom of the holes is another way of reducing the strength. Empirical knowledge was previously in use in Sweden for the evaluation of excavation damage caused by blasting. Commonly used explosives were listed in order of their equivalent linear charge concentration in terms of kg Dynamex per metre. The table suffered from many shortcomings and was only verified for very few explosives under specific circumstances. Also, a clear definition of damage was lacking. The revised standard specification in Sweden /Anläggnings AMA 98, 1999/ defines the depth of the theoretical excavation damage zone as a function of the charge concentration in kg Dynamex/m, see Table 4-3

The excavation damaged zone in the rock, or length of the blast induced cracks, will be reduced by using decoupled charges and simultaneous detonation of the contour holes. The crack length is also dependent of spacing and the presence of water in the holes. /Olsson and Ouchterlony, 2003/.

Table 4-3. Theoretical damage zone depth in relation to charging concentration. AMA 98, CBC/2.

Maximum theoretical damage zone depth ¹ (m)	Maximum charging concentration (kg Dynamex/m)
0.2	0.1
0.3	0.2
0.5	0.3
0.7	0.4
1.1	0.7
1.3	0.9
1.7	1.3
2.0	1.6

¹ Micro cracks caused by blasting and that also may be created outside the defined damage zone stated here. The theoretical damage depth should be decided in consideration free coupling, water in the drill hole, rock properties, type of initiation method, charging weight and true drill bit diameter.

Cautious blasting

The purpose of cautious blasting is to reduce the vibrations from blasting, the air shock wave, fly rock and noise. For a repository both smooth and cautious blasting is necessary underground. The cautious blasting underground is necessary not to impair the function of the deposited spent fuel, monitoring and measurement devices and to protect the installations. No limits have yet been defined. The frequencies of interest for stability of openings are related to wave lengths $\lambda/4$ and with p-wave velocity around 5,000 m/s we find frequencies in the range of 125 Hz to 625 Hz for openings with diameter 2–10 m.

The vibration level is mainly influenced by the weight of co-operating charges, the distance to the object and the geology. By reducing the number of detonators of the same delay number the co-operating charges will be reduced. Conventional detonators with the same period number always have a certain time scatter due to the difference in burning time of the pyrotechnic delay element, Table 4-4, Table 4-5. This means that only some of the detonators within the period will interact. The interaction will depend of the frequencies of the ground vibrations. The frequency spectrum is a function of distance from the blast and the attenuation of the p-wave. As can be seen on the high-lighted areas in the table, the reduction factor for the time distributions in the proposed initiation pattern higher than Delay Number 5 between 1/3 and 1/6 for frequencies around 50–100 Hz.

Table 4-4. Time delays for Dyno Nonel long period detonators that also are typical for other detonators using pyrotechnic delay elements.

Delay number	Delay time Milliseconds (ms)	Time distribution ms (\pm)
0–2	25–200	5
3–5	300–500	8
6–10	600–1,000	30
11–20	1,100–2,000	50
25–60	2,500–6,000	150

Table 4-5. Time scatter for different detonators.

Frequency Hz	Time distribution (ms (\pm))				
	5	10	25	100	200
5	1	1	1	1/2	1/3
10	1	1	1	1/3	1/3
20	1	1	1/2	1/3	1/3
50	1	1/2	1/3	1/3	1/6
100	1/2	1/3	1/4	1/6	1/6
200	1/2	1/3	1/4	1/6	
400	1/3	1/3			

Monitoring of Drill & Blast Operation

In order to obtain a good final product, the requirements on all unit operations will be stiff. A control program will therefore be required to monitor the Drill & Blast operation. The most common methods for measuring and controlling the blast result are:

1. Control of drilling to ensure that drilling and drilling deviations are within the specifications.
2. Functional control of the initiation (vibration and detonation velocity). By using functional control of the detonation sequence in a tunnel round it is possible to see whether the holes are detonating in the way it was planned. Not detonated holes may yield a poor particle size distribution and high emissions.
3. Vibration measurements may be a useful tool to control the round. A high vibration level could be a warning signal of a bad function of the round. Vibration measurements are also used for estimating damages in the remaining rock. However, the method may be unreliable.
4. Counting the number of half pipes is a simple method to evaluate the smoothness of the tunnel contour.
5. Studying the cracks in the remaining rock is a better way to evaluate the quality of the contour. One of the main objectives in the design and construction of the deep repository is to keep the EDZ low.

Scaling

Drill & Blast operation generally produce loose blocks and that is why we assume scaling (the work to clear a newly blasted area from loose rock) to be a part of the work cycle.

At present, mechanized scaling or manual scaling is common practice in many mining and construction applications. Water jet scaling may be considered to be a slowly emerging technology. A brief description and evaluation of various contemporary scaling methods is presented below.

Manual scaling

The old standard method of scaling loose rock by hand with scaling bars is still in use. The advantages are: 1) No further damage to the rock surface is made. Only loose rock can be torn down. 2) The sound of hitting loose rock is a very good indication of how stable the remaining rock is. The disadvantages are that the method is physically demanding and potentially unsafe, and is therefore not recommended for scaling of rounds in ordinary operation.

Mechanical impact scaling

A hydraulic hammer on a powerful boom, mounted on a stable rig, is the most common scaling equipment. A large amount of energy is available, which is especially beneficial in order to accomplish scaling of the tunnel face. There is also a good possibility to knock down protruding rock. However, the powerful hydraulic hammer may bring about new cracks in the rock surface, which is a disadvantage of this method. The method is expensive and is subject to high maintenance costs. Also, a high degree of vibrations constitutes a serious problem in terms of working safety and health.

Scaling by ripping

With this method, a powerful hydraulic arm is mounted on a stable bearer. The working tool rips/tears down loose rock with similar methodology as for the roadheader see Figure 2-19. This method is less harmful to the rock surface, but also less effective for face scaling. It is also more difficult to estimate scaling results. The working environment is similar to that of percussive scaling.

Other mechanical methods

Tests have been carried out on various alternative mechanical scaling methods, such as hitting the rock with weights mounted on a rotating wheel, as well as using milling cutters. These methods are not standard methods today. Combinations of percussive and reaming/ripping scaling methods can also be found.

Water jet scaling

High pressure water jet scaling (pressure 100–250 bar and water flow of 100–300 L/min) has been tested in several places, for example in the ongoing railway tunnel construction at Hallandsås and in the LKAB mines, but does not yet provide a standard method. An example of the recent progress of this work in Canada was presented by /Swan et al. 2003/. In this method, the water jet nozzle is mounted on a relatively light and flexible hydraulic boom. Advantages with this method are that the equipment is not subject to any high mechanical stress, and maintenance costs are therefore low, the scaling is careful towards remaining the rock, and only loose rock is torn down, the water jet cleans the rock surface, which is beneficial to the adhesiveness of shotcrete and also creates good visibility for rock surveying and the working environment is good, with limited vibrations. Disadvantages with this method are that the method is ineffective during face scaling, especially when the advance per round is poor, the spraying of water and fog generation results in reduced visibility in certain angles, it is difficult to “hear” scaling results and that water consumption is rather high also imposing need for drainage systems. High pressure water jet scaling would be particularly suitable when smooth blasting is used and where the advance per round is high. This method is also judged to be efficient for re-scaling purposes.

Rock support and grouting

A part of the work cycle is rock support and grouting but these technical aspects will not be dealt with explicitly in this report. However we evaluate how rock support and grouting may be integrated in the work cycle for the different technical options. The Drill & Blast operation is very flexible and therefore the excavation work itself does not put severe restrictions on how to execute the rock support

Mucking and haulage

We assume here that conventional Load-Haul-Dump vehicles (LHD) are used see Figure 2-12. Loading may use an electric-driven vehicle with standard trucks for the haulage to the skip station.

4.2.2 Excavation of deposition tunnels

The description of excavation is following the work cycle as shown in Figure 2-12. In the reference design it is assumed that several work faces (around 10) are available at any time.

Shape of tunnel and drill patterns

The reference design assumes a deposition tunnel with

- approximate section is 25 m²,
- size of section approximately 5.5 m · 5.5 m and
- each deposition tunnel 100–300 m in length with total length of tunnels around 32,000 m.

The main priority for the operation is to limit excavation damage by smooth blasting and to limit disturbance on repository operation by cautious blasting.

To limit excavation damage the shape of the tunnel should be rounded. A new proposal for the shape of deposition tunnels is shown in Figure 4-5. The section is designed to contain both the need for required space and to allow for easy breakage of the tunnel perimeter. This section is also favourable from a rock mechanical point of view. The bottom corners have been strongly arched in order to release the confinement of the corner holes. The flat middle section of the bottom is kept horizontal to facilitate mucking during excavation but also to allow accessibility for all kinds of transports in the tunnel during its entire lifetime. After the road bed layer has been spread out, the floor width of the tunnel is approximately 4.5 m.

The drill pattern and charging plan is designed to minimize damage to surrounding rock and to reduce the surface roughness of the tunnel contour. This is reflected in the large number of blast holes especially in contour and helper rows, Figure 4-6. Consequently, burden (distance from explosive to free surface) and spacing of drill holes are comparatively short. The specific drilling is 4.3 drillmeter/m³ of solid rock with a drilled length of 4.5 m. The 4-hole cut is drilled with the rest of the round and can be drilled using the same equipment as for the ordinary 45–48 mm drill holes. Longer rounds are expected in the future due to improvements in drilling accuracy. Longer rounds will improve the surface roughness of the tunnel contour due to the increased distance between the round intersections and the typical unevenness from the look out. The expected advance using the single hole cut is 4.35 m.



Figure 4-5. Suggested shape for the deposition tunnel. The tunnel section area is 27.4 m².

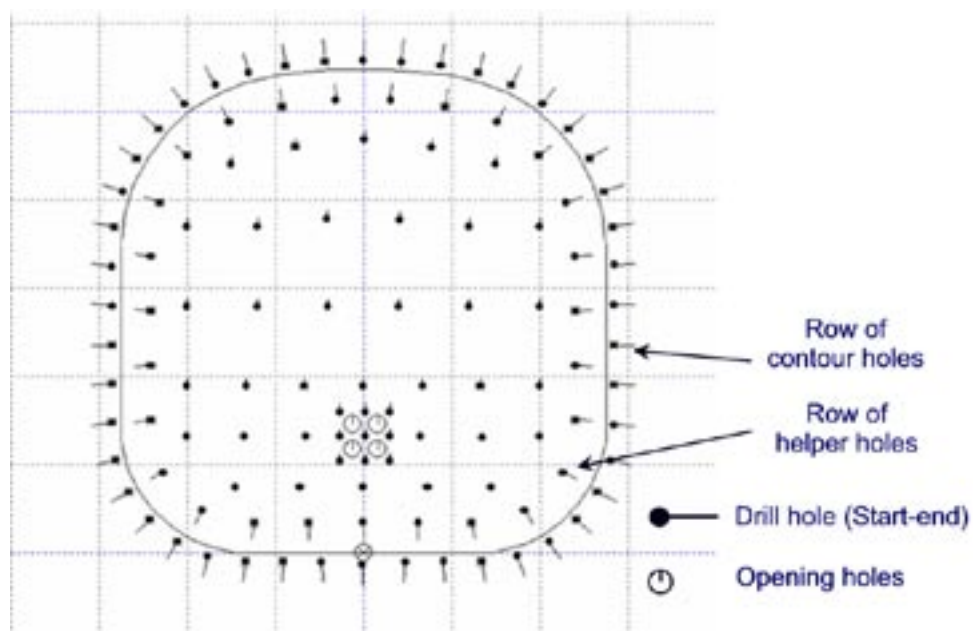


Figure 4-6. Drill pattern for deposition tunnels using a 4-hole cut where opening holes are 102 mm in diameter.

An option for the opening cut is to use a single 300 mm diameter pre-drilled large-hole cut, see Figure 4-7. The single 300 mm hole cut have been found effective in tests at the LKAB mine in Malmberget /Olsson and Fjellborg, 1996/. Very good advances were reported with improved tunnel contour. The large hole could be pre-drilled up to 50–100 m, thus enabling a faster drilling time in the cycle. This hole may very well be combined or integrated in the geological pre-investigation work. The disadvantage using this kind of cut is the requirement for special drilling equipment. This drilling device may be integrated into the blast hole drilling rig.

It is also possible to blast the tunnels in two steps, for example by separate slashing of the floor, see also page 67.

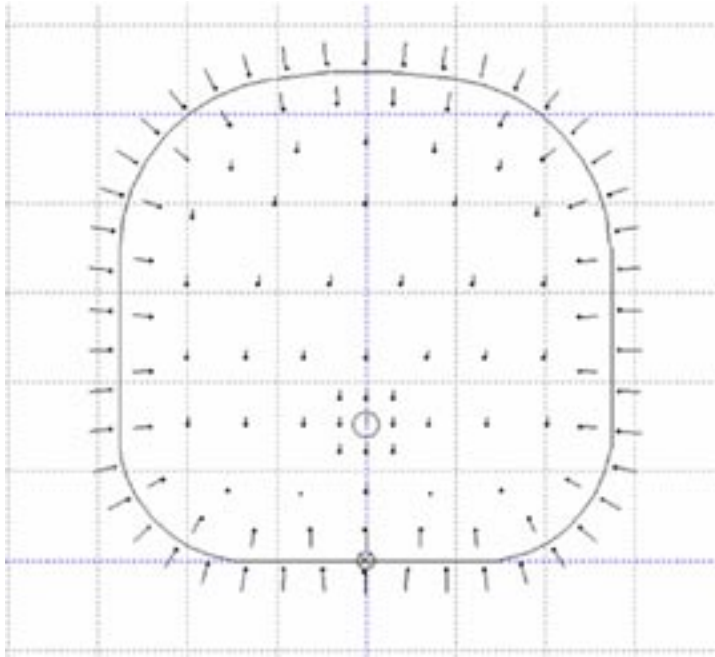


Figure 4-7. Drill pattern for deposition tunnels using a single 300 mm hole.

Drilling rig

To achieve high-quality drilling with small geometrical deviations special equipment may need to be developed as the drilling rigs that will be available in 10 years according to the equipment suppliers are similar to the drilling rigs of today. The booms of the drilling rig then are similarly equipped, which is a compromise since the requirements when drilling a tunnelling round vary depending on the type of borehole to be drilled. Present drilling rigs are constructed in a way that all booms should be able to drill all types of boreholes which is no ideal situation to minimise the excavation damaged zone.

One option would be to develop a portal rig with rigid feeder beams, guided on a beam system with the same shape as the final tunnel section. This rig would drill contour holes without having to depend on long-reaching booms that may be mechanically bent, followed by bad precision during collaring and alignment. Inside the beam system, a number of booms with drilling systems for openings and other types of boreholes would be placed. The portal rig is transported to the face by a rubber wheel vehicle, and is then attached with hydraulic or mechanical jacks. A supply unit (electricity etc.) is mounted on the equipment. The rig may also be equipped with a charging module that automatically charges the boreholes. The operator is seated centrally and is able to monitor several operations simultaneously.

Explosives and charging

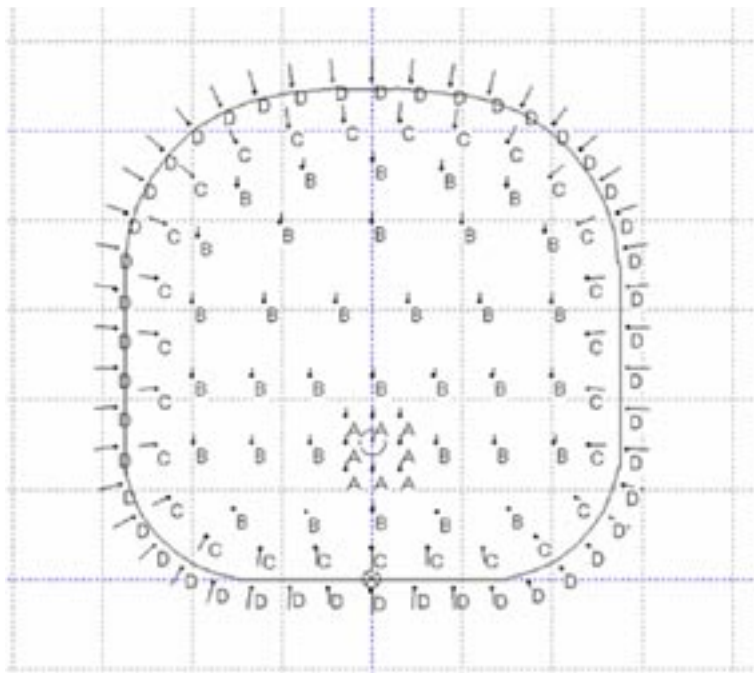
All blast holes are charged with a pumpable emulsion explosive which is sensitized in conjunction with charging, see the previous section. Many different manufacturers are presently developing this charging technique and it is expected to be the main explosives system during the next 10–20 years. Charge concentrations from 1.7 kg/m in a 48 mm hole down to 0.2 kg/m would be available. The explosive and the detonators could be automatically charged by using a charging module on the drilling rig or by a separate module positioning itself to the correct position so that boreholes are easily located. The explosive is pumped from a separate tank and detonators and primers are stored in a special cassette that is automatically moved to the charging hose and the charging system remotely

monitored from a protected location. Another way of charging the blast holes is to use prefabricated charges. However, with this method the advantages with pumpable emulsions cannot be taken advantage of.

Figure 4-8 shows the proposed charging plan for the deposition tunnels. The charge concentration of the emulsion explosive is adjusted to each individual row in the blast round. Maximum co-operating charge is expected to be up to 5 kg. Each hole is time-separated by the use of electronic detonators. Simultaneously detonated contour holes are preferred to reduce the excavation damaged zone.

Excavation sequence

The main proposal is full face excavation. Alternatives are shown in Figure 4-9. It may be possible to use pilot-and-slashing, (a smaller pilot drift is excavated, which is later enlarged) provided the blasting of the helper and contour holes are executed directly after the pilot round. Another alternative would be to blast a gallery and bench. Due to the small cross section of the tunnel, the slashing or benching should not be delayed more than one round after the pilot or the gallery round. In case specially designed drilling equipment with rigid booms and feeder beams are used, full-face excavation would be sufficient.



Type	Charge density [g/cm ³]	Charges length [m]	Uncharged length [m]
A	1,100	4.0	0.5
B	1,100	3.7	0.8
C	350	4.0	0.5
D	200	4.0	0.5

All holes are primed with 22 · 150 mm primer.

Total amount of explosives per round (Site Sensitized Emulsion): 257 kg. Specific charge: 2.2 kg/m³.

Figure 4-8. Charge plan for the deposition tunnel using an emulsion explosive which is sensitized in conjunction with charging.

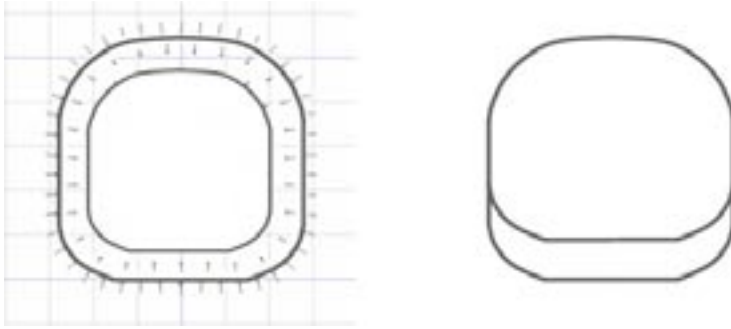


Figure 4-9. Options for the excavation sequence. Pilot-and-slashing (left), gallery and bench (right).

4.2.3 Excavation of access ramp, main tunnels and central area

The reference design assumes standard good Drill & Blast practice as described in Section 4.2.1 for the access ramp, main- and transport- and pilot tunnels and this is a reasonable approach. Specific drilling would be around 2.5 drillmeters/m³ for the main tunnel and 3.5 drillmeters/m³ for the pilot and transport tunnel. Specific charging (powder factor) is around 1.8 kg of explosives/m³ for the main tunnel and around 2.3 kg/m³ for the pilot and transport tunnel. Due to the size of the main tunnel, pilot-and-slashing would be even simpler than in the deposition tunnels if deemed necessary.

4.2.4 Future developments of Drill & Blast technology

Future development of technology is very much market driven by the customers and the suppliers. Discussions with major suppliers like Atlas Copco, Sandvik Tamrock for drilling and Dyno Nobel, Kimit for explosives and charging and experience from customers as LKAB (mining), Skanska (construction) and researchers (SweBrec) has taken place and the general summaries are:

Drilling

No major breakthroughs are expected during the next 10-year period. However, minor developments will be made successively.

- Drill machines will be more powerful which requires the different parts of the rig (booms, feeders, carriers) to be more rigid and heavier to withstand the impact of the stronger machines. An increase in mechanical bending etc should be expected as a result which requires development on positioning and directional systems.
- Booms will be developed with an even more developed and integrated transducer system for sensing of movements and positions in booms and beams.
- The accuracy during positioning and alignment, and during logging, will improve considerably.
- Parallel alignment will be controlled from a computer.
- Navigation of the rig will take place through automatic measurement of the rig environment.
- The rig will be included in an information network.
- Intelligent profile measurements will be carried out from the rig in order to keep track of possible protruding rock that may affect the drilling of the next round.

- The whole layout of the tunnel system will be stored in the rig computer.
- Surveying information will be passed on to the scaling and shotcreting equipment.
- Borehole information will be passed on to the charging equipment.
- A steering pipe at the front of the feeder beam will increase collaring precision and facilitate automated collaring.

It is also foreseen that remote control of the drilling operation would not be an expected outcome of the next 10 years of development.

Explosives and detonators

- Nitrate-based explosives will dominate the market and be developed according to different needs.
- Stable emulsions down to densities of 0.4–0.5 kg/l will be available.
- Gassing of the explosive will increase to lower the costs and to be able to vary the strength of the explosive.
- Less water in emulsions will provide better detonation stability but also more sensitive explosives.
- Electronic detonators are commonly in use while they become cheaper and easier to handle.

Charging technique

The future points towards the development of an automated charging operation, where increased safety, productivity quality and less spillage are key words. The automated charging technique may build on a concept where the charging operation is performed together with the drilling operation using the same rig.

Challenges are to develop a complete automated system that connects the detonators to each other or systems that handles “troublesome” drill holes.

4.3 Use of Tunnel Boring Machine

The first Tunnel Boring Machine (TBM) constructed was used to excavate a tunnel below the Thames at London 1825–1842. The first modern TBM as a mechanical rotary excavator was developed in the 1950’s. Since then thousands of km of tunnels have been excavated worldwide in a range of rock conditions and in diameters from around 1.5 m up to 12 m. A few tunnels have been excavated in Sweden by TBM, like the Kymmen and Klippen hydropower tunnels and the “Ormen Långe” drainage water tunnel in Stockholm. At Äspö Hard Rock Laboratory 409 m, partially as a decline, of the facility was excavated using a 5 m diameter TBM. In spite of the very short tunnel an impressive best week advance rate of 49 m was reached with best day advance as 15 m.

This Section 4.3 contains a description how a TBM may be used at a KBS-3 repository starting with a general description of the technology and applicability. The main focus of this study has been excavation of the deposition tunnels, where novel conceptual designs have been developed within this project.

4.3.1 General description of TBM technology for the SKB applications

As explained before the idea is to produce kerfs and chips by the cutters. The cutters are mounted on a cutterhead that rotates. Grippers are pressed against the rock and hydraulic cylinders are used to generate the thrust, Figure 4-10. The chips produced are mucked and delivered to a belt-conveyor, which transfers the muck from inside the cutterhead back through the main beam to the rear of the machine. Here the muck is transferred to the back-up muck handling equipment for removal from the tunnel. We assume that standard trucks are used for haulage from the TBM to the skip station.

To avoid the need to work ahead of the tunnel face, like in Figure 2-14, the machine is designed so cutters can be replaced from the back of the cutterhead. The machine would be equipped with 19" face and gage cutters (483 mm), where individual thrust per cutter is up to 311 kN. The "main beam" TBM configuration proposed allows for continuous steering capability throughout the actual boring cycle that is an important feature for curve boring. An operator may perform minute, immediate steering corrections to maintain line and grade based on the computerized guidance system readouts.

The main beam TBM configuration consists of three main structural components:

- the cutterhead with cutters,
- the cutterhead support with the main beam and
- the gripper and thrust assembly.

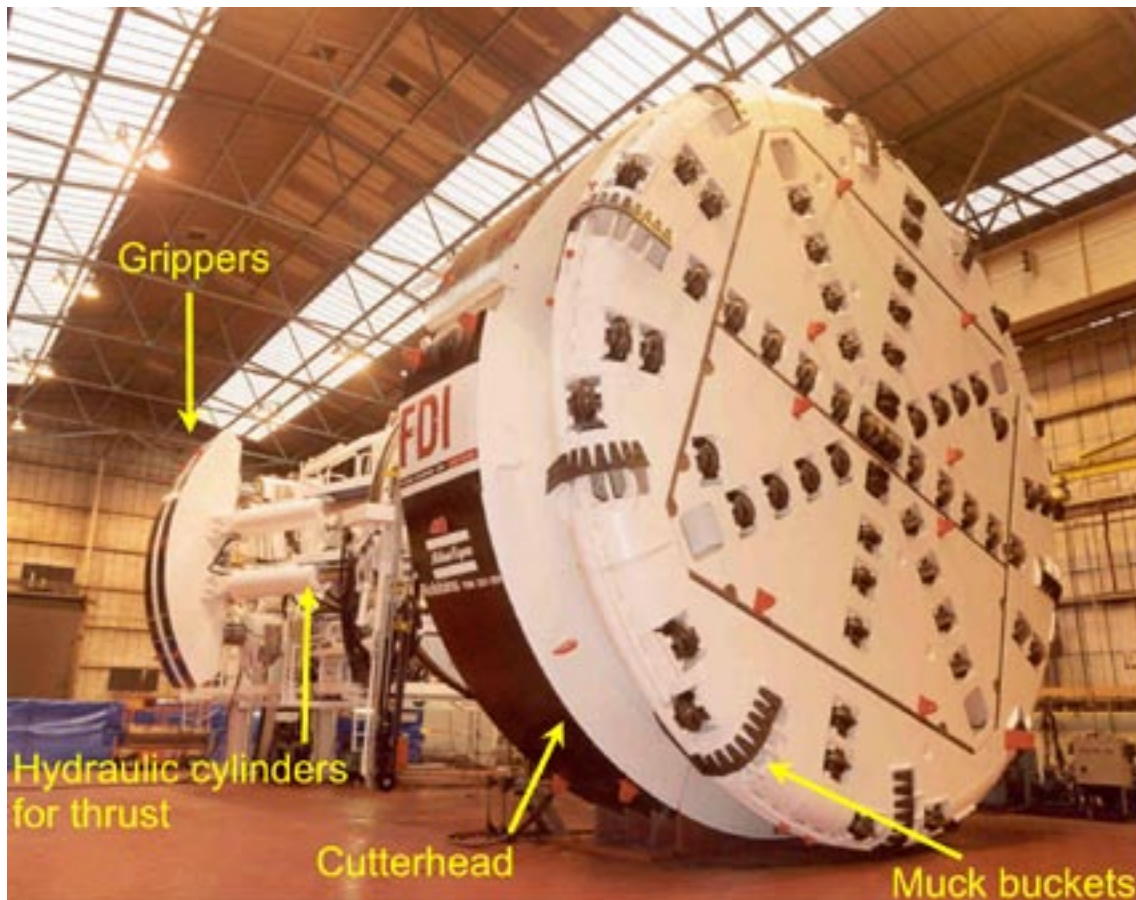


Figure 4-10. Picture of a Robbins manufactured "main beam" TBM. The grippers are pushed against the wall with hydraulic cylinders generating the necessary thrust. The cutterhead and cutters rotate to produce the chips. The chips are mucked and delivered to a conveyor belt.

The standard design for a 6.3 m diameter main beam TBM is shown in Figure 4-11, with length of 22 m excluding the backup systems. Stroke length is here 1.8 m, that is boring is for 1.8 m before regripping.

The main components include electric drive system, main bearing and ring gear assembly, bearing and seal lubrication systems, dust shield, cutterhead water spray, muck buckets and machine conveyor, operator station and controls, and electrical and hydraulic equipment. Together these components form an integral machine capable of ground control, hard rock tunnel excavation and muck transport to a haulage system.

A steel dust curtain maintains a seal between the bored tunnel and the cutterhead support in the area directly behind the head. Dust generated at the face is trapped in the cutterhead area and sucked out from muck dump area through a vent duct in the main beam. The ventilation ducting extends from the TBM to the dust scrubber. Dust control is enhanced by a network of nozzles that spray a water pattern from the cutterhead on to on the rock face. There are additional nozzles in the muck chute/TBM conveyor area to further dampen dust circulation. The water mist assists in preventing the dust particles from becoming airborne.

An expandable roof support is maintained in full contact with the ground to minimize vibration. The roof support is hydraulically adjustable and is located as close to the cutterhead as possible. The roof support structure extends past the rear of the cutterhead support to give a protected area for workers on the TBM. The aft end of the roof support is fitted with “fingers” which, when properly supported with roof bolts and/or ring beams, act to give further protection to ground support installation workers.

The main beam acts as a lever for steering, using the cutterhead support shoe as a pivot for vertical steering, and the side supports as a pivot for horizontal steering. The design of the main beam and gripper carrier way, in combination with the various cylinders, provides for precise control of steering. For horizontal steering, the barrel of the gripper cylinder is moved sideways over the continuously pressurized gripper rods and pistons.

For vertical steering, the torque cylinders, which are mounted between the gripper carrier and the main beam, are actuated. When the torque cylinders are extended, the main beam rises relative to the gripper cylinder and the TBM will steer down. Conversely, when the torque cylinders are retracted, the main beam is lowered relative to the gripper cylinder and the TBM will steer up.

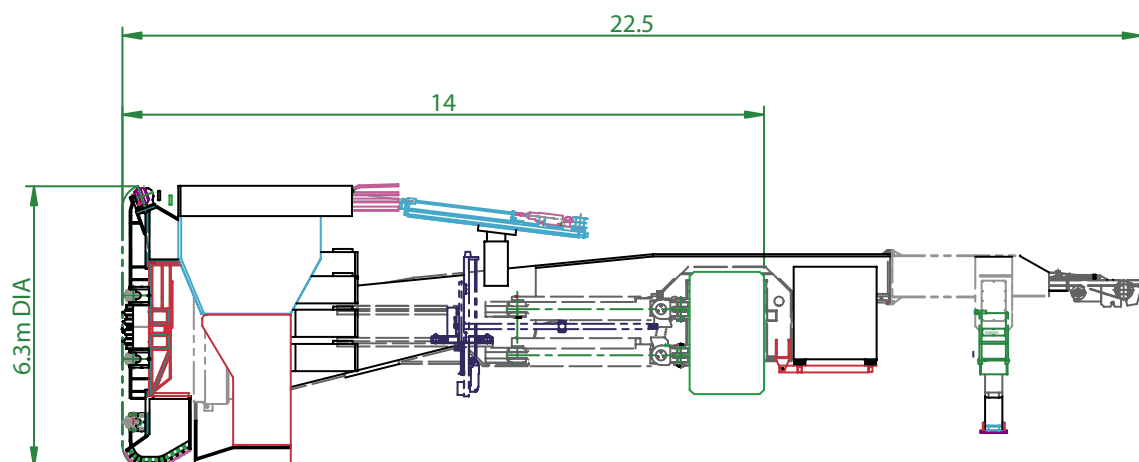


Figure 4-11. General design for a TBM, diameter 6.3 m. Stroke is 1.8 m. The machine is equipped with roof and probe drills ahead of the grippers. The operator cabin is located on the TBM gripper assembly.

The cutterhead is driven by variable frequency electric 315kW water-cooled drive units. Each drive unit consists of a motor, and a speed reducer with output pinion. The drive motors mount directly to the speed reducer assembly which allows installation and removal as a modular unit. The speed reducers are planetary-type, two-stage in-line units. Each reducer output shaft is fitted with a drive pinion which engages with the ring gear. Two drive unit types have been investigated for the SKB-applications: Variable Frequency Drive (VFD) and hydraulic drive, each providing a cutterhead speed range, the VFD being the recommended solution.

Continuous monitoring and recording of thrust, torque, cutterhead speed, advance rate, main drive motor amperage, etc. is possible.

Industrial grade equipment will be provided for the TBM hydraulic system. Major circuits power the thrust/steering cylinders, gripper cylinders and forward stabilizer. Double ended electric motors will drive both high pressure and low pressure, high volume pumps. Components will be manifold mounted, using cartridge style configurations to minimize the number of plumbing connections, compact installation size, lower system pressure losses, and will simplify troubleshooting and maintenance problems. The hydraulic power unit will be located on the back-up system.

Roof and probe drill systems will be available on the machine. Shotcreting and grouting equipment will be included.

4.3.2 Excavation of deposition tunnels

Machine design

It is evident that the standard TBM (Figure 4-11) is not very well adapted to short multiple-entry projects, like for a geological repository. A conceptual design for a TBM for deposition tunnels has been developed by Robbins as a task within this project.

Because there are a large number of very short tunnels (approximately 300 m in the reference design), the main challenge for efficient operation is to be able to set up the tunnelling system rapidly when commencing a deposition tunnel, and to be able to remove the system and move it to the next tunnel with minimum loss of time. It is understood that groups of around eight to ten tunnel will be constructed at a time in one branch of the repository, after which a similar number will be constructed in the other branch, and so on. There are thus two types of tunnelling system move to be addressed. First, transport between the two deposition area branches, and secondly, between a group of deposition tunnels. The time constraints for the inter-branch move are not as critical as those between the deposition tunnels.

Given the large number of deposition tunnels that will have to be constructed, it is prudent to design a TBM that is as short as possible, to minimize excavation and concreting requirements for the start-up or launching chambers. A short TBM will also be easier to manoeuvre and handle in the confined space underground.

Figure 4-12 shows a concept suitable for the deposition tunnels. The approximately 10.4 m long, 6.3 m diameter TBM uses a very compact standard “core” element comprising the main bearing, a powerful drive system, and supporting structure, around which the guiding control shoes are attached. The core envelope is small enough that there is room to place the rod ends of the TBM propelling cylinders between the periphery of the cutterhead support and the side support shoes, which provide lateral stability and guidance. Normally the propel cylinders are attached to the rear face of the cutterhead support or to brackets attached to the main beam behind the drive motors. Relocation of the cylinders shortens the machine by the length of the cutterhead support.

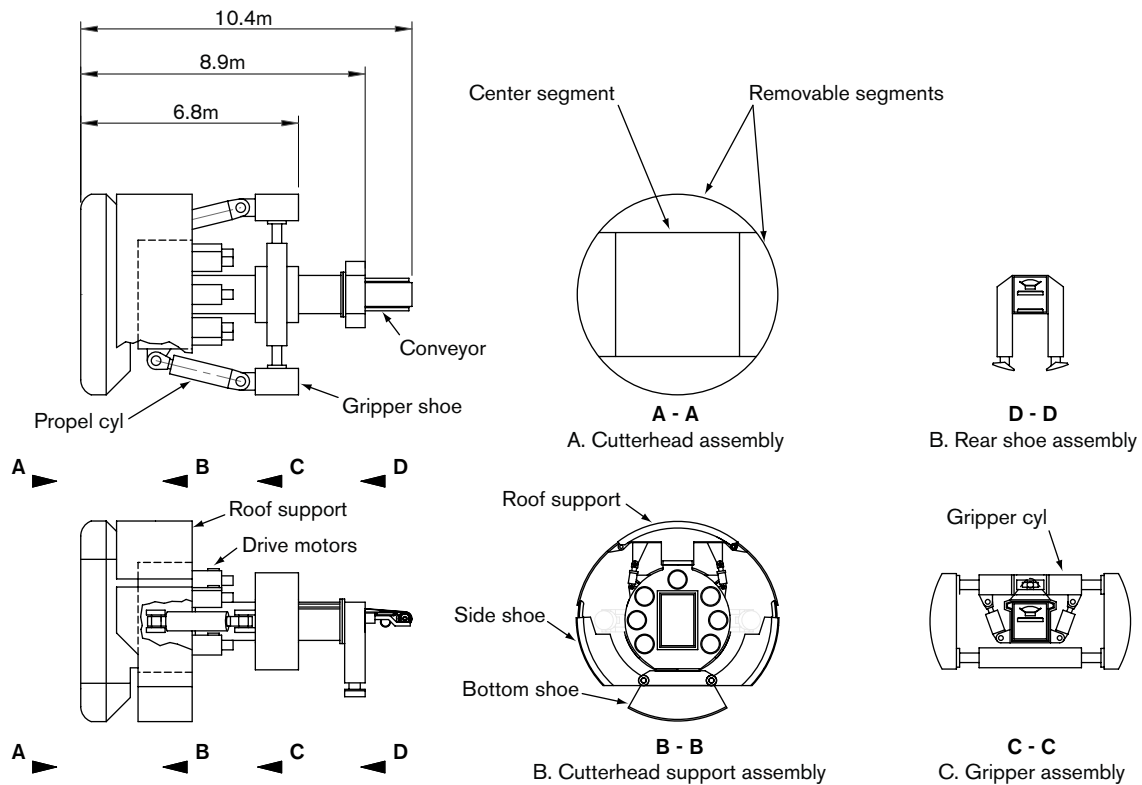


Figure 4-12. Conceptual design of the core elements of the TBM for the deposition tunnels.

The small core element also facilitates transport of the machine, as by removing the guide shoes (roof support, side shoes, and front shoe) plus the removal of outer cutterhead segments, and, if necessary the gripper shoes, the maximum machine cross section is reduced to a square 3.9 by 3.9 m. The core machine leaves intact most of the electrical and hydraulic systems, and disassembly and reassembly can be performed relatively quickly. The total weight of the TBM is around 450 tonnes and the weight of the core elements in Figure 4-12 around 225 tonnes. 42 cutters are used so total thrust is up to 13 MN.

A boring stroke of 1 meter has been chosen for the “core TBM” as reducing the stroke of hydraulic cylinders would result in an overall length decrease of twice the amount of the reduction. There is a decrease in overall advance rate because of the higher frequency and number of gripping cycles, but high advance rates are not necessarily required for the deposition tunnels compared to fast track and typical civil engineering projects. If a simplified muck transport system is used (like employing standard dump trucks) the expected advance rate should be well matched for the system.

No mechanized rock bolting equipment can be furnished immediately behind the cutter head support, but narrow platforms can be provided on top of propel cylinders to enable limited rock bolt installation using handheld drills. A more extensive rock support operation may take place immediately behind the TBM if additional rock should be deemed necessary.

Because of the shorter machine, the area behind the cutterhead support is no longer available for the installation of a drill having the required peripheral coverage. The drills will instead be mounted on a ring mechanism mounted on the front end of the back-up gantry. The gantry conveyor passes through the centre of the ring. With this arrangement, 360-degree coverage of the tunnel is possible, with the exception of those areas blocked by the TBM structure, but this is typically overcome by tilting the drill booms to reach the theoretical endpoint of the probe or drill hole. The drill is mounted on a mechanism

allowing its orientation to be adjusted relative to the tunnel axis. The drill can be angled from parallel to the axis to approximately 7 degrees from the axis. By using it parallel to the axis, drilling can be performed through the cutterhead support and cutterhead and thus directly into the tunnel face, at a few set locations. The 7-degree angle is the minimum necessary to satisfactorily break into the peripheral tunnel wall.

Figure 4-13 depicts the proposed back-up scheme, designed to be simple and reasonably easy to manoeuvre in and out of the deposition tunnel bored perpendicular to the main tunnel.

The main machine controls are located in an enclosed operator's cabin on a single gantry immediately behind, and towed by, the TBM. The gantry is connected to the core of the TBM through a tow cable and an umbilical cable and hose system. Muck is discharged from the rear of the TBM onto a second conveyor supported by the gantry, which elevates the muck for discharge into the dump truck. The conveyor tail section is elongated over the truck location enabling the truck to be efficiently filled from front to rear.

Other components located on the gantry include the main electrical switchgear cabinets, variable frequency drive system, the hydraulic power unit (reservoir, pumps, etc.), the probe drill hydraulic power supply, grout mixing and pumping equipment. Because of the short tunnels, and to reduce the size of the gantry, the high voltage transformers will be located in the main tunnel, close to the deposition tunnel entrance. If possible 1,000 V electric cutterhead drive motors can be used, to reduce the tunnel cable and associated reels, although this depends on the development of 1,000 V variable frequency drives in the coming years.

The major components of the tunnel ventilation and dust cleaning system are likewise planned to be located in the main access tunnel, rather than on the back-up system as is necessary on long tunnels. Rigid ventilation ducting will be extended at the gantry as tunnelling proceeds. Ventilation will be based on suction systems.

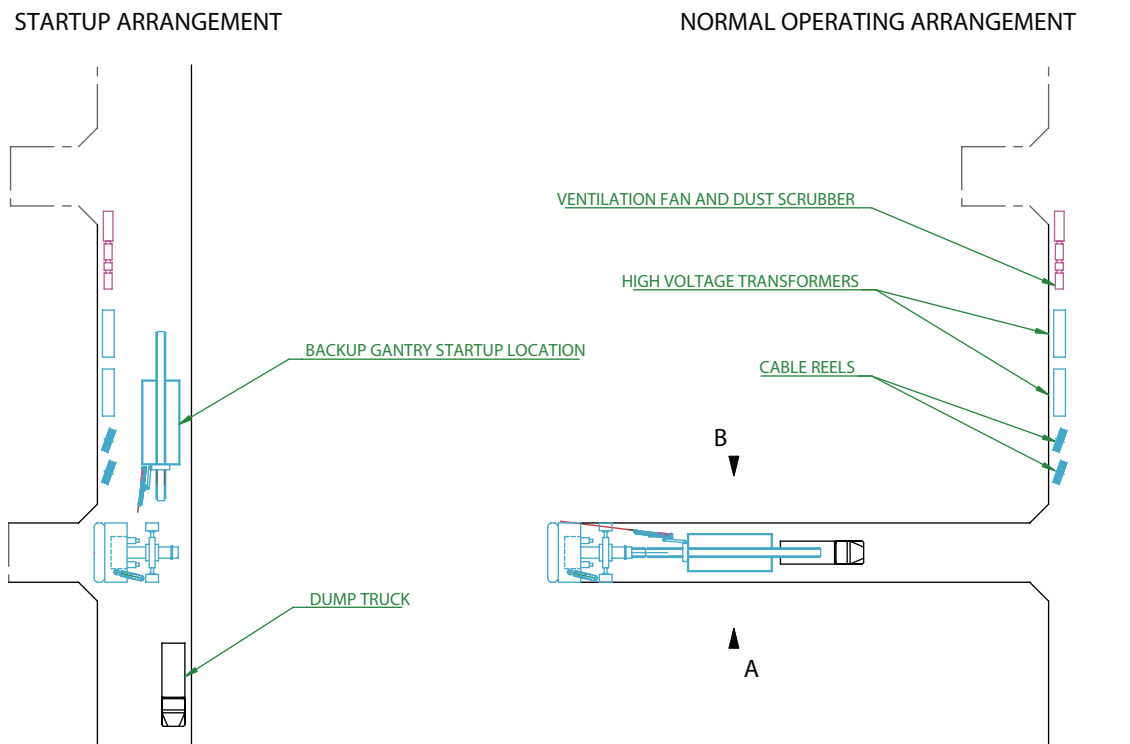


Figure 4-13. Back-up arrangements.

Move and operation of the machine

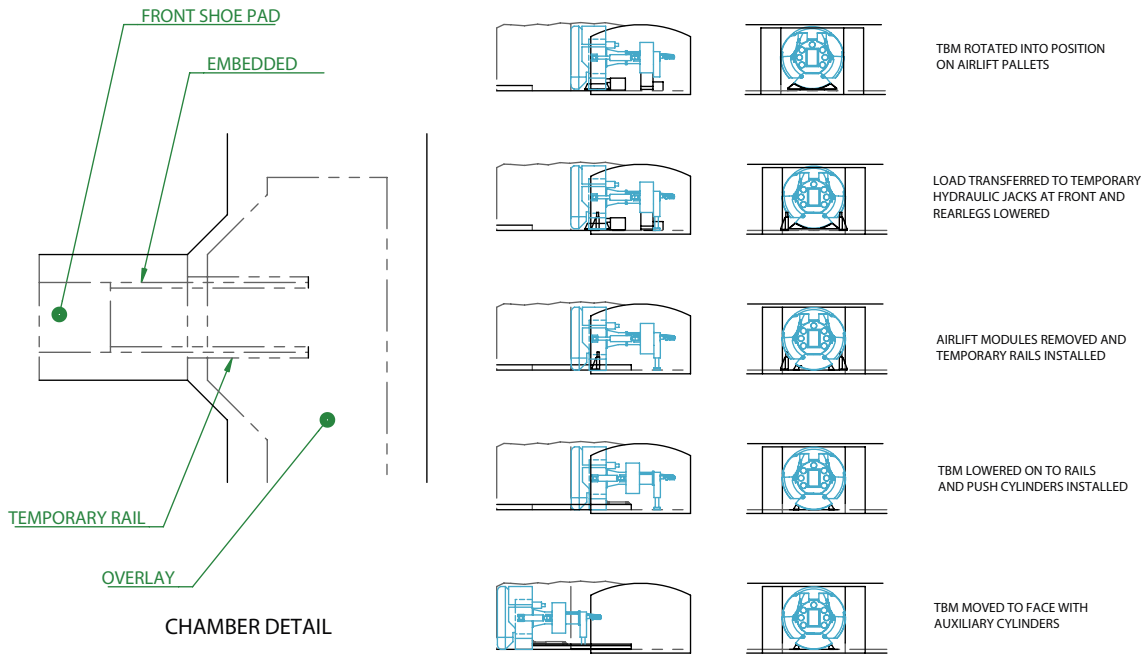
A number of options exist and have been studied for moving the back-up gantry along the tunnel depending on what type of invert is selected. The final selection of invert type may depend on the type of roadbed required for the machine for drilling the vertical canister deposition holes and the canister deposition vehicle. SKB preference for transporting the excavated muck to the mine skip would be by standard 12 m³ dump truck. For such a truck a flat roadbed is necessary. This can be constructed by creating an invert from muck excavated by the TBM. A portion of the excavated muck will be diverted in front of the gantry, and distributed and levelled by a spreader bar system towed by the TBM.

The launch of the cutterhead from the main tunnel to the deposition tunnel is shown in Figure 4-14 for two configurations. At the entrance of each deposition tunnel a launch chamber is excavated by Drill & Blast.

It is assumed that the TBM is transported and rotated into position outside the launch chamber on the flat concrete floor, using an airlift system (described on Page 77 and in reference /AeroGo, 2004/) The TBM is being launched in a chamber that is offset 850 mm below the top-level of the main tunnel. The cutterhead segment and the front shoe are placed in the launch chamber before the TBM is positioned at the chamber entrance. The TBM less segment and shoe are moved into place on the airlift system. The front end weight of the TBM is transferred to Hilman rollers moving on a short steel track embedded in the concrete floor. The Hilman rollers can support a great weight in compact space and have very low friction (see description Page 77 and in reference /Hilman, 2004/). The extended TBM rear legs support the weight of the aft section of the TBM. The machine is jacked forward by hydraulic cylinders that engage with the roller track until it is lined up with the front shoe, which is bolted in place. Jacks are then used to transfer the weight on to the front shoe. The machine is then jacked forward, using the TBM auxiliary cylinders built in to the front shoe. Cylinders react against steel segments placed in the chamber invert as the machine advances. When the machine is lined up with the cutterhead segment, it is bolted on to the cutterhead, and then the machine is jacked forward until the gripper shoes can be lined up with the concreted wall of the launch chamber. The machine can now commence boring. A temporary conveyor system is used to carry the excavated muck from the TBM to a dump truck located at the tunnel entrance area. The electrical and hydraulic supplies are fed from the back-up gantry, which is placed close as practical to the tunnel entrance area. When there is sufficient clearance behind the TBM, the back-up gantry is moved through the launch chamber, and tow cables attached. The maximum length of the umbilical cable and hose system is estimated to be about 15 m.

After the deposition tunnel has been excavated the launch procedure is reversed, Figure 4-15. The procedure is essentially the reverse of the launch procedure described above. As the end of the tunnel is approached, the lower outer cutterhead segment, and then the front shoe are unbolted. The machine has now to be pulled back over the steel segments installed during the launching operation. This is accomplished by a using Hilman rollers and temporary rail system running the full length of the segments. Hydraulic cylinders attached to the track supply the pulling force. The TBM is pulled back into the main tunnel where it is transferred on to the airlift system that is used to move the TBM from one deposition tunnel to the next, a stretch of about 40 m.³

³ If the deposition tunnels would be angled 45 degrees to the main tunnel design of the TBM and move between tunnels are simplified. The back-up system, complete with high voltage transformers, and the ventilation extension system and dust scrubber, can be located immediately on line behind the TBM requiring an entrance curve of a moderate 30-meter radius. Part of the back-up system at start-up will be in the main tunnel. The most efficient system will be rail-borne on track extending along the main tunnel as part of a comprehensive scheme. This provides a means of transporting the TBM system in its tunnelling configuration, and would allow the umbilical electric/hydraulic package to remain connected at all times.



NOTE: RECOVERY PROCEDURE IS THE REVERSE OF THE SEQUENCE

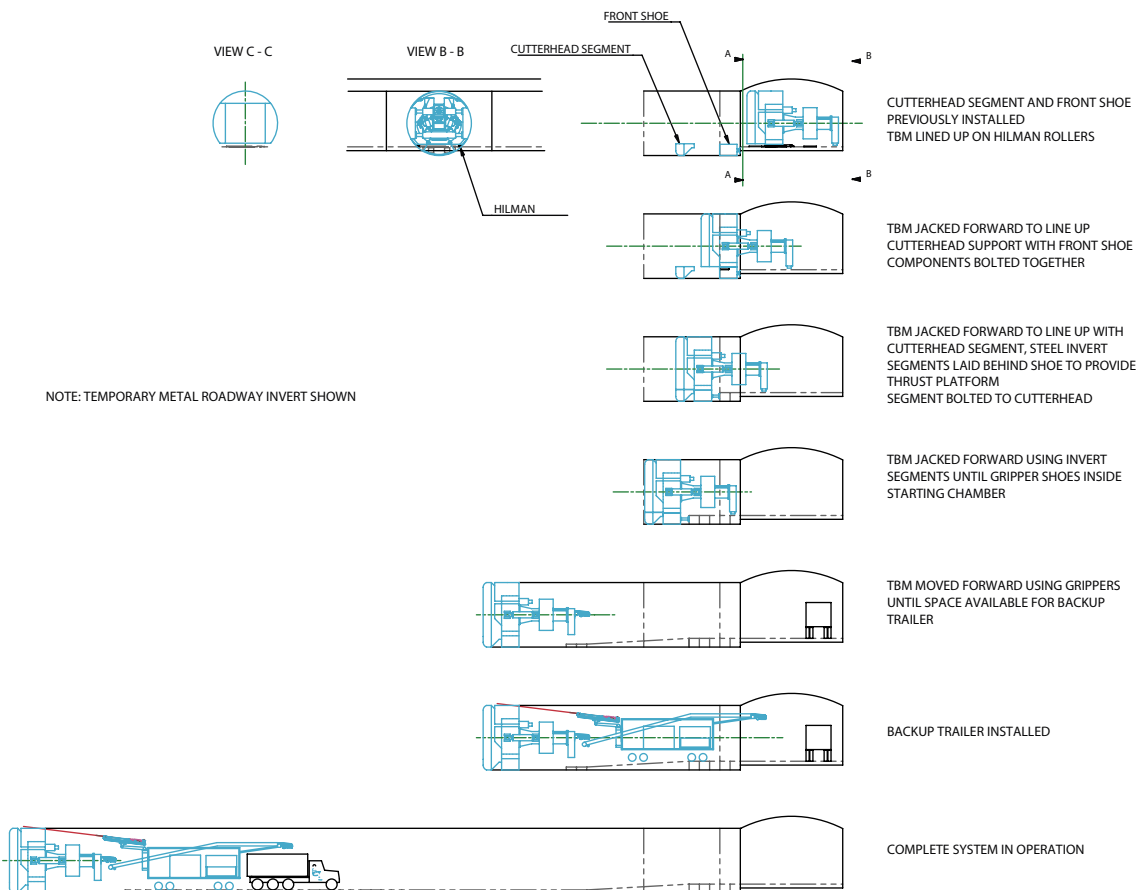


Figure 4-14. Top: Launch of complete TBM. Bottom: Launch of partially disassembled TBM.

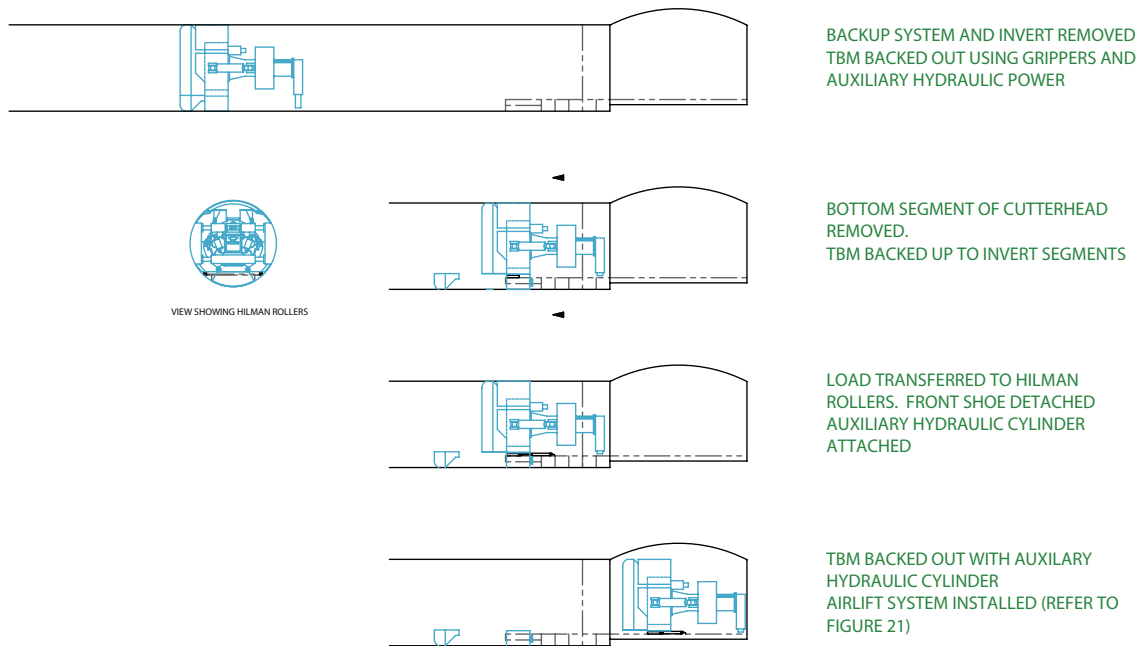


Figure 4-15. Removal of the TBM.

The airlift system /AeroGo, 2004/ reviewed is a well-established method of moving heavy loads with comparative ease. The lift is provided by pressurized air trapped under a rigid load carrying structure. It is impossible to completely seal in the air, but a specially designed flexible air caster system (toroidal cushion) reduces air losses to practical levels. Current designs operate with pressures up to 3.5 bar, although pressures exceeding 5 bar have been used. The units are manufactured as modules that can be grouped together to provide great lifting capacity.

Once the system is pressurized the load is in effect floating, and two-dimensional control is necessary. Special drive units are attached either to the lift system or to the load itself. The units are equipped with spring loaded traction wheels that can be oriented in any direction. By using two of these units spread a reasonable distance apart, full directional control of the TBM load is obtained, including the ability to rotate the TBM. The 1% main tunnel slope should not create special moving problems. This system has been previously used to handle various type and large diameter TBMs. On the Channel Tunnel a shield machine was turned around after completing one tunnel so that it could commence boring in the opposite direction and also certain Japanese projects did use Aero-Go. The condition of the surface on which the air modules float is critical. Even smooth concrete that is not sealed allows leakage. For use on average concrete road surfaces, it is recommended to cover the move path with heavy-duty polyethylene sheeting. The overlay sheet is strong enough to withstand the traction force of the drive unit wheels. According to the supplier, 400 tonnes Boeing aircraft parts are being moved using the airlift system with overlay sheeting at the Seattle factory. Figure 4-16 shows how the airlift system would be used to move the TBM between the deposition tunnels and indicating the overlay in place.

The Hilman roller used to move launch and recover the TBM is a very compact device for moving heavy loads with very low friction. It basically consists of a number of cylindrical rollers chained together in an endless loop passing over and under a loading plate. The load is transmitted from the plate sitting on top of a group of rollers to the ground. As the unit moves forward the rollers gradually rotate around the plate under the action of the chain. Because there are no axles or bearings in the unit, very high load carrying capacity

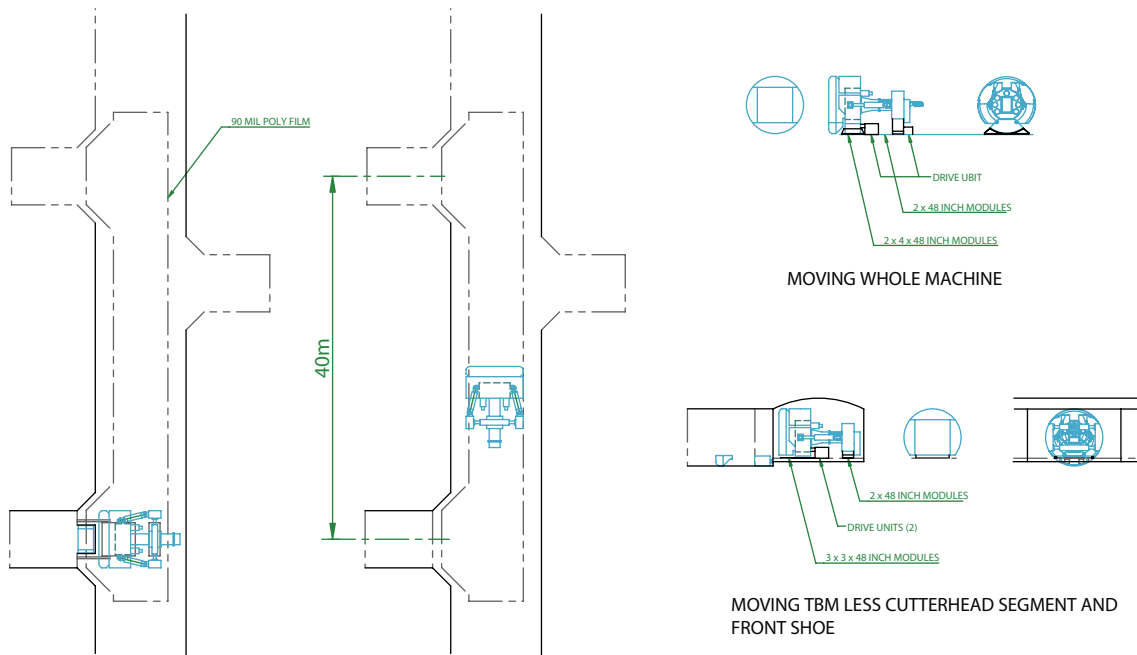


Figure 4-16. Moving the TBM from a deposition tunnel to the next using an airlift system.

is available in a small space. A unit capable of carrying of carrying 200 tonnes measures approximately 500 mm long, 250 mm wide and 170 mm high. The roller contact loads are too high to run on a concrete surface, so a steel track of some kind is necessary, such as channel or plate, to spread the load. Channel provides guidance to the rollers. The higher the strength of steel used in the track the better, but for limited usage, standard mild steel is acceptable. Ideally the track is embedded in the concrete floor, or at least secured in some way. Allowance has to be made for potential misalignment between the rollers and the track to avoid component damage. The effective friction factor (to start the load rolling) is approximately 5 percent, meaning that to move a weight of around 360 tonnes a propelling force of 180 kN is required, easily obtained with a hydraulic ram. The ram can be mechanically engaged with the track to provide thrust reaction. On an incline slope, additional force to overcome gravity would be necessary. For moving unit on a 1% upgrade main tunnel requires a propel force of approximately 40 kN. The Hilman roller is best used for short straight moves, where space for supporting means is limited, such as when entering or exiting the TBM launch chamber. Rotating the load in the horizontal plane is possible but not easy. Swivelling roller units can be used in conjunction with a special track arrangement. Longer moves pose more difficulties in economically installing and guaranteeing a straight and level track. For these reasons, the airlift system appears better for moving the TBM between the deposition tunnel locations, as there is plenty of room for the airlift modules in the main roadway, and the free floating characteristic allows easily controllable linear and rotational movement.

As described before, repository operation is split in a “Construction Area” and a “Deposition Area”. The construction and deposition areas are swapped at regular intervals. After a set of deposition tunnels have been bored, the TBM will be moved to the other branch. This will involve traversing distances of up to 2,600 m. Because TBMs are very complex machines, incorporating a large number of electrical and hydraulic connections, it is always best to keep at least the core assembly together to avoid unnecessary tear down and reassembly time. The most practical non-rail system, tolerant to road surface imperfection, appears to be using tracked type carriers, as depicted in Figure 4-17. The supplier /Caterpillar, 2004/ supplies undercarriage units, as independent components,

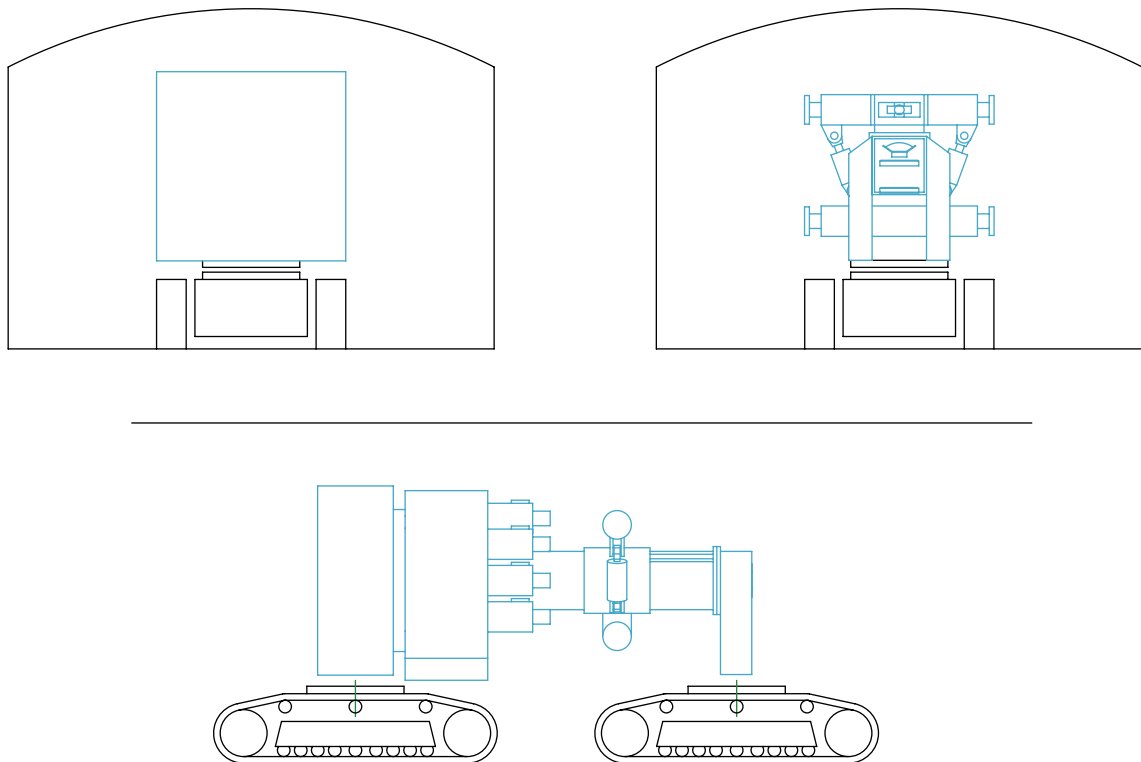


Figure 4-17. Move of the TBM from “Construction Area” to “Deposition Area” using a tracked type carrier.

complete with hydraulic drives that have sufficient capacity to support the TBM. Two units, one at the front and one at the rear using differential track steering can easily guide the load. The undercarriage units are connected to the TBM via swivel bearings, which allow them to rotate as the load is steered. The tracks would be equipped with flat plates to avoid gouging the roadway surface. Urethane rubber pads are used on smaller machines, but do not presently have sufficient load capacity for adequate durability, according to information from the supplier. There should be very little skidding effect in the main tunnel curves – even at 50 m radius curves. The track suspensions can accommodate any crowning of the road surface. The travel speed will be approximately 1.5 km/hr.

Typical performance

Based on the geological conditions assumed, typical advance rate is around 2.1–2.7 m per machine hour with a daily advance of 12–15 m and weekly advance of 60–75 m. The estimate is based on a comparative low, average utilization factor of 40% during boring and not accounting for probing and/or grouting operations, and no major rock support and/or stabilization work. It is further assumed that normal TBM maintenance, minor repairs and cutter changes will take place during the night shifts. Around 31–34 workdays are needed to bore a full deposition tunnel, around 265 m, not accounting for time to prepare the launch chamber, Table 4-6.

Table 4-6. Anticipated work cycle to bore a full deposition tunnel.

Activity	Average number of workdays
Moving TBM equipment from previously completed tunnel to prepared portal area 40 m away by use of airlift system	1 day
Turning and setup of TBM in launch chamber. Work includes relocation of transformers, scrubber and fan unit, utilities, etc. Connection of hoses, cables, TBM system checks, cutter changes and repairs as required	5–8 days
Boring of approximately 20 m with limited back-up and mucking capacity	3 days
Boring of full deposition tunnel (265 m) based on 13 m/day average	20 days
Moving of equipment back to portal area. Disconnect hoses/cables and prepare for move TBM system check	2 days
Total cycle for boring one deposition tunnel	31–34 workdays

4.3.3 Excavation of pilot and main tunnels

While the TBM for deposition tunnels would be a novel design, a TBM for pilot tunnels (and main tunnels) can rely on standard designs, however adapted to the particular needs. The main idea with TBM for pilot tunnels is quick access to the full extent of the site for detailed investigations of the bedrock conditions. Pilot tunnels, around 5 m in diameter will later be slashed to the full area needed for the main tunnels (10 · 7 m).

Machine design

Based on the diameter 5 m, 35 cutters are used each designed for cutter load of 311 kN, so total maximum thrust is close to 11 MN. Total weight of the TBM is approximately 300 tonnes.

The specific design of the TBM is dependent on the curve radius decided: From point of flexibility, it is advantageous that small radius can be bored. It is confirmed that R=50 m is feasible, but is not recommended due to very low advance rate etc. The designer recommends that the pilot drive tunnel be kept to radii not less than 150 m for tunnel efficiency and speed of operation.

Operation

Both railed and rubber tyre mucking can be used, but mucking by conveyor belts having less than 300 m radii curves is not recommended based on current technology.

At this study stage it appears that a railed type mucking system provides the simplest technical solution with no interruption by drill and blast operations.

The most efficient mucking appears to be using locomotives and railed muck systems and intermediate installation of switches to provide train passing at certain tunnel intervals. At tight turn radii at 150 m it will be necessary to keep overhead back-up conveyors (sliding on top of the back-up gantry), to individual lengths of 15 m or less. The muck trains may comprise of locomotives and rolling stock similar to the Mühlhäuser-type muck cars or the Hägglund shuttle train system depending upon pull and grade.

Mucking by using a rubber-tyre muck system using mining trucks or regular 12 m³ dump trucks may be considered for use depending upon availability and the overall issue of costs. Since the rubber tyre vehicles require flat invert area, part of the TBM muck will normally

be diverted to the invert area immediately behind the TBM conveyor. An invert spreader system towed by TBM can provide a level driveway and to the appropriate invert elevation. Since there are no places for the vehicles to pass or turn in a 5 m diameter tunnel, it will be necessary to drill and blast turnout niches in the bored tunnel at approximately 300 m intervals. This will give the incoming trucks a place to turn around for backing in under the loading conveyor and provide passing stations for incoming and outgoing vehicles. The niches will disappear into the main tunnel cross section when slashing the pilot tunnel drive to 10 m by 7 m main tunnel.

Typical performance

Based on the assumed geological conditions, typical advance rate would be around 2.7–3.4 m per machine hours with daily advance of 15–20 m and weekly advance of 75–100 m. Curve boring in general will require more time and contractor expertise compared to boring straight alignments: The tighter the curve the slower the overall advance rate. It is anticipated that utilization factor will be dropping from around 45% to about 15% on the case R=50 m curves are bored due to reduced thrust levels, surveying requirements and special mucking procedures. Daily advance in 50 m curves is thus anticipated to be 5 m or possibly even less. No probing or grouting operations and no major rock support and/or stabilization work are included in estimate and it is further assumed that normal TBM maintenance, minor repairs and cutter changes will take place during the night shifts.

4.3.4 Excavation of access ramp

SKB has previously studied options and decided alternatives for access from surface down to the deposition areas /Bäckblom et al. 2003/. In the study the option of using TBM for the access ramp was left open stating that TBM would be an option if the ramp is reasonably straight and the rock conditions are favourable. Ongoing site investigations at Forsmark⁴ indicate very good rock conditions and therefore a conceptual TBM access ramp alternative has been prepared, see the previous Figure 3-2, with a continuous 10% decline gradient. We assume the access ramp to be excavated with less free area than the reference design (46 m²) assuming haulage with standard trucks rather than electric.

Machine design

Based on the selected diameter 7.1 m for the access ramp, 46 cutters are used. Each cutter is designed for the cutter load of 311 kN, the total maximum thrust thus being more than 14 MN. Total weight of the TBM is approximately 600 tonnes.

As for any decline drives precautions must be taken regarding water inflow to prevent loss of life and inundation of machine and equipment. The system must be designed with drainage pumps and pipes to carry water out of tunnel to prevent water ponding at the heading, and standby generators must be provided for immediate engagement in case of electric power failure. To reduce water inflow in critical zones, the TBM can be equipped with equipment for pre-excavation probing and grouting in order to reduce or eliminate the water inflow problems. Grades from 18 degrees decline to 50 degrees incline have been bored by Robbins TBMs equipped with custom designed back-up equipment and special features.

⁴ Investigations at Oskarshamn are at an earlier stage, but similar rock conditions may prove to exist.

Operation

To ensure the most efficient TBM production it is recommended that the “spiral” curve radii be kept to 300 m or more if using a conveyor system even if conveyor belts have successfully been operated at $R=228$ m in short curves.

Railed transport can also be used, but special considerations to the 10% grade must be taken and a typical locomotive traction will not be sufficient. Rack and pinion drive have been used on decline drives, and the Swedish designed locomotive type RHS (Rapid Haulage System) used on the Klippen Hydro project provides possibly the most interesting solution: RHS locomotives have a haulage capacity about 68 m^3 of muck, meaning that each train can manage a TBM stroke length of about 1 m, and operate at a speed of about 10 km/hr fully loaded.

However we think that by usage of mining trucks or regular 12 m^3 dump trucks would be the most attractive alternative and for this alternative we assume that $R=200$ m is a reasonable minimum radius. To provide a flat road bed in the invert, part of the TBM muck will be diverted onto the tunnel floor immediately behind the TBM machine conveyor. Invert elevation and level will be controlled by a spreader-bar arrangement and in front of the short gantry type back-up equipment on skids. The system considered at this time will in concept be similar the TBM mucking system used previously in Norway at Bergen – 7.6 m diameter – and Bergen at Svartisen – 8.2 m diameter, Figure 4-18. In order to be able to turn the vehicles around, a small niche will be required at intervals which typically are based on a time study for the mucking operations. However if ramp diameter is increased from 7.1 m to 8.2 m there would be no need to excavate niches for truck turnaround. A hydraulic lift turntable may be used to turn the trucks 180 degrees around in a matter of minutes by a push-button system operated by the truck driver.

Performance

Based on the assumed geological conditions, typical advance rate would be around 1.9–2.4 m per machine hours with daily advance of 13–17 m and weekly advance of 65–85 m. No probing or grouting operations and no major rock support and/or stabilization work are included in estimate and it is further assumed that normal TBM maintenance, minor repairs and cutter changes will take place during the night shifts.

An average utilization factor ranging from 40–55% is foreseen depending on choice of mucking transport methods, a straight or curved decline.



Figure 4-18. Photo from truck that removes muck from the belt-conveyor at the TBM-project in Bergen. Photo: Courtesy Odd G Askilsrud.

4.3.5 Future developments of TBM technology

The TBM technology in hard, massive granitic gneiss formation is well proven and highly recognized construction method. The main beam and open gripper TBM type proposed for the SKB applications is known as “the workhorse of the industry” and current machines are based on years of experience.

There is continuous programs and feed back from actual field applications to improve efficiency and system utilization of TBM systems. This includes improvements in cutter steel technology, wear resistance of materials, gears and bearings, drive systems like Variable Frequency Drive, conveyor and belt developments, mucking, intelligent controls systems, electronics, and automatic guidance systems. There have been small, but important steps taken over the last 10 years. There is no doubt that improvements will continue in the next years to come.

The developments in TBM technology are very much driven by the needs of the market. Multiple-entry, short tunnelling using TBM would always be a very small niche market with few actors and these actors would have to specify and pay for these special developments

In general the following modifications may be available within a 10-year period.

- Remote control operation would become more typical. However, it is realized by most that it will be very difficult to replace the eyes, feel, smell and listening by an experienced tunnel person when it comes to practical machine operation and rock support installation: TV cameras and a digital output on a computer may not just be the same.
- Seismic investigation method to (reasonably) predict upcoming ground characteristics in a fast, simple and understandable way for a tunneller. If such technology is successful it should include detection of rock mass features ahead of the TBM such as mixed rock conditions, faults and water inflow characteristics in order to prepare the crew for coming events.
- Improved pre-excitation grouting techniques to provide quicker and better results with lower overall costs.
- Improved and faster rock stabilization methods in poor ground conditions.
- Step by step improvements of the overall TBM system design and technology. This may include use of higher hydraulic working pressure (this has already been included in the 6.3 m diameter deposition tunnel TBM specifications, 1,000 V cutter head drive units, improved variable frequency drive technology, and various methods and means to increase the overall utilization of the TBM system complex.
- Improved methods for transporting much by rail, rubber-tyre vehicles and conveyors. It is expected that conveyor applications will include 200 m radii curves in years to come.

4.4 Use of raise-boring technology for shafts and tunnels

Vertical communication shafts are preferably constructed by raise-drilling as this method is more efficient and safer than constructing shafts from the ground surface. Raise-drilling has been in use since the 60's and the technology has matured for over 40 years. The general technology is outlined in Section 2.3.2 but the study here has been focussed on feasibility of drilling horizontal deposition tunnels and drift.

The basic raise boring machine consists of: a derrick⁵ assembly, a hydraulic system, an electrical system, and a control system, Figure 4-19. These systems work together to develop, transmit and regulate the thrust and rotational torque needed for raise boring. The thrust and torque are transmitted to the pilot bit or reamer via the drill string,

The pilot hole is drilled with a pilot bit with three conical rollers fitted with cemented-carbide buttons, Figure 4-20. The buttons are pressed into the rock, and break the rock in a quite similar way to that described for raise boring. The rock cuttings are flushed up and out of the pilot-hole by means of one or two flushing media.

The technology shown is similar for excavation of vertical shafts and horizontal tunnels.

⁵ The name denotes an apparatus with a tackle rigged at the end of a beam for hoisting and lowering. Its name is derived from that of a famous early 17th-century hangman of Tyburn, England. He was so proficient at building scaffolds that he gave his name to the structure, a term which has become adapted to the derrick of our time, such as in oil drilling.

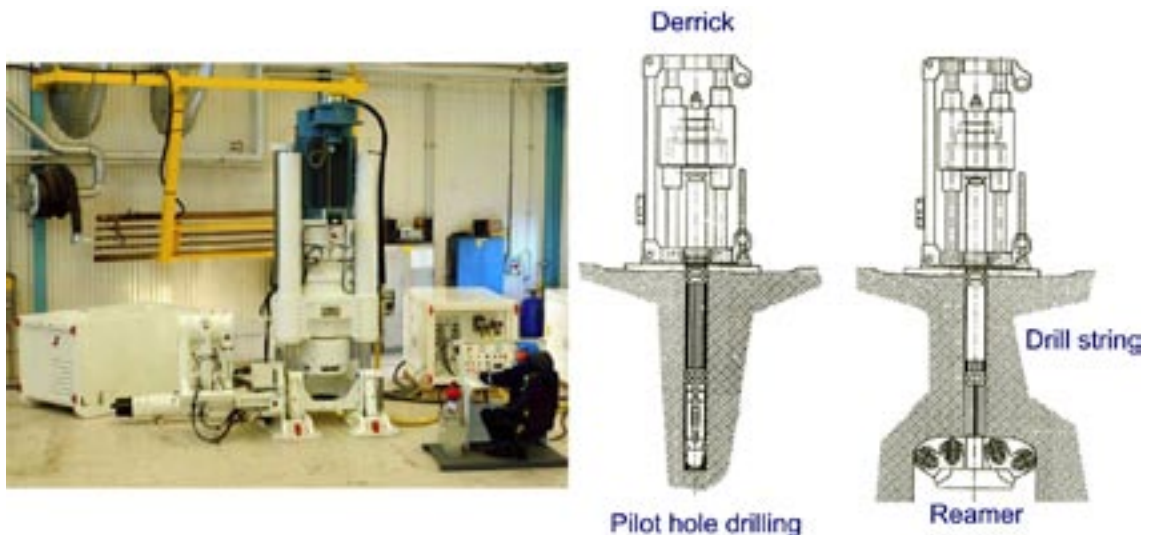


Figure 4-19. Main components of the raise-drilling equipment. Courtesy Atlas Copco.



Figure 4-20. Drill bit for the pilot hole. Courtesy Atlas Copco.

4.4.1 Excavation of shafts

The SKB reference design with lengths of around 500 m and diameters of 2.5–5.5 m would pose no special excavation problems. The longest raise in one step is in South Africa, 1,250 m in length and with diameter 2.4 m. There is also a 1,000 m raise, diameter 6 m in South Africa. Largest diameter used for a raise is 7.1 m. Boring has been successful in rocks with uniaxial compressive strength up to around 600–700 MPa.

SKB has experience from raise-drilling at Äspö Hard Rock Laboratory, where the hoist shaft (diameter 3.8 m, length 420 m) and two ventilation shafts (diameter 1.8 m, 420 m) were drilled in stages to –220 m, to –340 m and to full depth –450 m. For the hoist shaft a free cylinder of 3.0 m was needed for the hoist and it was first deemed that a 3.8 m reamer would be sufficient by drilling a 360 mm pilot hole from the surface down to level –220 m.

While the deviation was slightly larger than planned a 4.1 m reamer was necessary. For the next two 100 m stages, the 3.8 m reamer was sufficient. Pre-grouting for the first part (down to -220 m) was performed by grouting the pilot hole and at lower stages by grouting periphery holes.

Drilling straight vertical holes is licensed standard technology using special tools for surveying (gyro) and steering. Typical maximum deviation when using these tools is 0.025% of the length (0.25 m @ 1,000 m), but 0.010% has been achieved (0.10 m @ 1,000 m). Without tools, deviation is typically 1% of the length of the hole for a vertical application but 0.5% has been achieved on occasions.

4.4.2 Excavation of deposition tunnels

We have studied the possibility to use raise-drilling equipment for excavation of the 6.3 m diameter deposition tunnel and shortly describe design, operation and typical performance.

Design

A condition for horizontal pull-reaming is that there is service tunnel where the reamer can be mounted after the pilot hole has been drilled. A possible repository layout is shown in Figure 3-4.

Readily available design (Robbins 191RH) can be used for the assembly. The machine thrust is 11.6 MN and torque up to 800 kNm. The weight of the derrick assembly is 45 tonnes and the reamer head around 40 tonnes. The reamer is equipped with 36 cutters, Figure 4-21.

In horizontal pull-reaming, a minimum amount of stabilizers on the drill string is to be used since these will add to the friction coefficient between the drill string and the rock mass. In most applications one stabilizer will do. On the reamer at least two, so called wings or scrapers, is attached to facilitate removal of the muck from the cutters and to avoid muck getting stuck between the cutters.

Transportation of the derrick can be facilitated by means of a self propelled crawler, a sled or similar rubber-tired or rail-mounted equipment. In any case, effective mobilization of the basic machine will provide for a high utilization of the equipment which in turn foster for a high productivity. At this stage a self propelled crawler is recommended, which also will serve as foundation while reaming. From a working environmental and an emissions point of view, an electric powered crawler with rubber tracks is recommended. Other power choices include pneumatic and diesel propelled solutions. Transportation by means of a crawler is also a fast and working friendly solution.

The reamer has to be transported to its original location in the service tunnel, and further moved to the next drift for hook-up and collaring. Hence, the shortest transportation distance will be through the completed horizontal tunnel. A conceptual transportation alternative using a dolly (platform on a roller) is shown in Figure 4-22.

Different systems for mucking have been investigated and normal Load-Haul-Dump equipment would be preferable due to the short tunnels. If the drifts inclination is 3% or higher, water flushing only is sufficient for transportation of muck to the lower level. In this case, which require quite a flushing capacity, the reamer has to be provided with jet nozzles for dust collection and to help flush the muck.



Figure 4-21. Design sketch for horizontal reaming of the deposition tunnels



Figure 4-22. Transport of reamer out of the tunnel using a dolly.

General site set-up

A niche is excavated at the entrance of each deposition tunnel of sufficient space to contain the necessary equipment, Figure 4-23. This equipment typically includes derrick and mounting system, hydraulic system, electrical system, control console, pipe loader, drill string components, tool boxes, auxiliary (mine) transformer, bailing fluid and cuttings handling system etc. In addition to supplying adequate floor space for all necessary equipment, the site layout must allow the assemblies of the raise boring machine to be positioned to enable all electrical cables and hydraulic hoses to be interconnected. The raise boring site must have adequate overhead clearances for the setup and operation of all necessary equipment. The full height of the working site is deemed to be higher than the reamer diameter.

With the steel beam mounting system, the derrick assembly is resting on its transporter or a steel frame and secured to the rock face via a steel frame mounting system, see Figure 4-23.

Operation

Site Preparation

A concrete pad is constructed to support the machine set-up. Equally, a flat surface vertical concrete wall is erected for support of the wall mounted part of the steel beam drilling platform in the back tunnel. The steel beam drilling platform attached to the drilling face by means of rock bolts.

Reaming

The speed and thrust is gradually increased to a continuous operation mode. Pipes are removed as required while pulling the reamer towards the machine set-up. Reaming is continued until completion/breakthrough or will be stopped some meters before breakthrough leaving a piece rock acting as bulkhead.

Mucking

Assuming a penetration rate of 0.4 m/h, the rock volume to muck is around 12 m³ of rock per hour or roughly 25 m³/h, including a swell factor of 2.0. A small size electric LHD, with a scoop capacity of 3 m³ (nominally heaped) and a vehicle speed of around 5 km/h (fully loaded) and 10 km/h (empty bucket) would suffice. With a maximum distance of 600 m per round (300 m back and forth), and a total bucket fill and dump time of 60 seconds, the capacity of loading will be around 30 m³/h that is > 25 m³/h produced by the RBM.

Rock support and grouting

The cored hole and the pilot hole will provide early information of rock conditions. Some grouting work might be done from the cored/pilot hole, but it assumed that it would not be sufficient for the 6.3 m diameter tunnel.

It is here suggested that any needed grouting and rock support work is executed from the back of the reamer with entrance from the service tunnel. The deposition tunnel is wide enough to accommodate a standard drilling rig that can drill holes for the grout fan and rock bolts, and standard equipment for grouting, rock bolting and for possible meshing/shotcreting.

Reamer Transportation

Rails are installed in the tunnel invert and dollies mounted. The reamer is retracted a few meters, and one of the reamer spokes dismantled. The reamer is lowered to a fixed proper position on the dollies. The reamer is dragged backwards and prepared for docking of the reamer to the transporter. The reamer is put in proper position on the transporter, the dollies dismantled and the reamer rotated 180 degrees to facilitate assembly of the dismantled spoke. The reamer is then moved to the next work site.

Machine Demobilization

The derrick and equipment is dismantled and moved to next working site.

Typical performance

While the machine may drill vertical raises up to 1,000 m length we here assume that maximum length is 300 m for the horizontal drifts. Based on the geological conditions assumed, typical best penetration rate is around 0.4 m/h during reaming. It is here conservatively assumed that one equipment may drill two deposition tunnels per year based on machine availability of 90% and utilisation of 70%, Table 4-7.

Table 4-7. Estimated work cycle for the 265 m long deposition tunnel.

Activity	Average number of workdays
Coring and pilot hole	(Coring 15 days, Pilot holes 5 days not included, executed before site preparation machine set-up)
Site preparation	(Not included, executed before machine set-up)
Machine installation	4 days
Drill string travers	2 day
Reamer handling	(3 days, not included)
Reaming	67 days
Machine demobilisation	4 days
Reamer demobilisation and transport	(7 days, not included)
Equipment move	2 days
Maintenance	7 days
Total cycle for reaming one deposition tunnel	86 workdays

4.4.3 Future developments of raise-boring technology

Application of raise-boring technology is also described in Section 4.6 but we summarise here some trends.

Current machine developments in the industry are focusing on both smaller and larger units, for applications ranging from slot holes to hoist shafts as well as horizontal reaming of drifts and tunnels. Very large diameter machines (5 to 7 m) are an ongoing trend in South Africa but also in North America and Australia. Essential for long hole applications is a so called ‘Rotary Drilling Vertical System’ (RDVS), to drill as vertical pilot holes as possible – thus reducing the deviation that will be significant over a very long hole. Two or three proven systems are available on the market, frequently used by large contractors. Another option is to drill a straight pilot hole by means of other equipment, core drilling for instance, which also make sense in order to not keep the high capital investment large diameter machine occupied pilot drilling small diameter holes.

An advantage of the raise boring method includes the capacity of blind reaming (boxhole boring), either up, down or horizontally. For special applications whereas vertical development is only feasible from the lower level – e.g. shafts in infrastructure projects with a severe topography or mine environments with the mine entrance at the bottom level – the interest in and feasibility of boxhole boring has indeed increased.

Future applications for boxhole boring includes infrastructure projects with a severe topography, where tunnels require ventilation or pressure shafts that are more feasible to develop from the lower level and upwards. Another market is mines, which can be found in the Andes, requiring development from the lower level and upwards to the future upper mining levels. Hydropower projects with surge and pressure shafts, is another market that find boxhole boring interesting.

The requirements of automated raise boring equipment will most likely increase in the future with truly semi-automated systems to be developed, and in the longer run also fully robotized systems to follow. The suppliers have this technology available and the first step towards automated raise boring has since a few years back already been taken. The CanBus system developed by Bosch in the early 1990’s for the automobile industry has since then been used in many other applications, such as forest machinery, textile industry, traffic control and cranes. Atlas Copco has already introduced CanBus based rig control systems to their Boomer C-rigs’ and implementation for raise-drills are under way. With true

semi-automated raise boring systems since long ago on hand, the next step to go is for full automation.

Another logical future event is to dismiss the mechanical connection between the raise boring machine and the rock tools – i.e. the drill string. This step would require the rock tools to be an integrated part of the machine, which also would enable the unit to directly apply thrust and torque on the tools without having to be transferred by a drill string. Systems like this have already been tested on the market, and future demands from modern mines together with the drive with innovative manufacturers will most certainly show the route to the future.

4.5 Excavation of deposition holes

SKB and Posiva have jointly tested technology for mechanical excavation of deposition hole. The first tests were done at Olkiluoto using down-reaming to prove feasibility of drilling the deposition holes /Autio and Kirkkomäki, 1996/ where three test holes of diameter 1.5 m and depth 7.5 m were drilled. Seventeen deposition holes with diameter 1.75 m and depth 8.5 m have been excavated at Äspö Hard Rock Laboratory using the alternative shaft boring technology /Andersson and Johansson, 2002/. The techniques used both meet the geometrical tolerances for straightness, diameter variations etc. The techniques are however not yet fully efficient.

Atlas Copco has in this project further studied technology for down-reaming and suggested a conceptual design for down-reaming.

4.5.1 Down-reaming

Down-reaming is basically excavation using ordinary raise-boring equipment with a derrick, a drill string and a reamer. However removal of cutting needs special technology; here vacuum sucking through the drill pipe was used.

Design of equipment

A separate pilot hole is drilled ahead of the reamer as a guide and stabilizer, Figure 4-24. The machine used at Olkiluoto developed thrust of 500 to 630 kN depending on position of the cutter head and a torque of 74 kNm during reaming. The weight of the derrick was close to 9 tonnes and the weight of the reamer close to 4 tonnes dressed with 8 roller button cutters and 4 gauge rollers, Figure 4-25.

The excavated rock was sucked through the reamer and the drill pipes to a suction line see /Autio and Kirkkomäki, 1996/ for details. The use of vacuum flushing and transport was found to be advantageous to water flushing, as the equipment is compact and easy to move and that it is possible to observe the surface and the bottom of the deposition hole during drilling.

The tests showed that the following details should be improved to increase efficiency:

- Wear-resistant vacuum suction system, including the suction nozzles.
- Set-up and move of equipment between holes.
- Cleaning of filters and removal of cuttings.
- Filling of the pilot hole.

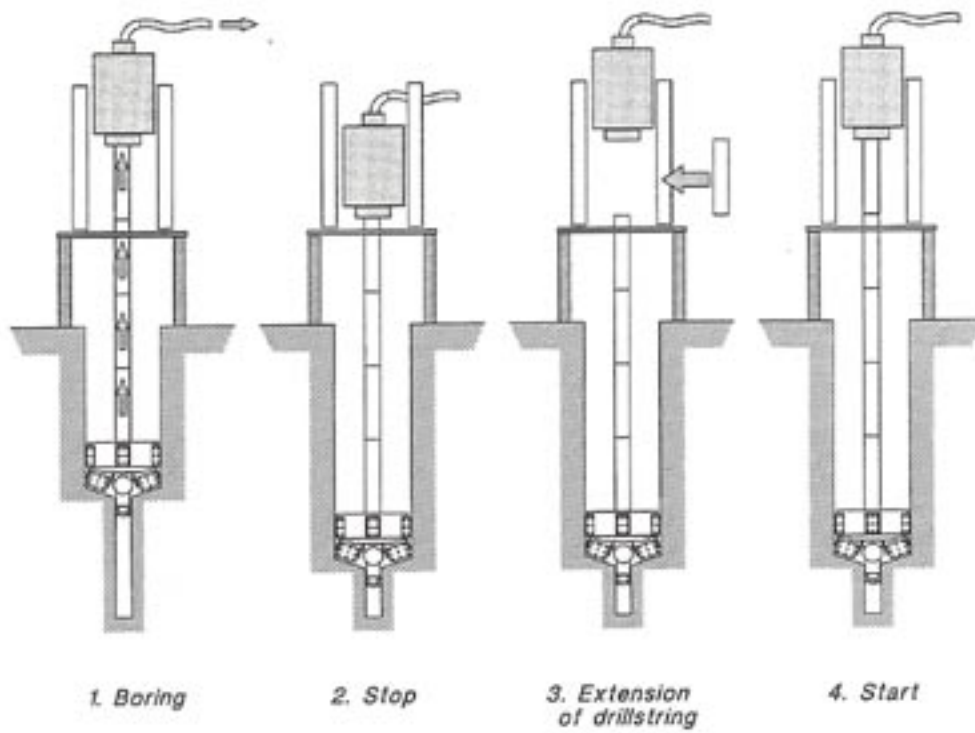


Figure 4-24. Principle for down-reaming /Autio and Kirkkomäki, 1996/.



Figure 4-25. Photo of the reamer above the excavated hole /Autio and Kirkkomäki, 1996/.

Typical performance

The test performance showed that the technology is feasible; however penetration rate was low due to maintenance and repair work. Only 13% of the working time of 64 days for the three holes was used for the pilot hole drilling and reaming, Table 4-8. The low utilisation is due to novel technology and the general test conditions where neither equipment nor organisation is optimised. It has been estimated that a single hole of 7.5 m would be drilled in 6 shifts using a crew of two persons without interruptions and maintenance of the vacuum suction line /Autio and Kirkkomäki, 1996/.

Maximum penetration achieved was 1.2 m/h with an average of 0.9 m/h while the machine actually performed. The penetration is limited by the efficiency of vacuum cleaning in keeping the bottom of the hole free from excavated rock.

The geometrical measurements of the excavated holes showed that variations in the average radii are less than 3 mm. The theoretical and true centreline at the bottom of the three tests holes were offset 32 mm, 27 mm and 17 mm respectively corresponding to deviation of 0.2–0.4% of the hole length.

Recent developments

SKB is presently considering a slight design change of the deposition hole, here assuming a circular shape of the tunnel, Figure 4-26.

A conceptual design of the machine has been prepared. The total weight of reamer and derrick assembly is around 25 tonnes. Air (minimum 10 m³/min at 4–6 bar) or water (minimum 550 L/min at 4–7 bar) is used for muck removal through an inner pipe of the drill string. It is assumed that the pilot hole is around 311 mm and the diameter of the drill pipe is 508 mm. A sketch of the operational procedures is outlined in Figure 4-27.

After site preparations, the complete machine with reamer is positioned and docked to base plates. The machine is run at a low speed during initial collaring until the tricone bit and bit reamer stabilizer is partly buried. When all cutters are in contact with the drilling face, speed and thrust is gradually increased to recommended values and the machine run in continuous operation mode to the end of the stroke when additional drill pipes are added. Drilling is continued to full depth of the deposition hole, when the drill string is retracted and operation reversed.

Table 4-8. Activity division for the three test holes at Olkiluoto.

Activity	Percentage of time
Set-up	16%
Preparation for boring	23%
Boring of pilot hole	4%
Reaming	9%
Repair and maintenance	24%
Emptying of tank for crushed rock	3%
Outside service	9%
Move of equipment	12%

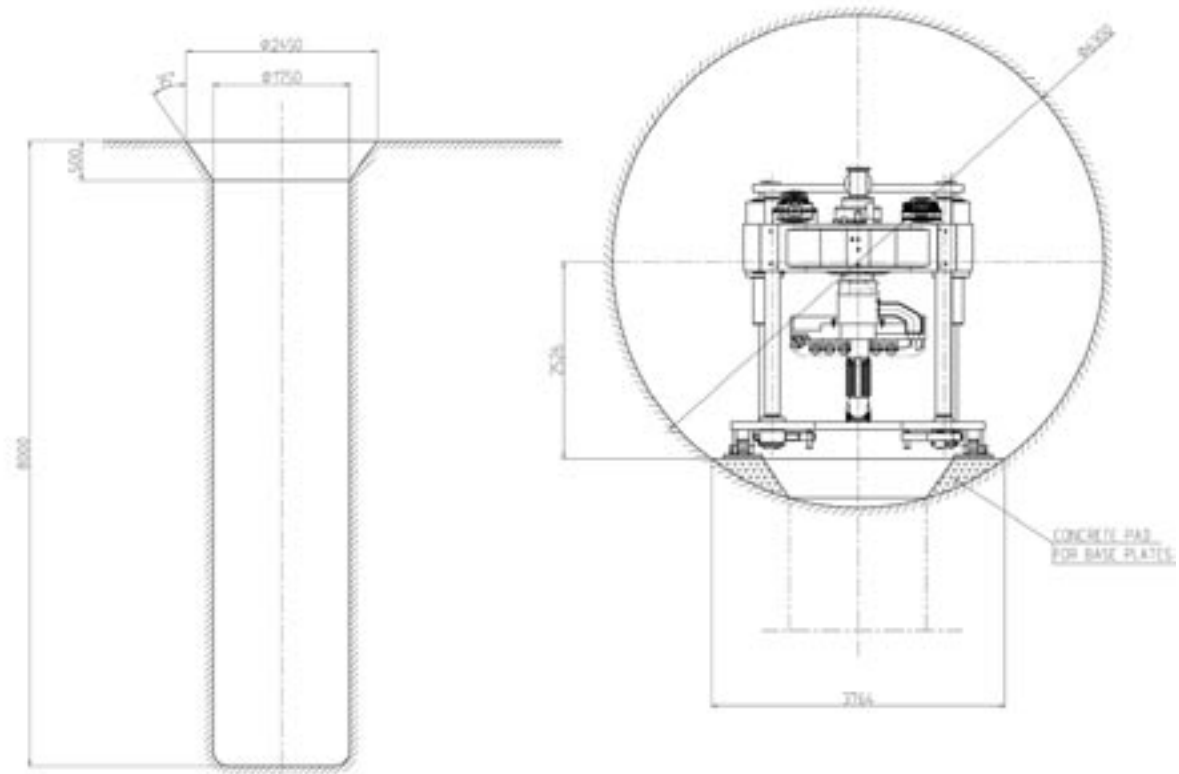


Figure 4-26. Possible deposition hole design and a conceptual design of a down-reaming machine.

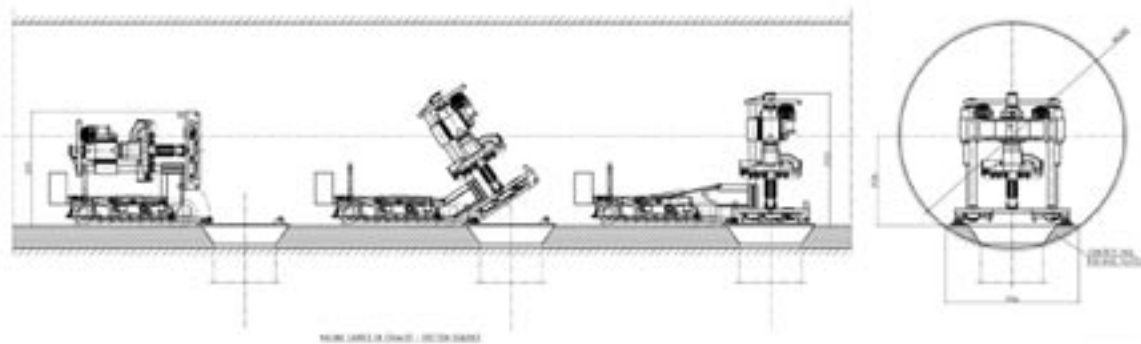


Figure 4-27. Operational procedure for drilling of the deposition hole.

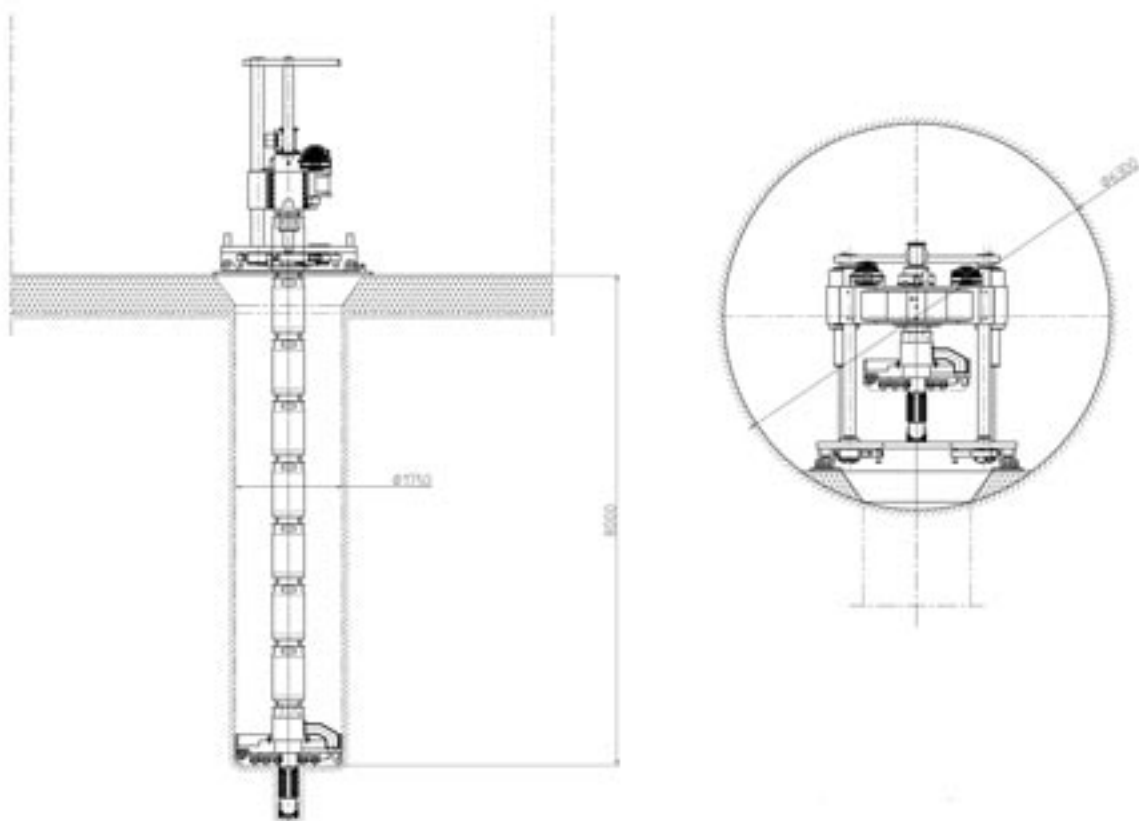


Figure 4-28. Drilling of deposition hole

4.5.2 Shaft Boring Machine

The experience from drilling 13 experimental deposition holes at Äspö Hard Rock Laboratory with diameter 1.75 m and depth 8.5 m is reviewed in /Andersson and Johansson, 2002/.

Design of equipment

The design of the machine is similar to a TBM, Figure 4-29 and Figure 4-30 but here denoted as a Shaft Boring Machine. The machine is designed for thrust up to 3,500 kN, but only 2,000 kN of the thrust was employed in the tests. The cutter head is dressed with 19 cutters of two types, two row carbide button cutters and disc cutters. Cuttings are removed by a vacuum suction system through pipes in the cutter head. 900 mm casings are used to transfer load from the thrust cylinders and these casings are mounted along with the boring.

Operation

Transport and set-up

The machine is transported with a trailer and moved on place using skids. A laser system is used in the tunnel to pin-point the location of the deposition hole and the machine and also to align the machine vertically. The machine is bolted to the floor or braced to the roof using crown reaction pads.

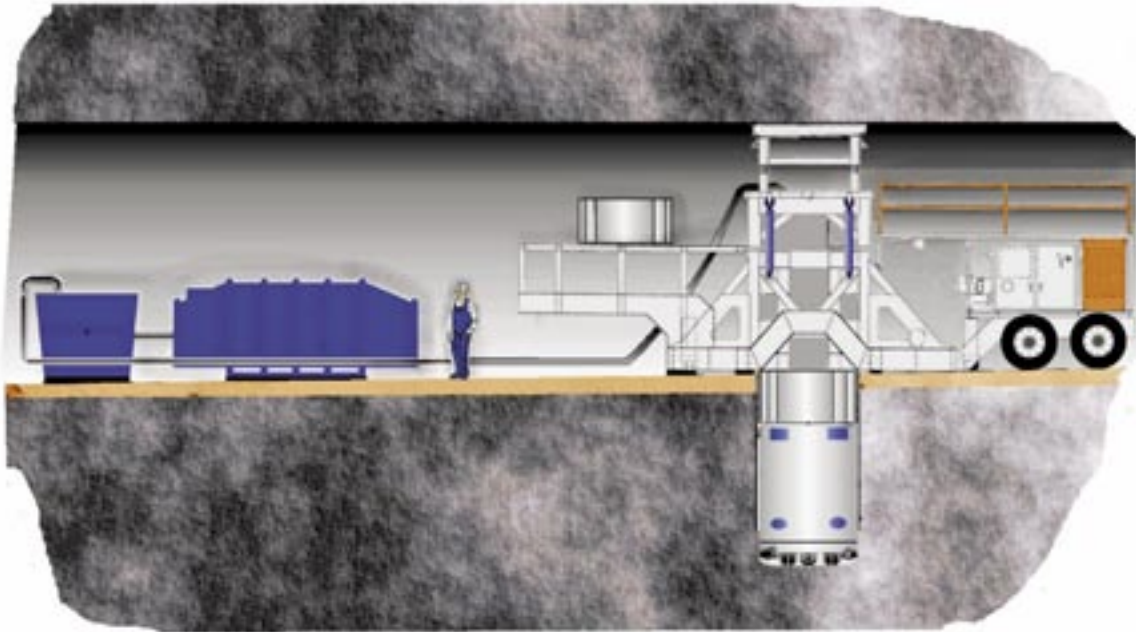


Figure 4-29. Vacuum system (left) and Shaft Boring Machine (right)



Figure 4-30. Photo on the Shaft Boring Machine used to drill test deposition holes at Äspö HRL.

Boring

Thrust cylinders are fully retracted and the start casings emplaced. The casing is bolted to the SBM and the head frame. The front stabilizers are released and the machine retracted to the steering position where steering of the machine is controlled by the front stabilizers. The boring begins until the thrust cylinders are fully extended when the front stabilizers are regripped to the wall of the hole. The previous casings are unbolted from the head frame, the thrust cylinders retracted and additional casings mounted.

Change of container for the cuttings

The cuttings are sucked from the hole to a collecting unit with a main container for the coarse material and a vacuum pump for the fine material. The filters in the vacuum pump are automatically cleaned approximately every 10th minute.

After boring a length of around two casings (1.8 m) the container with cuttings are changed which needs disconnection of the pipes.

Typical performance

The effective average rate of penetration is 0.45 m/machine hour with a maximum penetration of 1.1 m/h. The overall efficiency is however low as the average duration to complete full deposition hole is 105 hours, excluding time for measurements. The time-split is shown in Table 4-9.

Measurements of deposition hole geometries shows that the SBM produces holes that fulfils the requirements. The average deviation from start of hole to end of hole is 5 mm with 13 mm as highest value measured. The average diameters varied in between 1,757 mm to 1,762 mm dependent on gauge cutter configuration and also straightness of holes was within specifications.

4.6 Excavation of KBS-3H deposition drifts

SKB is investigating an alternative design where the canisters are deposited in horizontal drifts rather than in vertical deposition holes. /Lindgren et al. 2003/. Each deposition drift is supposed to be 1.85 m in diameter and around 300 m in length. The geometrical tolerances needed, see Figure 3-7, put stiff requirements in straightness of drift, small diameter changes where diameter should be in the interval 1,840 to 1,850 mm (± 5 mm) It is therefore supposed that a cored hole would be used to guide the later pilot holes needed. Three options for excavation of horizontal drifts are then shortly described, drifting by horizontal pull-reaming and horizontal push-reaming and drifting by water-percussion hammers.

Table 4-9. Time-split for the 13 deposition holes at Äspö Hard Rock Laboratory.

Activity	Percentage of time
Transport	11%
Set-up	19%
Boring	18%
Repair and maintenance of boring machine	18%
Repair and maintenance of vacuum system	10%
Handling of casing	11%

4.6.1 Drilling of the cored hole and the pilot hole

For each deposition drift (and tunnel) it is planned that coring and investigations in the borehole precede the drift excavation. We assume that a diameter 76 mm hole is drilled. To fulfil the stringent requirements we need measurements in the hole to track any deviations and directional devices to guide the core barrels. One example of system for measuring deviations is the Reflex Maxibor, an optical device that with high precision measures the bending of its own rods when inserted in the drill hole. Based on the measurements, the further coring is guided by a directional system. Such a system is the Liw-In-Stone barrel that is designed to guide the coring.

The penetration rate for a 76 mm diameter hole is assumed to 1.5 m/h at an accuracy of 0.2% with deviation measurements every 9th m and using a so called wire-line equipment with triple barrels for the coring.

After the cored hole is finished a pilot hole is drilled in steps. Depending on drill string diameter, the pilot hole will be developed in two or three steps. The first step is from 76 mm to around 140–165 mm the second to around 215 to around 280 mm and the third up to around 445 mm or as needed. The pilot holes can be drilled with the RBM machine or the cluster drilling machine (see below) but we here assume it would be more efficient to use a dedicated Down-The-Hole machine (percussion drilling with the engine at the bottom of the drill hole). Drill time for a 285 m hole would be in the range of 40 to 60 hours for a 280 mm and a 315 mm hole respectively.

There also exist commercial systems to guide pilot holes drilled by rotary crushing drilling. The principle of control is based on the interaction of interior and external drill pipe within the area directly behind the bit, Figure 4-31. Steering along the predetermined borehole course as mentioned above requires measurements and transmission of the relevant data to the drilling rig control stand. The measuring data are transmitted wireless to a receipt unit at the drilling rig, where data are displayed and visualized. By comparing the planned and the actual deviations are detected, so that afterwards a control can be initiated by reorientation of the external drill pipe.

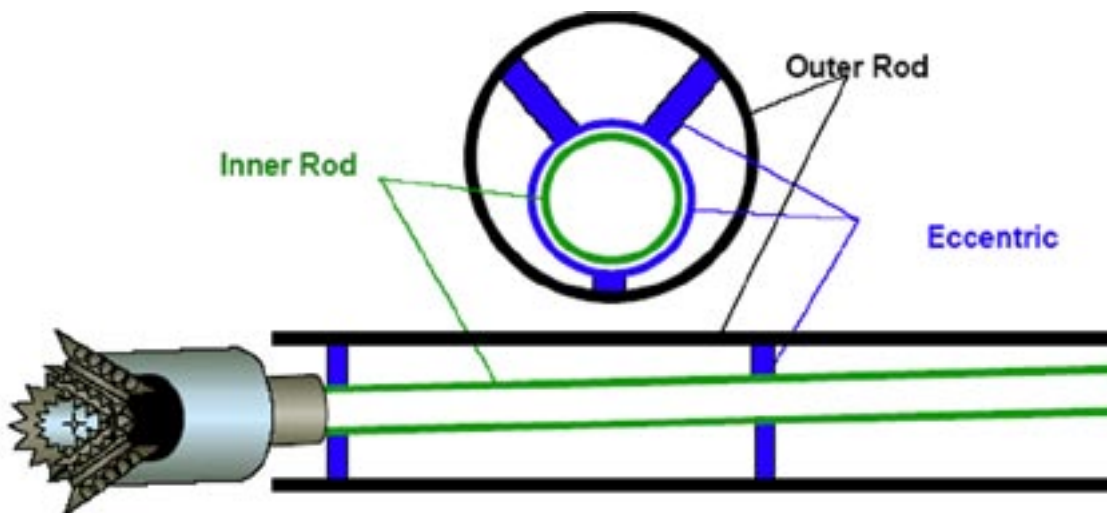


Figure 4-31. Principle for guiding the pilot hole with rotary crushing drilling. Courtesy DMT GmbH.

4.6.2 Horizontal pull-reaming

The technology is analogous to the description for drilling the 6.3 m diameter deposition tunnel (Section 4.4.2). As also here assume that there is a service tunnel available where the where the reamer can be mounted after the pilot hole has been drilled.

Readily available design (Robbins 73-RM-H) can be used for the assembly, Figure 4-32. The machine thrust is 4.2 MN and torque up to 225 kNm. The weight of the derrick assembly is 10 tonnes and the reamer head around 5 tonnes. The reamer is equipped with 10 cutters, Figure 4-21. The straightness of the excavated drifts is assured by the straight cored hole before the pilot hole of 311 mm is drilled and the drift reamed to full dimensions. The diameter changes due to cutter wear is down to a few millimetres and the true change is a matter of cutter button design and change of cutters. It is deemed feasible to meet the stringent requirements on geometry.

While the machine may drill vertical raises up to 700 m in length we her assume that maximum length is 450 m for reaming the horizontal drifts. Based on the geological conditions assumed, typical best penetration rate is around 1.3 m/h during reaming.

It is here suggested mucking is by a slurry pump rather than using LHD equipment.

The proposed slurry pump system consists of a flushing water inlet pump (type Grindex Master) with a maximum pump capacity of around 80 L/s. The inlet pump is feeding the reamer flushing water through a standard pipe. The reamer scrapers lift the cuttings into the non-rotating muck collector, trailing behind the reamer. The outlet slurry pump is (type Weir Warman 4/3 C-AH) lined with hard metal alloy and mounted to the non-rotating muck collector. Slurry, rock cuttings and water are transported to the drift entrance through a hard metal alloy lined steel pipe with diameter 100 mm. The inlet and outlet pipe is extended when every third to fifth drill rod is removed depending on the pipe length selected for the in and outlet system. Power to the outlet pump is most conveniently supplied through a cable reel. Cuttings are settled outside the tunnel entrance and the water is pumped back to the reamer. A 16.3 L/s slurry pump flow will give the cuttings a transport speed of 2.1 m/s in a 100 mm pipe that should be sufficient to transport the cuttings without the risk of sedimentation in the outlet pipe. A higher transport speed for the cuttings will increase the wear rate of the outlet pipe. The selected pump is able to pump cuttings in sizes of up to 36 mm in diameter. The estimated energy to pump the cuttings is 0.75 kWh/ton solids. (The slurry consists of 15% solids by weight). Based on a maximum theoretical rate of penetration at 1.35 m/h over a 2.69 m² section, the weight of the muck volume produced during one hour with full system utilization of 63% and a rock density of 2.74 ton/m³ is approximately 10 tonnes/h. The pump capacity is (9.7 tonnes/h) is in parity with the production, but more thorough studies on mucking would be needed.

It is here assumed that one equipment may drill around 5 deposition tunnels per year based on machine availability of 90% and utilisation of 70%, Table 4-10

Horizontal pull-reaming is standard technology. Incidentally a tunnel with almost identical measures as the KBS-3H drift – 1.8 m in diameter and 285 m long and horizontal – was successfully reamed in Norway in October 2003, Figure 4-33 and Figure 4-34.

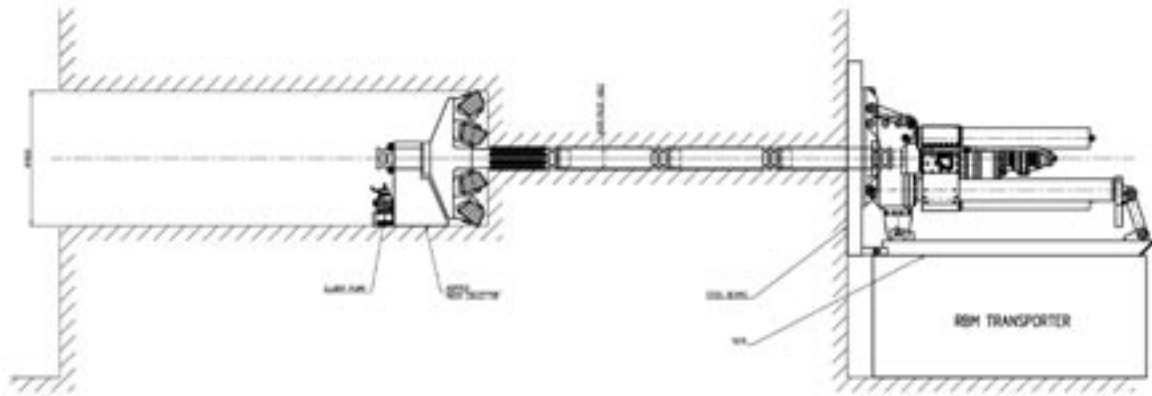


Figure 4-32. Design sketch for horizontal reaming of the 1.85 m deposition drift.

Table 4-10. Estimated work cycle for the 265 m long deposition tunnel.

Activity	Average number of workdays
Coring and pilot hole	(Coring 15 days, Pilot holes 5 days not included, executed before site preparation machine set-up)
Site preparation	(Not included, executed before machine set-up)
Machine installation	3 days
rill string travers	1 day
Reamer handling	(1 day, not included)
Reaming	20 days
Machine demobilisation	3 days
Reamer demobilisation and transport	(7 days, not included)
Equipment move	2 days
Maintenance	7 days
Total cycle for reaming one deposition tunnel	36 workdays



Figure 4-33. Picture from the Norwegian Sandvika project in October 2003. The photo shows the reamer from behind and the slurry system. Courtesy Atlas Copco Construction & Mining.



Figure 4-34. The reamer head used at the Sandvika project. The tunnel is 1.8 m in diameter and with length 285 m. Courtesy: Atlas Copco Construction & Mining.

4.6.3 Horizontal push-reaming

The drawback with the pull-reaming is the need to construct and backfill a 30 m² service tunnel at high costs. This disadvantage is eliminated by using push-reaming where the reamer is pushed rather than pulled, Figure 4-35.

The technology is many ways analogous to the description for pull-reaming the 1.85 m horizontal drift.

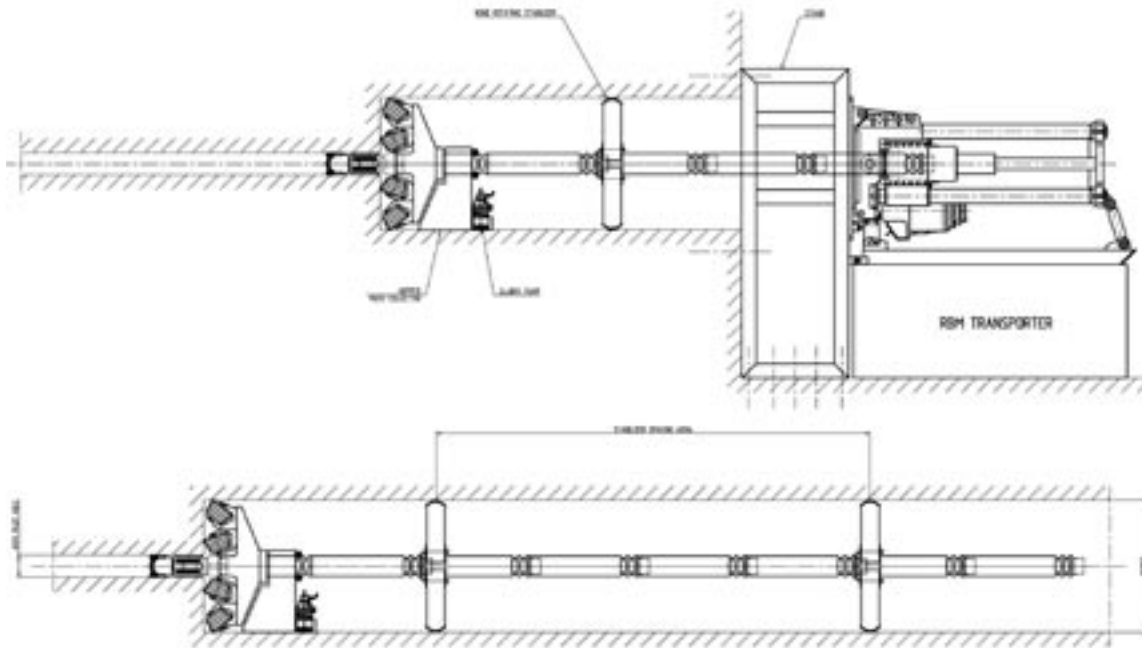


Figure 4-35. General design of set up for horizontal push-reaming.

Readily available design (Robbins 53-RH) can be used for the assembly. The machine thrust is 2.5 MN and torque around 150 kNm. The weight of the derrick assembly is close to 17 tonnes and the reamer head around 5 tonnes. The reamer is equipped with 10 cutters. The straightness of the excavated drifts is assured by the straight cored hole before the pilot hole of 311 mm is drilled and the drift reamed to full dimensions. The diameter changes due to cutter wear is down to a few millimetres and the true change is a matter of cutter button design and change of cutters. It is deemed feasible to meet the stringent requirements on geometry.

The reamer design is obviously different from the pull-reaming, but also the drill string is different as stabilizers are used to avoid bending and buckling of the drill string. Every 5th drill pipe or so is a specially designed pipe with an outer bearing where the stabilizer spokes are attached. The stabilizers are not gripping the rock, so they may rotate occasionally. The design should then also account for that the slurry pipes should pass the stabilizers. To stretch the efficiency of the system it is likely that the slurry system is doubled to increase availability and utilisation of the overall excavation system.

The operation is similar to other reaming operations but with a non-rotating stabilizer mounted around every 5th pipe and that the slurry system is removed before the reamer is retracted.

The typical performance is in general similar to the description in Table 4-10, but reaming is somewhat slower. Reaming and retracting of the drill string is estimated to 33 days. Maximum penetration rate is estimated at 1.15 m/h and it is expected that around 1,000 m of deposition drifts are drilled per year.

4.6.4 Cluster drilling

The principle for water-percussion drilling is discussed in Section 2.3.3. SKB has developed and initially tested a machine where first a pilot hole of 254 mm is drilled. Two sets of cluster with percussion drilling machines are then used to ream the hole to 1,440 mm then to full diameter 1,850 mm, Figure 4-36. The clusters are rotating while the drill machines drill constantly in new formation. Stabilisers are used to prevent the drill string to bend down to the bottom of the hole and introduce bending forces on the cluster.

The weight of a cluster including hammers is around 6–7 tonnes. The equipment is designed for thrust of 400 kN and torque of 80 kNm. Field tests show that thrust during reaming is in the order 80 kN and the torque less than 1.5 kNm for short drifts. It is anticipated that thrust is less than 200 kN and torque less than 20 kNm for the 300 m long drifts. Water (around 5,000 L/min) is used for the percussion hammers as well as for the mucking by slurry pumps. The clusters house 12 percussion drill machines designed and manufactured by Wassara AB in Sweden.

A typical set up of all equipment is showed in Figure 4-37. The start chamber is around 7 m · 7 m in width and height and 11 meters in length.

With respect to typical performance it is expected a 265 m pilot hole is drilled in 7 days and that reaming to 1,440 and 1,850 mm takes 10 days respectively which makes in total around 27 days for excavating the drift assuming an overall system availability of 70%. With a straight pilot hole previous field tests underpin that the stiff requirements on geometrical tolerances are met.

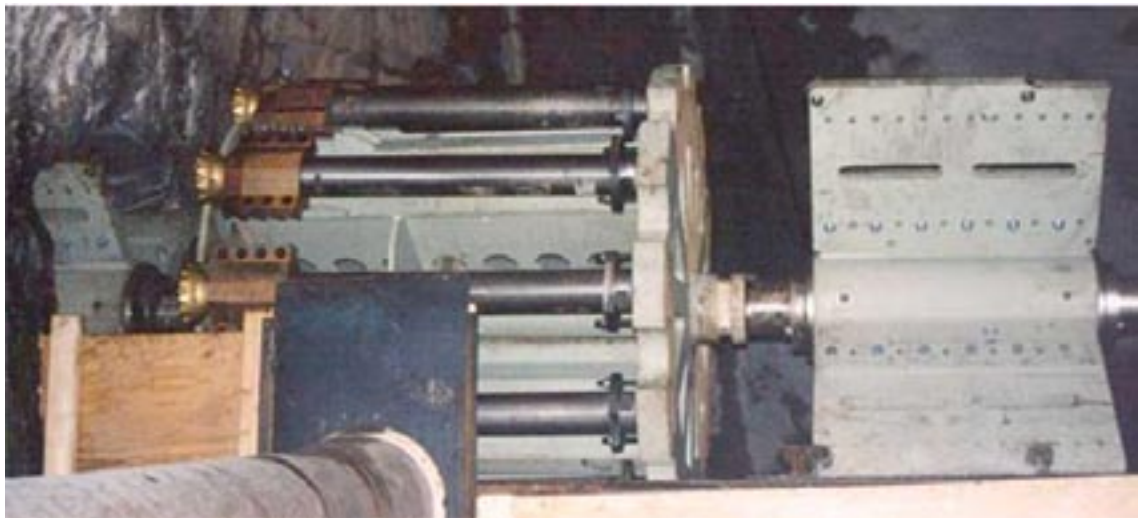


Figure 4-36. Picture from pilot tests of the equipment. (Left part). The cluster for reaming to 1,850 mm. (Right part) Stabilizer. Courtesy Wassara AB.

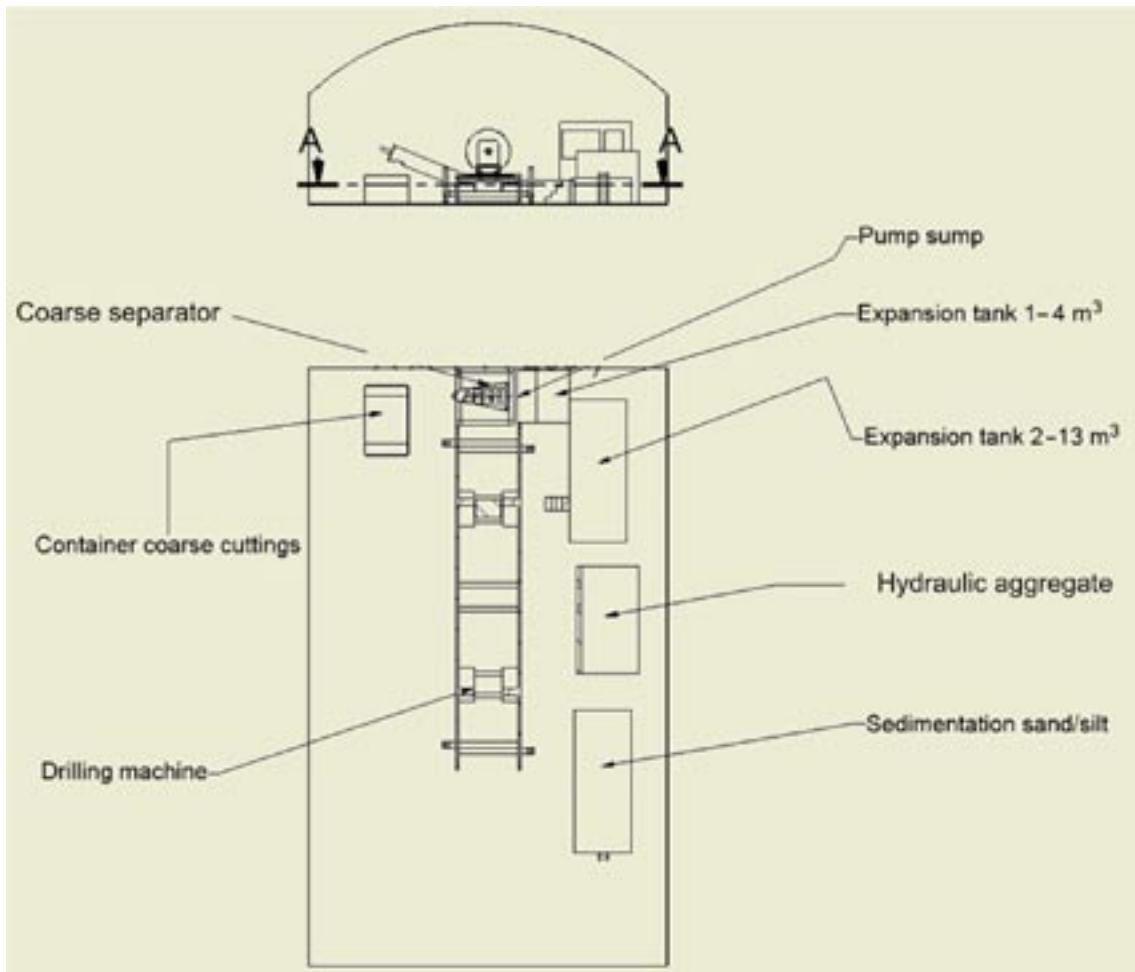


Figure 4-37. Typical set-up for the cluster drilling equipment.

4.7 Miscellaneous excavation methods

A few excavation methods still in an R&D phase have been studied, like plasma technology, hydraulic fracturing and oscillating disc cutter technology. None of the presented exotic fragmentation methods are fully developed up to date, and are not expected to reach the required maturity within a 10-year perspective.

The plasma technology is studied by CANMET in Canada. A plasma torch generates a very high temperature, enough to break the surface rock in a spalling effect. With sweeping movements of the torch, the flame creates a channel in the rock. The tests on a big granite block at the Hydro-Quebec laboratory during the summer of 2002 showed the best achieved excavation performance to be around 0.4 ton/hour.

Hydraulic fracturing is a half century-old technology used in oil and natural gas production. Water is injected at high pressures to effectively open up and extend previous fractures or to create new fractures in the rock. The technology is used for stress measurement but has also been studied in Australian mining by CSIRO with the idea to enlarge existing mining openings by induced caving.

The oscillating disc cutter (ODC) technology is an extension of existing cutter technology, but the cutters are oscillated at a controlled frequency further weakening the rock by causing it to fatigue. During the excavation a high pressure water jets directed at the rock

and tool interface help to propagate the crack initiated by the tool that forms the rock chip. Tests by CMTE in Australia showed that the cutter thrust could be decreased from 550 kN to 12 kN with same penetration using ODC that in contrast to conventional disc cutters, which require high compressive loads to fracture rock, exploits the tensile weakness of rock through “undercutting”. Current TBM and raise boring technology utilizes a pattern of circular cutting discs that rotate and fracture the rock as the cutterhead pushes forward. The ODC cutter oscillates at 40–80 Hz and uses a single tool, which rotates in a slightly eccentric circle at high speed, and is forced sideways into the rock face. This concept uses less energy, as it fractures rock in tension rather than crushing.

4.8 Implications of technology on the repository layout and operation

As evident from this chapter several technologies exist and it is not immediately obvious what Best Available Technology would be for excavation. Some discerning factors are here discussed as an introduction to the next Chapter 5 and to the comparison Chapter 6.

Shape

Different technologies will of course produce alternate tunnel shapes, Figure 4-38. TBM, reaming and cluster technologies will produce circular tunnels, while Drill & Blast, road-headers and mobile miners in principle can be used for any shape including the circular.

The shape of the tunnel may have influence on the long term stability of the opening and the rock-barrier interface. This study will only discuss this factor from a generic perspective as we need site-specific data for final selection of tunnel shape. From a practical point, the shape may influence design of equipment that will move in the deposition tunnel, where a flat floor is preferred in most instances when using rubber-tyres vehicles. Finally backfilling operation and backfilling quality in itself may be different due to the tunnel shape and the different perimeter roughness created by mechanical excavation and Drill & Blast operation.

Layout

The main layout with access ramp, shafts and central area may be more or less similar for different excavation methods, but the layout of deposition areas can be different, Figure 4-39. All methods would need a main tunnel to enter the deposition tunnel, but there is only need for an extra service tunnel when using horizontal pull-reaming.



Figure 4-38. Tunnel shape will depend on choice of excavation method. Drill & Blast, roadheader and mobile miner may in principle excavate any shape.

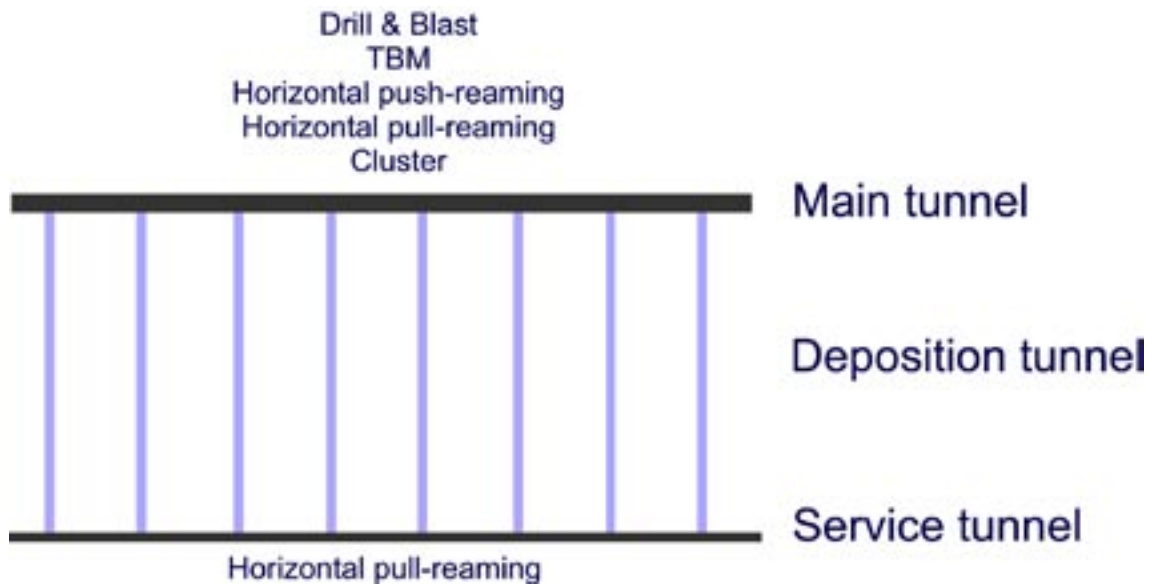


Figure 4-39. Principal sketch of layout.

The extra service tunnel may have advantages as an emergency exit and adding to flexibility in repository operation, but also disadvantages as the tunnel utilises rock volumes that might have been useful for deposition of canisters. The service tunnel would also need additional excavation and backfilling work.

The layout of the repository would also be influenced by the feasibility to construct curving tunnels. Drill & Blast can construct any curve. A TBM can be used in the range 50 m to 200 m radii depending on machine and diameter, while all other methods in Figure 4-39 only produce straight tunnels. This is no disadvantage for the deposition tunnels, but flexibility in excavation of curves is vital for the access ramp and for the transport and main tunnels.

Disturbances due to excavation

The general layout of the repository is based on the notion that construction and deposition work is separated. The reference design assumes a minimum distance of 80 m between construction activities and a backfilled and closes deposition tunnel, but this figure is based on judgement rather than studies on factual data. The concerns to address are that the excavation should not damage the long term function of the deposited canisters, buffer, backfill or plugs. Neither should installations nor instruments for monitoring be damaged.

Excavation by Drill & Blast obviously creates disturbances mainly due to the detonation. The detonation process is very fast with detonation velocities between 2,000–9,000 metre per second and the gas pressure from the explosion reach 1,000–40,000 MPa in a few milliseconds and reach temperatures of some 2,000–5,000 degrees C. The explosion energy is mainly used for fragmenting the rock but energy is also transmitted as waves in the rock. A typical ground vibration spectrum from blasting is 50–300 Hz, and Peak Particle Velocities (PPV) amounts to 5–100 mm/s at distances between 10–100 m, Table 4-11. The velocities are based on an established simple scaling relation $PPV = k \cdot Q^{1/2}/R$, where PPV is in mm/s and Q is the charge weight in kg and R is the distance between the blast and the monitoring location in m. The parameter k (not being a constant) is an entity that characterizes the attenuation from the charge to the monitoring station.

Table 4-11. Estimated Peak Particle Velocity values at different distances. The charge weight is 5 kg and the k-value varies from 600 to 250. Based on Koch's scaling law /Koch, 1958/.

Distance (m)	Peak particle velocity PPV (mm/s)	k-value
10	134	600
20	50	450
30	26	350
50	13	300
80	8	275
100	6	250

Monitoring of blast rounds often show a wide distribution of the result. Monitoring at same distances and same charge weight can give a difference in peak particle velocity of 2–4 depending on unknown factors along the travel path of the p-wave. The low PPV levels at 80 m (around 8 mm/s) are not supposed to impair the engineered barrier system and from vibration point of view it is likely that the distance between construction and deposition (80 m) may be lowered if needed.

Installations and design of instruments for monitoring should be designed for the PPV to occur. Hydromonitoring during blasting at Äspö HRL shows that the blasting would cause sudden and so far unexplained steps in the water pressure levels measured. This issue has to be dealt with in the interpretation and evaluation of monitoring data also for a repository.

The blasting creates a substantial volume of gasses, around 1 m³ per kg of explosives (around 250 m³ per round in the deposition tunnel) and these needs to be ventilated.

Compared to mechanical excavation being “continuous” operation, Drill & Blast is more of a “batch” operation. Due to the nature of blasting and the comparatively low advance rate needed it is assumed that blasting during repository operation would be made when there are no personnel underground.

Vibration due to mechanical excavation is much less than for a Drill & Blast operation. A TBM would, for example produce waves in the range of 15 to 65 Hz. Practical tests during construction of the Chattahoochee Tunnel in mica schist and quartzite gneiss (www.chattahoocheetunnel.com) show that the peak particle velocity (less than 1 mm/s) occur around 7–25 m directly ahead of the TBM. It is here assumed that similar low levels would be measured also for reaming operations.

4.9 Excavation methods and the human factor

The previous sections in this chapter have primarily dealt with technology. However, excavation is the combined effort of the technology, the human being and the organization operating the technology. Even if some excavation methods basically are automated the human skill will be very important for the overall quality and efficiency. The concept of “Best available technology” should include the human and organizational skill necessary.

It is obvious that the skill of the single machine operator has much more importance for the quality in a Drill & Blast operation than e.g. for horizontal reaming, where the final quality more or less is dependent on the machine. The quality of the excavation in the deposition tunnel would for the latter method depend on the straightness of a single pilot hole and the

cutter quality during reaming. For the Drill & Blast, the deposition tunnel quality depends on the quality of around 7,000 drill holes for about 60+ separate round where all holes should be drilled and charged and blasted in accordance with specifications for a range of rock conditions. Motivation and skill of the work force is then much more vital for reaching the required quality and efficiency.

For any excavation method selected it is vital that the work force is regularly trained, motivated and informed on how their quality may influence the occupational safety during construction, operational safety of the facility and the long term safety of the repository. For the Drill & Blast operation, the construction of the 5,000 m long spiral ramp would provide an excellent opportunity in this respect.

A common denominator for all methods is that the start-up period is challenging. The start-up is not only taking place at the tunnel face but also at work shops, at offices and along all logistic activities involved in a larger excavation project. In constructing the deep repository, the prerequisites for building up and improving the operation successively would be very good indeed as the work can be conducted over several decades.

5 Topical discussion on the excavation damaged zone

5.1 Background

The excavation of any underground opening causes some kind of disturbance on the surrounding rock. The first obvious disturbance is on the local stress conditions; any opening in a geological media causes stress perturbations. The impact of the stress change is dependent of the stress magnitudes, the shape of the opening and the mechanical properties of the host rock.

The excavation method in itself can also cause disturbance; the most significant effect may be caused by the Drill & Blast method, where fragmentation by explosives is the actual process for tunnelling.

The effect of tunnelling and the possible change of hydraulic properties of the host rock in the vicinity of the underground opening have been studied under the acronym EDZ. The EDZ is believed to be of specific interest for long term performance and safety assessment studies; development of a fractured zone in the vicinity to the tunnel perimeter may cause a permeable pathway parallel to a deposition tunnel, which could result in the more rapid transport of radionuclides in the rock mass immediately adjacent to the repository in the case the engineered barriers are impaired.

An extensive review of the EDZ for all rock types considered for geological disposal, granites, salt and clay is contained in the proceedings of the CLUSTER conference 2003 /CEC, 2003/. This report limits the definition of the EDZ to crystalline rock and the reader is referred to the reference for literature that covers other geological environments.

5.2 Terminology and understanding of the EDZ

From the résumé of previous industry workshops on EDZ /CEC, 2003/ it is apparent that there is no universally agreed definition of the EDZ, nor is there agreement as to what precisely the acronym means, but the acronym is related to Excavation Disturbed Zone or Excavation Damaged Zone.

The rock type may affect the view of the EDZ. At the Underground Research Laboratory (URL) in Canada, studies of the EDZ were carried out in a granitic basement of the Canadian Shield with extremely low fracture density and with unusually high in situ stresses. Studies were also carried out at the Stripa mine and at the Äspö Hard Rock Laboratory in Sweden. These two sites have a considerable lower stress magnitude compared to the URL in Canada.

5.2.1 Definition

The following definitions of EDZ are used in this report:

1. Excavation Damaged Zone: the part of the rock mass closest to the underground opening that has suffered irreversible deformation where shearing of existing fractures as well as propagation and/or development of new fractures has occurred.

2. Excavation Disturbed Zone: the zone in which only reversible elastic deformation has occurred.

5.2.2 Influence on the EDZ of processes during the lifetime of a repository

The definitions above shall consider all processes in the phases of the repository lifetime:

- Excavation phase; the phase that most likely causes the most significant change of the rock mass in the vicinity of the repository.
- Operational phase; drainage of openings and ventilation when local changes to the rock, primarily the fractures (precipitation, drying out) may occur. Maintenance of rock support, scaling etc. may create further irreversible changes in the rock. Backfilling of the deposition tunnels.
- Early post-closure phase; resaturation and heating are processes that can be both positive and negative for the further development of irreversible changes in the rock mass.
- Late post-closure phase – cooling and degradation of rock support need to be considered as well.

Besides the excavation phase, the heating process in the early post-closure phase is most likely the most significant process that may cause irreversible changes in the rock mass.

Processes such as future glaciations or earthquakes are not discussed in this context.

5.3 Current knowledge – state of the art review

5.3.1 Influence of state of stress

The excavation for the Mine-by test at the URL in Canada was carried out by careful Drill & Blast. In practice the full tunnel perimeter was drilled by contour holes with no explosives directly affecting the tunnel perimeter. However the high stress magnitudes caused continuous spalling with “dog ear”-shaped overbreaks, Figure 5-1.

We cite from /Chandler et al. 1996/: “In the Mine-by test, the volume of rock having stress induced damage in which hydraulic conductivity was greatly enhanced is only about 1% of the volume of excavated material. However this zone is very highly fractured, having a hydraulic conductivity that is 6–7 orders of magnitude greater than that of the intact rock. Although small in area, this zone is continuous and traverses the length of the Mine-by tunnel” Notice however that the original hydraulic conductivity in the intact granitic rock at the URL is very low.

Whereas the Mine-by test was implemented in a high stress environment, the ZEDEX-experiment at Äspö Hard Rock Laboratory /Olsson et al. 1998/ was carried out in a relatively low stress regime. The possibility to exceed the stresses to a sufficient high magnitude to develop spalling has later been studied at the APSE experiment in progress at the Äspö HRL /Andersson, 2003; Fredriksson et al. 2003/. Many measures were taken in the design of the APSE experiment to exceed the stresses high enough to cause spalling.

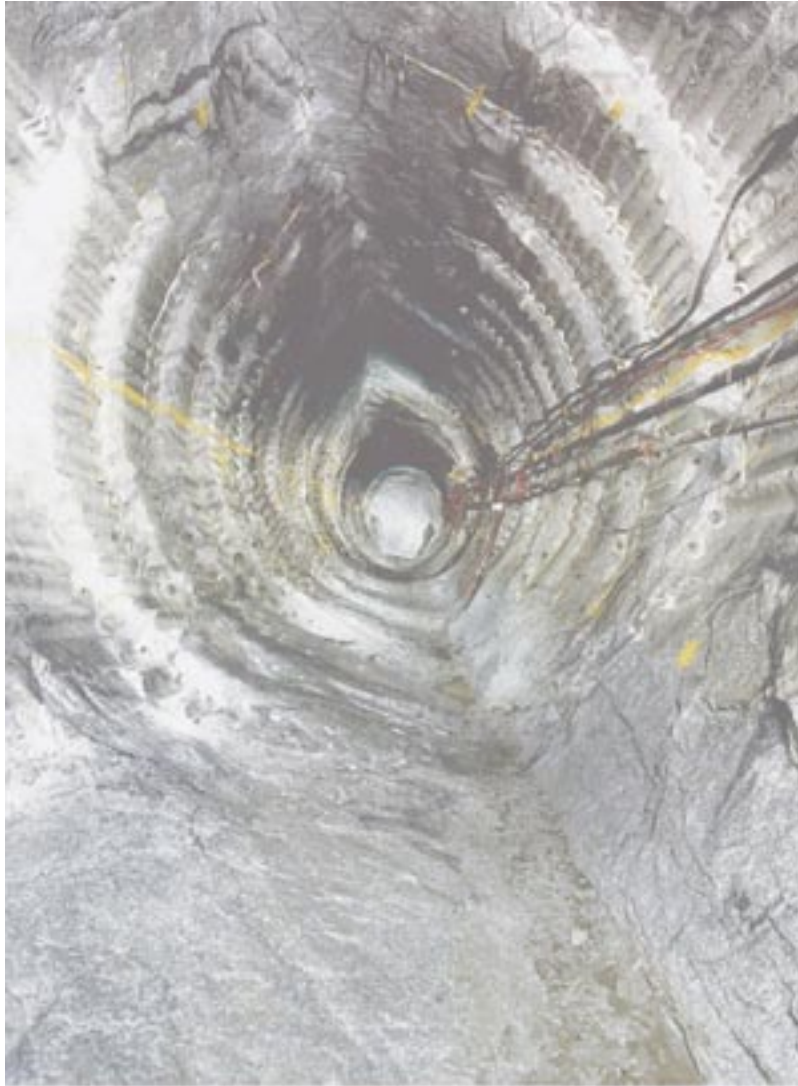


Figure 5-1. AECL's Mine-by originally circular test tunnel at the URL changed shape due to stress-induced brittle failure (spalling) /Martin et al. 2001/.

The rock mechanical effects caused by stress concentrations around an underground opening would also cause changes of the confining stress over fractures, besides possible spalling effects. Decrease of normal stress in combination with gravity may cause fall out of wedges, if fracture orientations are unfavourable from a stability point of view. It is also common that displacements caused by tensile stress or shearing cause changes to the natural fractures in the vicinity of the tunnel. Such processes are limited in extent from the opening due to the rapid drop in stress concentration with distance from a tunnel. The shape of the tunnel has influence on the stress concentration around the opening. Sharp corners causes locally higher stress concentrations than circular contours.

The possibilities to consider the rock mechanical conditions in the design are discussed by /Martin et al. 2001/. They concluded the practical experience indicating that stress-induced failure (spalling) occurs on the boundary of an underground opening in hard rock when the maximum tangential stresses on the boundary of the opening exceed approximately 0.3 to 0.4 of the laboratory uniaxial compressive strength of the rock. The design of a repository can include analyses to optimize the shape of the tunnels, the orientation of the tunnels relative to the far-field stress state, and deposition tunnel/deposition hole spacing.

5.3.2 Influence of excavation methods

The most extensive study on the influence of excavation method carried out in Swedish rock is the ZEDEX experiment at the Äspö HRL /Olsson et al. 1998/. The effects of excavation by TBM and Drill & Blast were compared in two parallel tunnels in a sparsely fractured granitic rock with moderate stress magnitudes at the Äspö Hard Rock Laboratory. It was concluded that the damaged zone around the TBM tunnel was in the range of 0.03 m, whereas the damaged zone around the Drill & Blast tunnel was in the range of 0.3 m in the crown and the walls, and 0.8 m in the floor. The wider Excavation Damaged Zone in the floor is caused by the blast design /SKB, 1999/. The ZEDEX experiment also concluded that the Excavation Disturbed Zone caused by stress redistribution with no noticeable irreversible changes was in the same order of magnitude independent of excavation method. This is as expected due to the fact that both tunnels had the same shape and dimensions, causing similar elastic responses to the rock mass, Figure 5-2.

As seen in Figure 5-2, the excavation by Drill & Blast in the ZEDEX experiment included a flat tunnel floor, and so called full face excavation. The Drill & Blast design used heavier explosive charges in the floor, which is normal practice in tunnelling.

The excavation for the APSE project in the Äspö Hard Rock Laboratory during year 2003 was specially designed to reduce damage in the tunnel floor. The main concern in the blast design of the new tunnel for the experiment was to develop as high stresses as possible in the tunnel floor for the planned experiment. The measures adopted included 1) increased height – width ratio of the tunnel to increase the secondary stresses for the prevailing maximum principal stress that are 2) a circular floor to create the highest stress concentration under the centre of the floor and not in the corners under the walls that is the case for a traditional horse-shoe shaped tunnel with a flat floor and 3) to limit the

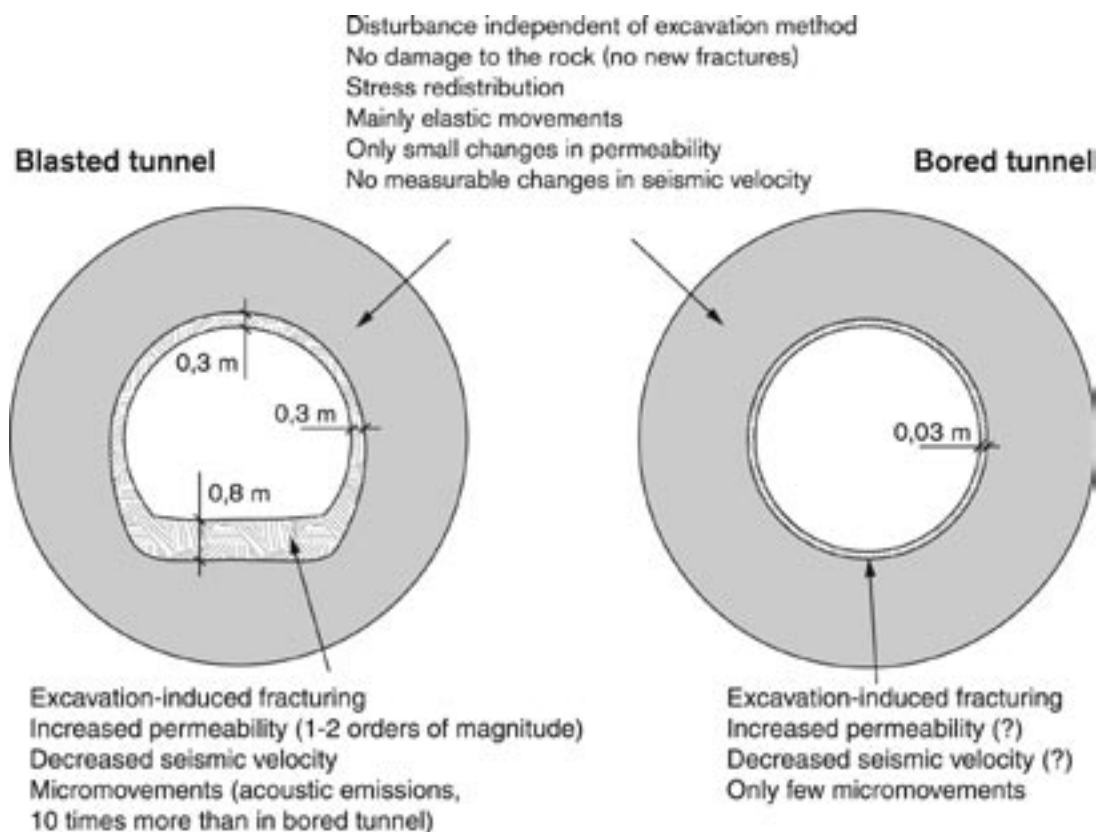


Figure 5-2. Summary of the ZEDEX results /SKB, 1999/.

excavation damage in the floor by very smooth blasting with the purpose to minimise the reduction of the Young's modulus that possibly could reduce the stress reduction in the floor /Olsson et al, 2004/. Limitation of excavation damage was achieved by excavation of a traditional horse shoe shaped tunnel, 5 · 5 m with separate blasting of the bottom bench, radius of 2.5 m, Figure 5-3.

Visible observations on the tunnel wall show that the transition zone between the top heading and the bench is more affected from the blasting than the rest of the wall, Figure 5-4. This is caused by the denser charge in the floor of the top heading, see Figure 5-3 and the extent of this damaged zone can be compared to the damage zone in the floor of the Drill & Blast tunnel in the ZEDEX experiment, Figure 5-2. The contour in the experimental area for the APSE experiment was however blasted with special precaution to minimize the damage, Figure 5-5.

Investigations in the floor of the APSE tunnel have so far shown a limited damage. Measurement of the sonic velocity perpendicular to cores shows a very small influence in terms of reduced compressional P-wave velocity /Staub et al. 2004/. The slightly decreased measured velocity is related to the occurrence of fractures sub-parallel to the floor that may have been sheared, Figure 5-6, but it is not obvious if some of the uppermost horizontal fractures (large angle to the cores) are natural fractures or new fractures induced by blasting, Figure 5-6.

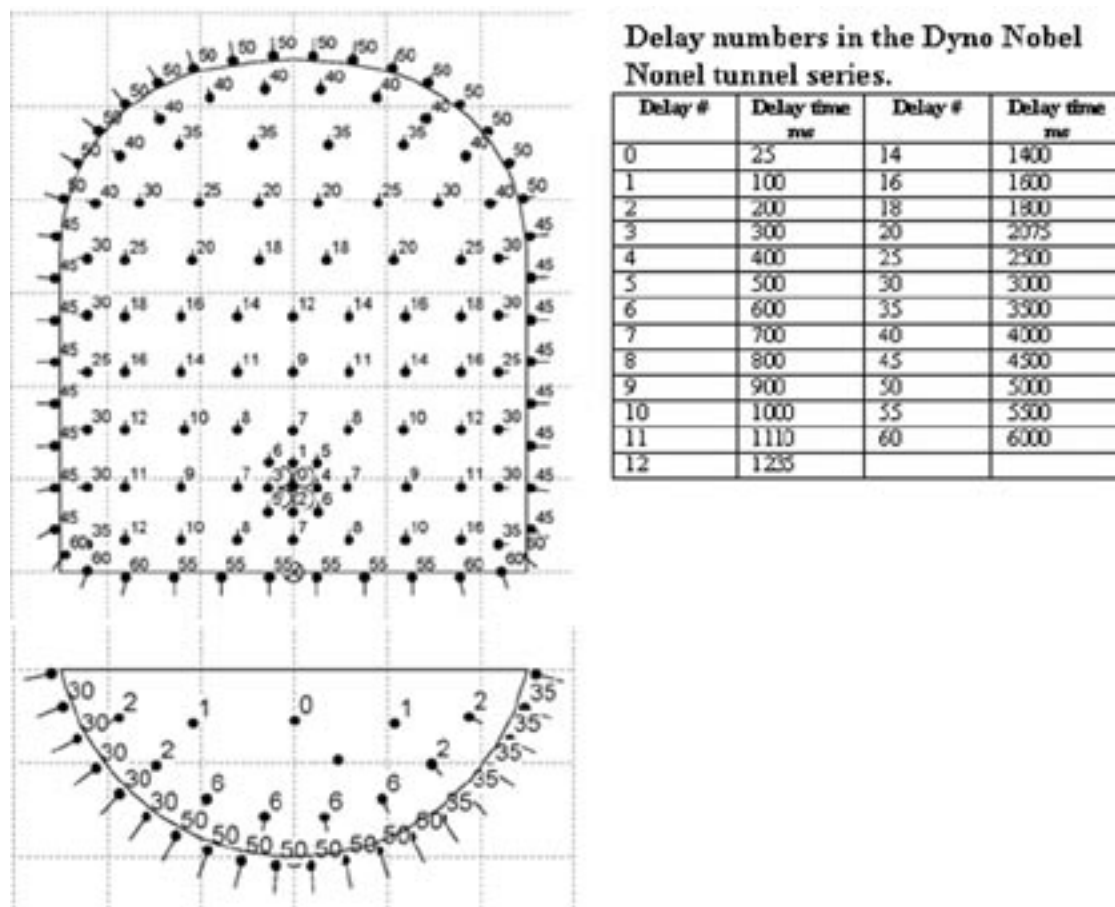


Figure 5-3. Initiation pattern for top heading and bench of the APSE tunnel /Olsson et al. 2004/. The delay numbers refer to time intervals shown in the table to the right.

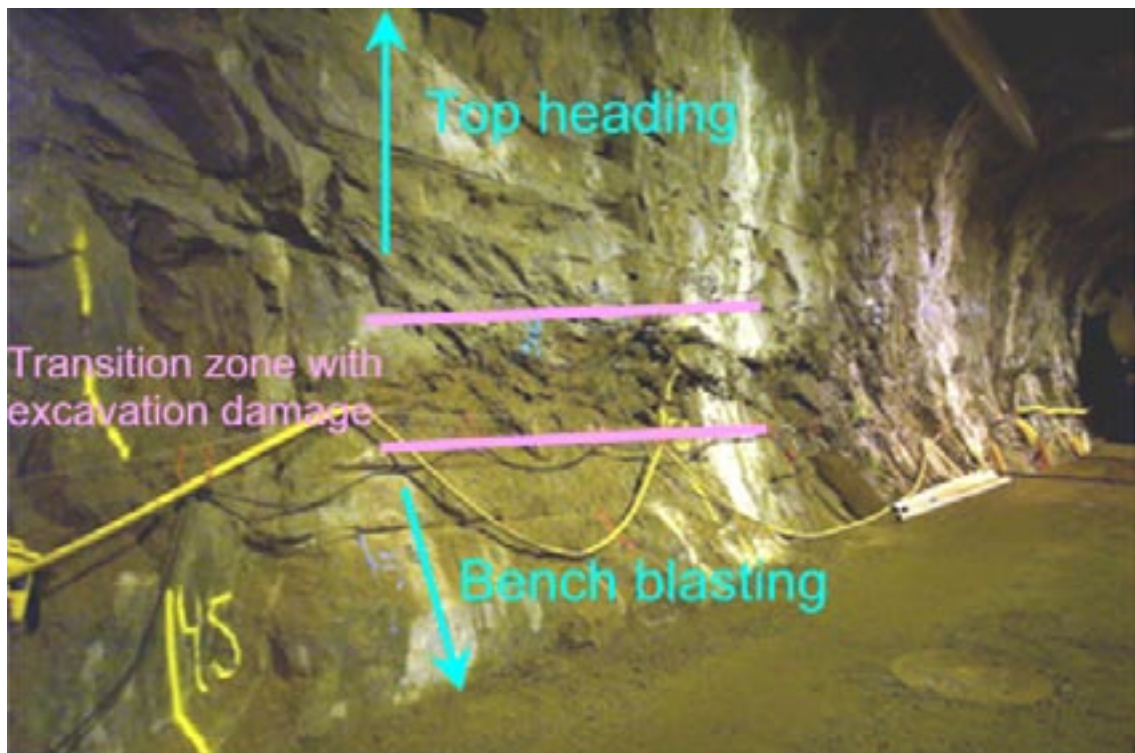


Figure 5-4. Visible signs on the lower part of the tunnel wall show the transition zone between the top heading and the tunnel wall.



Figure 5-5. The contour of the completed APSE tunnel.

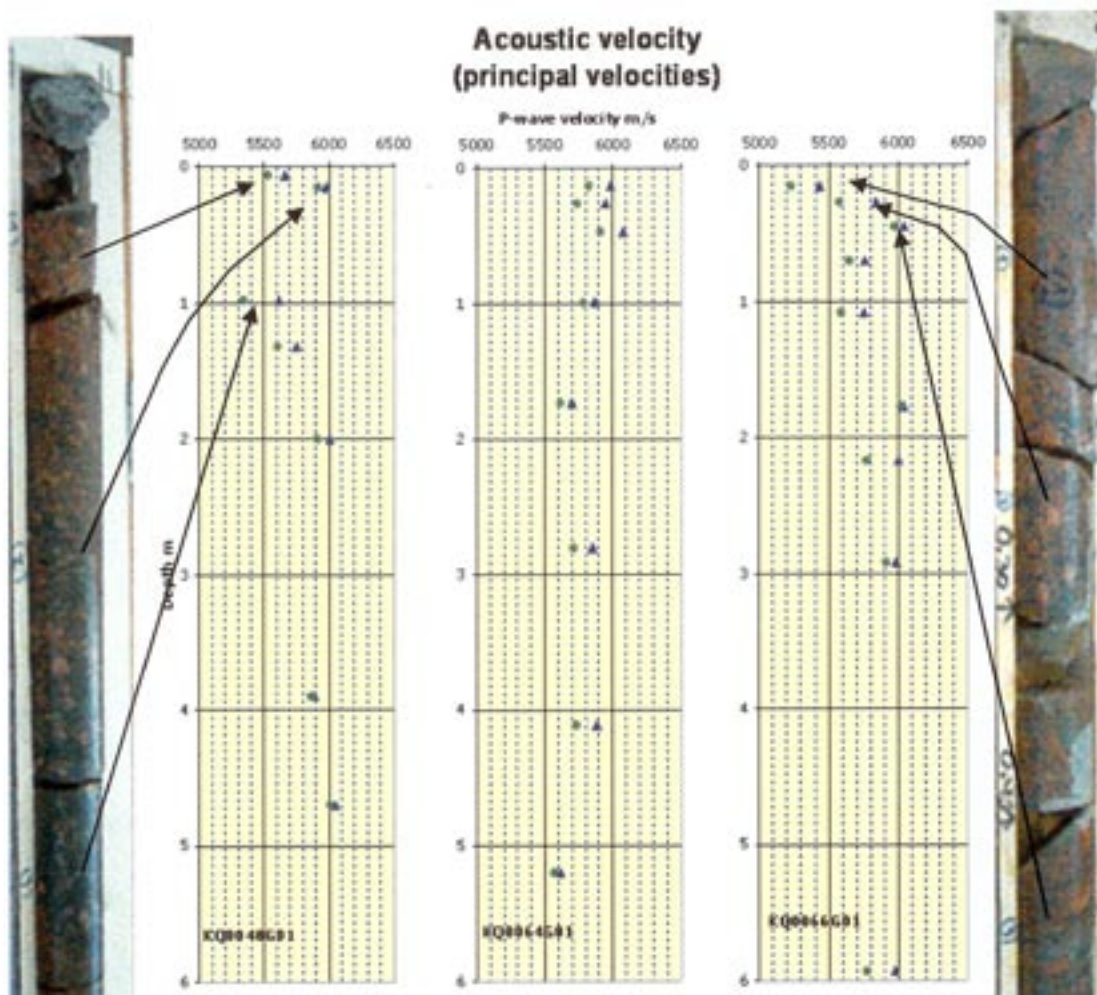


Figure 5-6. Maximum (blue triangles) and minimum (green dot) p-wave velocities perpendicular to three 6 m deep cores drilled out from centre-line of the floor. The upper meter of two cores are also shown, measurement locations are marked with arrows.

The investigations included also cross-hole seismic measurements of the velocity in a section, Figure 5-7. The experimental deposition hole in the centre of Figure 5-6 is in the centre of the cross section in Figure 5-7.

The investigations of the EDZ in the APSE tunnel will be carried out further with similar methods as in the ZEDEX tunnel. The preliminary results indicate however that the EDZ in the tunnel floor is significantly reduced by the proper tunnel- and blast design aiming at EDZ-reduction that was not an objective of the ZEDEX-experiment.

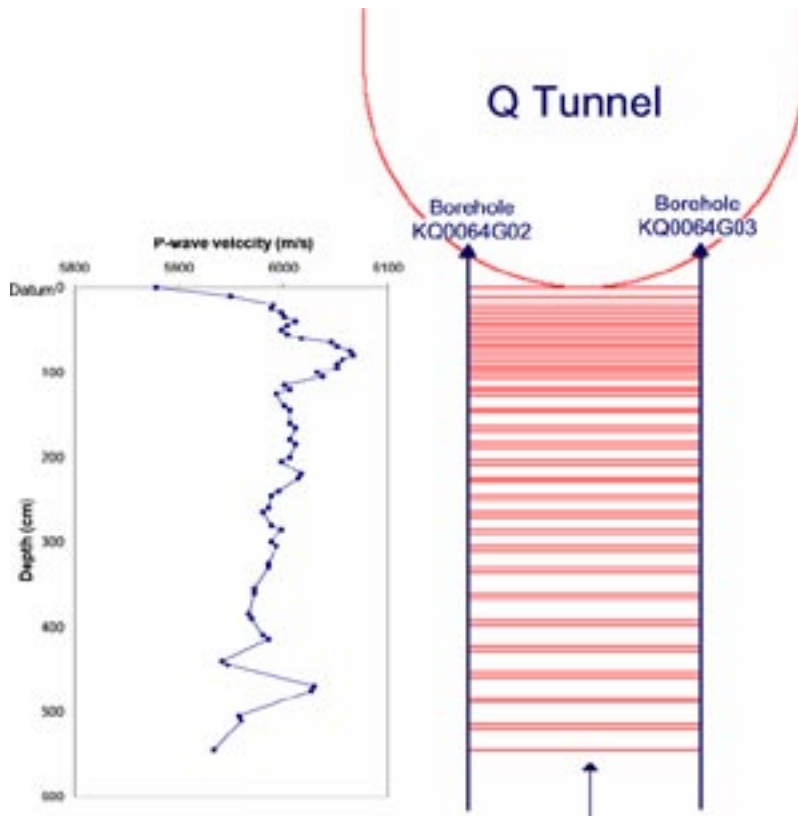


Figure 5-7. P-wave velocity in the floor of the APSE tunnel at chain gage 064 m.

5.3.3 The axial homogeneity of the EDZ

It may be assumed that whatever extent of the EDZ under any given site condition, the EDZ is continuous along a tunnel if the excavation method is continuous, such as for the TBM or if the stresses are continuous high enough to cause systematic spalling.

References for these kinds of EDZ's are from the TBM tunnel of the ZEDEX experiment at the Äspö Hard Rock Laboratory, respectively from the Mine-by test at the URL in Canada.

For Drill & Blast excavation it is not likely that the extent of EDZ is homogeneous along the tunnel. Estimation of the axial homogeneity of the EDZ caused by Drill & Blast is dependent on at least three factors, the charge density, the drilling precision and the local heterogeneities of the rock.

Although the specific charge (kg of explosives/m³ excavated rock) may be rather constant during tunnelling from round to the next round, the distribution of explosives within the round uneven. The tunnel floor is normally charged with more explosive resulting in deeper EDZ in the floor. Irrespectively of position of drill hole in the round, the bottom charge in each hole (the primer) is also normally the densest causing wider EDZ at the end of each round as well. If the bottom charge is too heavy in the perimeter holes more or less deep longitudinal cracks are created, but not along the complete length of the perimeter hole as this bottom charge only fills some decimetres of the perimeter hole.

Due to the geometry of drilling where a look-out is needed for the start of the subsequent round, the perimeter holes will not be drilled in the same positions as for the previous round, see Figure 5-8. Imperfections in drilling precision would also add further local damage and local variability of the EDZ. Finally, heterogeneities in the rock, such as

variable fracture density and orientations, different strength of rock types etc. also would cause local variability of the EDZ.

By the results from the ZEDEX-experiment and the APSE-experiment in progress it is reasonable to estimate that with proper tunnel and blast design and suitable site conditions (moderate in-situ stresses) the depth of the Excavation Damaged Zone will not exceed 0.3 m. Local variations may occur.

In consideration of all factors, variable specific charge along the round, variability in drilling and the local heterogeneities, it is not obvious that the fracturing in the Excavation Damaged Zone would be continuous along the axis of the tunnel.

The excavation of two 1.75 m diameter and 6 m deep experimental holes for the APSE experiment by the Shaft Boring Machine (see Figure 4-29) allowed observation of the Excavation Damaged Zone in the floor in a larger scale than by core mapping and by cutting out slices of rock from the EDZ. Observations in the two large holes in the floor of the APSE tunnel seem to confirm the results of the previous core mapping; the EDZ has a varying depth and is to a high degree controlled by the presence of the pre-existing natural sub-horizontal fractures under the floor. By earlier investigations it is known that there is in average one sub-horizontal fracture per meter depth in the APSE area. This fracture set is normally well sealed with rather rough and irregular fracture planes. It seems possible that this type of fracture has been mobilised to some degree by blasting and/or stress re-distributions and opened in the vicinity of the tunnel floor. The rock between the fracture and the floor has at least partly been damaged, Figure 5-9.

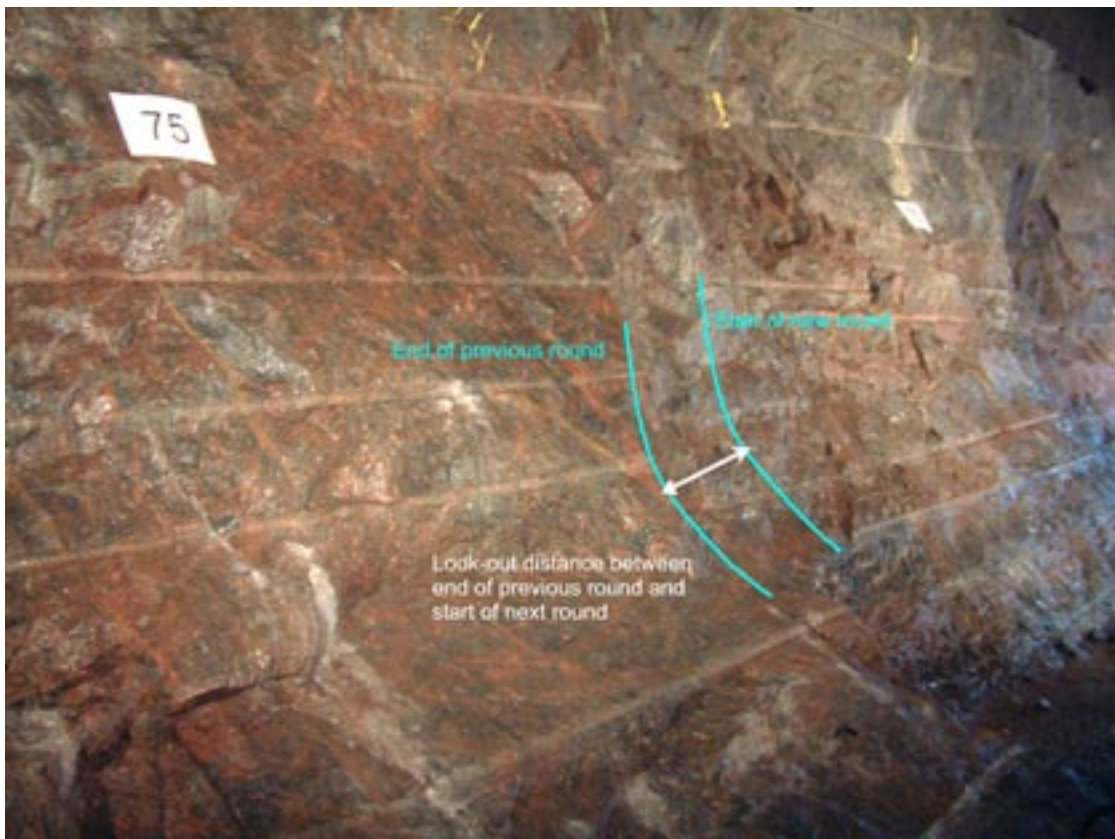


Figure 5-8. Photo of a section of the bench of the APSE tunnel. The tunnelling has been driven towards right in the photo. At the end of a round some of the perimeter holes deviate from the theoretical direction. The collar of the perimeter holes for next round are not all lined up with the previous holes.

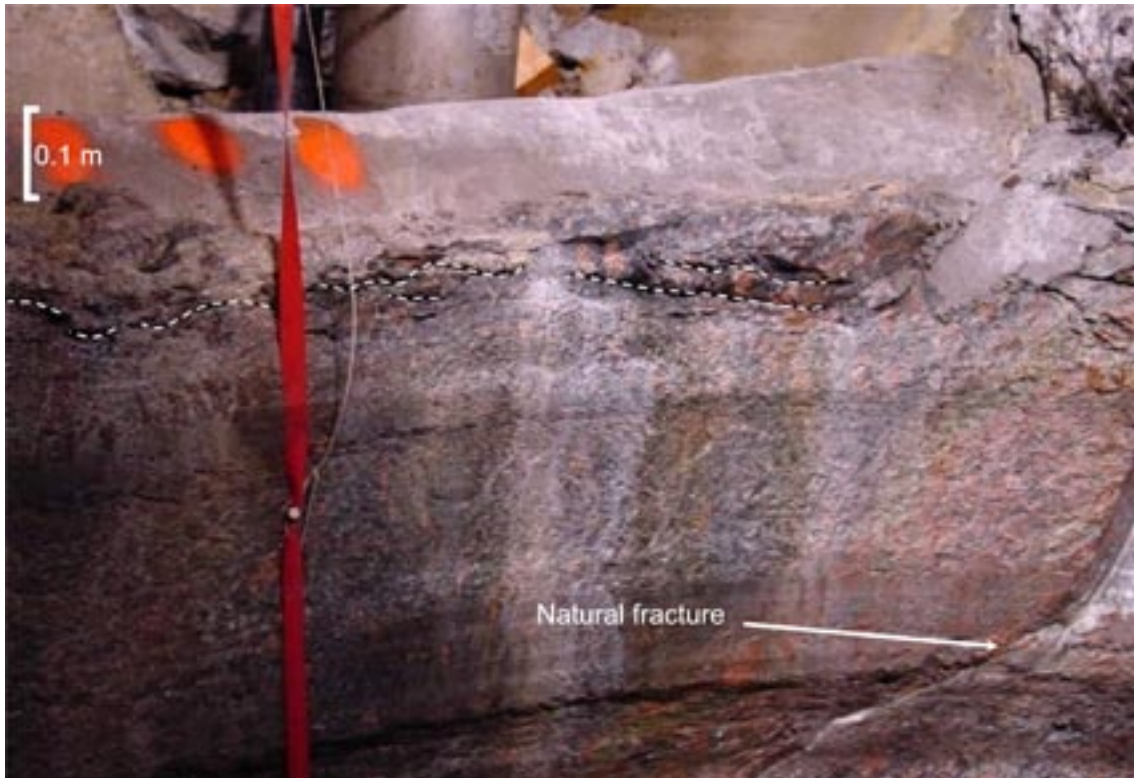


Figure 5-9. Photo of the upper part of a 1.75 m diameter hole in the floor of the APSE tunnel, close to the section showed in Figure 5-7. The concrete slab on the floor was casted for the set-up of the Shaft Boring Machine (see Figure 4-29). The slab is approximately 0.15 m thick. The damaged zone along the photo (covering 1 m) varies from 0.1 m (centre of photo) to 0.3 m (right side).

It is a likely hypothesis that natural fractures in the vicinity to and parallel with the tunnel contour may significantly contribute to the development of the Excavation Damaged Zone and in principle be independent of choice of excavation method. The Drill & Blast excavation however release more sudden energy that may mobilise displacements along the natural fractures. The reactivation of natural fractures aligned parallel to the tunnel may be one of the reasons that would contribute to the development of a continuous Excavation Damaged Zone over several rounds. However, the likelihood of long fractures that are perfectly aligned with the tunnels is very small of two apparent reasons: Long natural fractures are not very common in the “ordinary” rock mass, such as for example at the Äspö HRL. Secondly, there is normally a scatter in fracture orientation for any fracture set. This indicates that the number of natural fractures that would be mobilised within the Excavation Damage Zone of around 0.3 m is very small for the length of for example a deposition tunnel, around 250 m.

The methods to investigate the extension and continuity of the Excavation Damage Zone are discussed in the following.

5.4 Experiences of methods for investigations of the EDZ

The previous discussion on the understanding of the Excavation Damage Zone shows a number of investigation methods applied at the various experimental sites:

- Sonic velocity investigations of cores and around the perimeter of the tunnel.
- Direct observations through drilling of cores or cutting or drilling larger slots or holes to allow detailed studies of the damage.

The CNS Conference Workshop 1996 /CNS, 1996/ also highlighted the use of acoustic emission – microseismic monitoring technique (AE/MS) applied at the URL and Äspö Hard Rock Laboratory facilities and deemed this method to be useful in revealing the extent of damage that develops around excavations during construction. The AE/MS systems may have limitations to measure all events exactly as they occur during driving a TBM or at the blast moment for a Drill & Blast operation due to noise generated by the TBM and the high energy release during blasting. However, just after the blasting, or when the TBM is stopped, events can be picked up with a proven good possibility for source locations. If the registered events cluster in planes this is a strong indicator for displacements on pre-existing fractures, or for crack growth; the nature of anomalies has to be studied in detail with more direct observational methods to find out whether pre-existing discontinuities are reactivated or extended or if new fractures are developed.

The most effective way of studying the properties of the Excavation Damaged Zone is by hydraulic tests that also would be the most meaningful, as the axial hydraulic conductivity of the EDZ is the issue. Hydraulic testing of the EDZ is however not simple. Any attempt done to measure hydraulic characteristics of the EDZ in boreholes from the tunnel, face the problems of transient testing under partly unsaturated conditions that may be run with unknown hydraulic boundary conditions. Whatever result achieved may suffer of any of these unknowns, as well as the uncertainty in how homogeneous the EDZ would be along the tunnel.

Controlled large-scale experiments for hydraulic testing of the EDZ by backfilling and plugging of a tunnel seem to be the most promising method for large scale hydraulic characterisation of the EDZ. The Plug and Backfill Experiment at the Äspö Hard Rock Laboratory has the objectives to develop and test different materials and compaction techniques for backfilling of tunnels; to test the function of the backfill and its interaction in full scale with the surrounding rock and to develop technique for building tunnel plugs and test the function of these plugs. The experiment is carried out in the Drill & Blast tunnel of the ZEDEX experiment. The preliminary reporting so far after full saturation shows that the hydraulic conductivity is governed by the EDZ in the floor with the observed depth of 0.8 m, see Figure 5-2.

A similar experiment – the Tunnel Sealing Experiment (TSX) has been carried out at the 420 m level of the URL, Canada /Martino, 2003/. The results from a previous Blast Damage Assessment (DBA) study at the 240 m level are compared to the TSX study. Both tunnels are excavated by careful Drill & Blast. The BDA tunnel is located in a moderately high stress environment, and the TSX tunnel is located in a high stress environment. Hydraulic conductivity tests in both tunnels have allowed for a comparative study of the EDZ properties. The work concludes a similar inner EDZ of 0.1 to 0.2 m caused by the Drill & Blast method to be similar for both tunnel with an outer EDZ believed to be caused by stress concentrations that is more extensive for the TSX tunnel. The hydraulic transmissivity of the EDZ of the TSX tunnel is reported to be 10^{-10} to 10^{-11} m²/s, locally 10^{-9} m²/s over the test length around 20 m. The background transmissivity at the TSX site is reported to be 10^{-13} to 10^{-14} m²/s. The TSX tests show that the axial transmissivity can be limited by precautions in the blast design.

The CNS workshop 1996 /CNS, 1996/ pointed out the good progress that had been made in advancing the rock mechanics modelling to the point of being able to predict the location and extent of the EDZ around openings in hard rock. This has more recently been discussed also by /Martin et al. 2001/, and further demonstrated during the design of the APSE experiment /Andersson, 2003/.

Of specific interest for determination of the EDZ caused by Drill & Blast is the experience from the tunnelling for the APSE experiment /Olsson et al. 2004/. The indirect measures such as quality control of drill hole precision, the charging and vibration control can not describe the extent of the EDZ, but the individual activities in the tunnelling cycle can be monitored by a systematic quality plan and corrective measures taken to minimize the systematic errors that may have impact on the extent of the EDZ.

5.5 Implications of selection of excavation methods with respect to the EDZ

The reference from the ZEDEX experiment with the very limited extent of the EDZ (Figure 5-2) is reliable information for tunnelling with TBM in mainly moderate stress conditions and with primarily elastic response due to the excavation. The outcome from the URL Mine-by test indicates that if stress magnitudes are significant high to cause spalling, the development of an EDZ would be controlled by this process rather than by the choice of excavation method. The paper by /Martin et al. 2001/ discusses the limitation with a circular opening and the low flexibility of a TBM if stresses are sufficient high to cause significant spalling problems. Such conditions may end in larger EDZ, for example more similar to the extent reported at the TSX experiment at the URL.

The experiences from both BDA and TSX tunnels at the URL, as well as the APSE tunnel at the Äspö Hard Rock Laboratory show that a significantly less pronounced EDZ is developed with proper drill and blast design where precautions are taken to really limit the excavation damages. The hydraulic homogeneity of the EDZ for a D&B tunnel may be very limited, according to the reporting of the TSX experiment. The studies so far from the APSE tunnel at the Äspö Hard Rock Laboratory indicates that the opening of existing fractures as one of the processes that are involved in the development of the EDZ may have limited importance due to small likelihood of long fractures along the perimeter.

However, if the stresses are high enough to cause significant spalling a hydraulically continuous EDZ could be formed, independent of excavation method. The rock mechanics predictive capability is however sufficient to design for minimum effects of the EDZ, based on the good understanding of the site and selection of proper tunnel shapes as well as appropriate Drill & Blast design.

6 Evaluation of alternatives for deposition tunnels in comparison to objectives

The technical descriptions in Chapter 4 supplemented with detailed data are used to compare the excavation methods for a range of factors with the following objectives (see Section 3.3) in mind:

- Low radiation dose after closure (ALARA⁶).
- No accidents for employees and contractors during construction and operation (incl. ALARA).
- Small environmental impact during construction and operation.
- Sustainable management of natural resources.
- Low Net Present Value cost of construction and operation.
- Short duration of construction period from start of excavation to start of initial operation.
- High flexibility.
- Low project risks.

6.1 Long term safety after closure

An important issue is whether long term safety is dependent on the choice of excavation method. Some aspects can be studied in a generic sense but some aspects may need truly site-specific data for a precise answer. We discuss safety matters in a thermal, hydrogeological, mechanical and chemical (THMC) perspective where it is obvious that the thermal evolution would be independent on choice of excavation method. The waste, the buffer, the deposition hole and general layout would be similar for different excavation methods.

For evaluation of hydrogeological processes there are several factors to treat, like creation of possible flow paths due to an excavation failed, excavation damaged or excavation disturbed zone or would there be different quality of backfill due to the excavation method.

6.1.1 The excavation failed, excavation damaged and excavation disturbed zone

As described in the previous chapter several processes may be important for the development of the Excavation Damaged Zone, i.e. “the part of the rock mass closest to the underground opening that has suffered irreversible deformation where shearing of existing fractures as well as propagation and/or development of new fractures has occurred.”

⁶ ALARA is a common principle phrased as “keeping the radiation doses to humans As Low As Reasonably Achievable, economic and social factors taken into account” /IAEA, 2002/.

For high stress magnitudes, spalling may occur and if such a process develops it would be decisive for creation of axial conductivity along the deposition tunnel. Spalling can to some extent be mitigated by the tunnel shape and from this point of view the Drill & Blast method offers advantages as the method readily can excavate any shape while the TBM and RBM only creates the circular shape.

Drill & Blast is not a “continuous” method like the TBM or RBM but variability of charging, drilling and the heterogeneity of the rock would overall create an EDZ where the extent is varying along the perimeter as well as along the tunnel. An ongoing study at Äspö Hard Rock Laboratory shows that reactivation of pre-existing natural fractures is an important mechanism to develop the EDZ. Natural fractures that both are long and also parallel to the deposition tunnel however, would be very rare and the axial flow can then only be developed by a connected fracture network where several mobilised fractures interact within the limited extent of the EDZ. It is assumed that EDZ can be limited to around 0.3 m even in the tunnel floor with proper tunnel design and proper blast design whereas the extent is down to a few centimetres for a TBM or a RBM.

The conservative approach right now, based on the findings from tests at Äspö Hard Rock Laboratory and tests at URL in Canada is to assume that there is an increased hydraulic conductivity in the EDZ, however we cannot substantiate that the increase is solely due to the selection of excavation method. This difference would however be superseded in importance if an excavation failed zone develops, where blocks or slabs completely detach from the rock mass. This issue is site-specific.

The importance of possibly increased hydraulic conductivity along the deposition tunnels will be further studied in the ongoing SKB long term safety assessment SR-CAN where present reference design and site-specific, preliminary data from the site investigations at Forsmark and Oskarshamn are utilized.

6.1.2 Issues related to backfill

SKB is presently studying a suite of possible backfill alternatives, Figure 6-1 and it is of interest to evaluate possible excavation influences on backfill quality. As explained before tunnel shape is different for mechanical excavation and for Drill & Blast. Also the smoothness is different. The former methods yields smooth surfaces while the Drill & Blast perimeter is uneven. In addition a step of a few decimetres would be created between the end and start of the next round, due to the need for look-out, see Figure 4-4. The overall conclusion based on reasoning is however that the difference in quality would be insignificant in relation to other factors that would influence the backfill quality.

For the concept A and B a bored tunnel is deemed better, but a squared Drill & Blast tunnel would be preferred for concept D and also for concept E. Discussion on backfill is also included in Section 6.3.

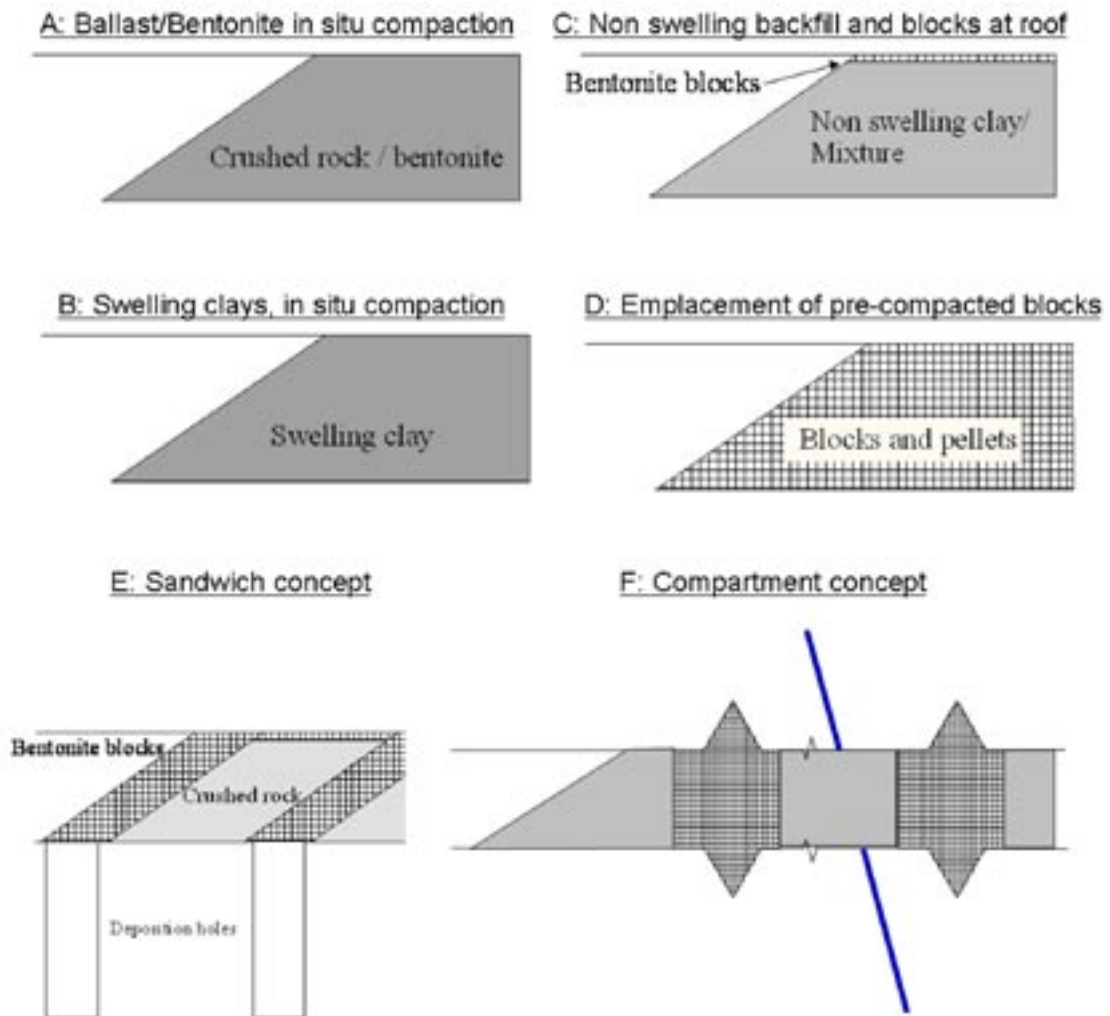


Figure 6-1. Outline of backfill alternatives.

6.1.3 Construction and stray materials

General experience is that less rock support is needed for mechanically excavated tunnels. While the rock quality is assumed to be good for the deposition tunnels this difference is not important and we can expect that types and amounts of rock support is method independent. For all excavation methods there will be stray materials as steel, hard metal and hydraulic oils. For Drill & Blast we can also assume explosives and detonators. However most of the material will accumulate in the muck in the floor of the tunnel. As the tunnel is cleaned before backfilling and the dirty muck is not re-used to produce the backfilling material the difference between excavation methods in this respect is of no consequence.

Additional discussions on stray materials are found in Section 6.3.

6.2 Occupational safety during construction, operation and closure of the facility

6.2.1 Fatalities and accidents

Underground work is in general still more dangerous than working in average industry. However there are many workplaces in industry that are more dangerous than the average workplace in underground. The accidents are not so much related to rock conditions, but rather to traffic accidents, sliding/slipping etc. The Swedish statistics of underground health and safety (based on mining operations) does not like in many other countries, distinguish in type of work conducted so it is not possible to explicitly discuss accidents related to the typical activities connected to excavation.

As a first estimate it is assumed that underground mining statistics is relevant for a repository operation as both would deal with work underground, heavy machines etc. The number of injuries/accidents is around 20 per million work hours and fatalities around 0.06 per million work hours. Violation of safety regulations is a common cause for accidents.

While the statistics is inconclusive with respect to type of work we can only perceive that Drill & Blast and mechanical excavation present different risks, which once understood can be mitigated by design, regulations, and education. As a general remark it is assumed that horizontal reaming would be a comparatively safer excavation method as the excavation mainly is with less people in the newly excavated drift.

6.2.2 Heat, noise, dust and gases

Heat during excavation is only of concern for TBM operation. Most of the heat generated is actually transported out with the fragmented muck. Based on calculations around 700 kW of air heat loss is generated. However heat is no concern with proper ventilation; an air velocity of 0.5 m/s (15 m³/s) is deemed sufficient.

Both a TBM and the drilling rig for Drill & Blast generate noise at around 100–110 dB(A) air borne noise in hard granitic rock, but the operators themselves are in cabins that are isolated with much less noise, around 70 dB(A) for a TBM.

All excavation methods generate dust, either due to the blasting/excavation or due to the mucking, haulage, but dusting is mitigated by water spraying and ventilation as standard practice.

Drill & Blast generates toxic gases (see Table 6-1) during excavation and ventilation is needed to remove toxic gases, possible radon and exhaust gases from underground vehicles.

6.2.3 Adverse conditions

Several adverse conditions may be anticipated but due to the good rock conditions assumed in the deposition tunnels, they would not be related particularly to the rock. One important issue however is fire hazards and fire safety. In the Swedish mines there are around 25 fire incidents every year (<http://www.mining.se/pdf/BK02.pdf>) with fires in vehicles being the most common cause. The amount of flammable materials for different excavation equipments are shown in Table 6-2.

Table 6-1. Gases from blasting developed from three different types of explosives. All data are taken from Dyno Nobel data sheets. The volume of gas is another important factor that plays a role in determining how effective an explosive is. Data in this table refers to the volume of the gaseous reaction products created under detonation. It is given in litres under standard conditions, with a temperature of 0°C and a pressure of 760 mmHg.

Components	Formula	SSE-Emulsion	ANFO	Dynamite
Water	H ₂ O	71%	64%	52%
Nitrogen	N ₂	22%	27%	25%
Carbone dioxide	CO ₂	7%	9%	18%
Carbone monoxide	CO	13 L/kg	23 L/kg	12 L/kg
Nitrous gases	NO _x	0.2 L/kg	29 L/kg	24 L/kg
Total volume		950 L/kg	975 L/kg	890 L/kg

Table 6-2. Flammable materials for different equipment.

Equipment	Plastics/ Rubber [kg]	Lubricants and fuel [kg]	TOTAL (rounded) [kg]
Drill & Blast			
Drilling rig	1,460	720	2,200
Loader (Volvo BM 150)	2,530	486	3,000
Mechanical excavation			
TBM	4,500	2,400	6,900
Horizontal reaming	975	1,500	2,500

For any excavation method, safety regulations must be clear and known by all personnel, including the handling of explosives, rescue plans etc. Furthermore, ventilation systems must be designed for the removal of gases in case of fire. No storage of explosives should take place underground, and no more explosives than what is required for one round should be transported underground. The system when the emulsion is sensitized as an explosive at the site offers some advantages in this respect. All vehicles used underground may be equipped with automatic fire fighting equipment like sprinklers to minimize the fire hazards.

In our study, we have found no record that a TBM has been on fire or any records of fires on raise-boring equipment.

6.3 Environmental impact and sustainable management of natural resources

There are several factors to discuss with respect to environmental impact and sustainable management, but we here focus on backfill properties, stray materials and whether Life Cycle Analysis would have an impact on selection of excavation methods. According to the Swedish Environmental Code (Ds 2000:61) Best Available Technology should be used (Chapter 2, Section 3): “Persons who pursue an activity or take a measure, or intend to do so, shall implement protective measures, comply with restrictions and take any other precautions that are necessary in order to prevent, hinder or combat damage or detriment to human health or the environment as a result of the activity or measure. For the same reason, the best possible technology shall be used in connection with professional activities.”

6.3.1 Backfill issues

The rock muck produced is preferably used for producing the backfill needed but different excavation methods will produce different particle size distributions, Figure 6-2.

The important conclusions from Figure 6-2 are:

- Mechanical excavation produces more or less directly a suitable material for manufacturing backfill with crushed rock. Simple processing like sieving may be sufficient.
- Drill & Blast muck would need downstream processing (crushing, milling) to be suitable for the backfill. By processing “any” particle size distribution is viable.

What is remarkable is that very few data seem to exist for the distribution of particle sizes for tunnel blasts. For mechanical excavation, the distribution is rock specific but also dependent on cutter spacing etc.

The volume of muck produced is much bigger than needed for manufacturing the backfill and the surplus generated is used for other purposes if possible. What is not needed for backfilling or can be marketed will be put into a land deposit. The market for surplus muck is truly a local factor, but it is assumed that the Drill& Blast muck would be easier to market than the muck from mechanical excavation.

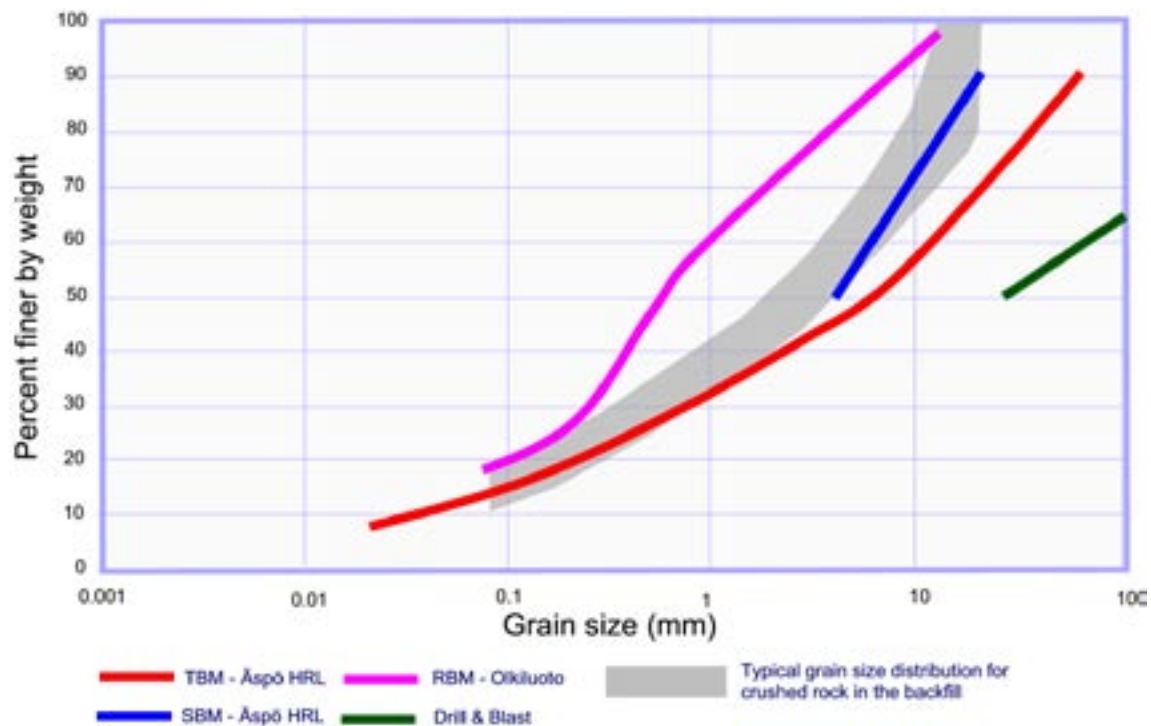


Figure 6-2. Particle size distribution of different excavation methods. The TBM-Äspö is from the excavation of the 5-m-diameter tunnel. SBM-Äspö and RBM-Olkiluoto are from excavation of deposition holes. Very few data seem to exist on fragmentation for tunnel blasts and the green line is illustrative only. The grey area is typical particle size distribution of crushed rock used for the backfill tests at Äspö.

6.3.2 Stray materials

Spillage of oil for lubrication and hydraulic systems would occur for any excavation method, but different methods would have different opportunities by design to mitigate spillage. Drill & Blast excavation would in addition to oil spillage also include spillage of explosives, detonators etc.

Spillage of oil is very much a matter of preventive maintenance, age of equipment, operator skill etc. Some information on typical spillage has been gained from Drill & Blast in mining and from construction project with TBM and a typical spillage would be around 0.01L/m³ of excavated rock (around 10 m³ of hydraulic oil for all deposition tunnels) and less for lubrication grease. The major share of the spillage will assemble at the tunnel floor and can be removed from the tunnel before backfilling.

To mitigate environmental impacts by spillage the following actions viable:

- Design of equipment where the machines are equipped with trays that collects oil spillage.
- Absorbing materials at the rigs to be used at major spillages.
- Selection of oil that is degradable.

The Swedish National Testing and Research Institute (SP) in consultation with the industry have provided environmental guidelines for use of hydraulic oils and lubrication greases. Most of the environment-friendly oils are based on biological oils and alcohols from the mineral oils. True biological oils based on rapeseed oil degrade in nature within a few weeks, synthetic oils within a few months and mineral oils within several years. It is assumed that SKB as Owner stipulates specifications on requirements for use of oils in the repository. The contaminated rock muck will be cleaned in accordance with environmental requirements but it is not likely this material will be used to produce the backfill.

Excavation with Drill & Blast is associated with some spillage of explosives. Spillage from charging depends on how explosives are handled and on the type of explosive used. If emulsions and novel charging technology is used, spillage from charging may be reduced to less than 1%. However not all explosives may detonate due to disturbances in the initiation of the holes, dead-pressing effects, breakage of neighbouring holes. The total not detonated amount of explosives may be as high as 10–15% of the total charged weight i.e. around 200 tonnes of spillage of explosives for excavation of all deposition tunnels. The environmental concern is the emission of nitrogen to water, the main compound in explosives. The spillage will assemble in the tunnel muck, but around 1/3 to 1/2, depending on amount of backwash water used, may be released with the drainage water as the compound is soluble in water. Only a very small amount of nitrogen is forming nitrous gases (NO_x). During the excavation of the deposition tunnels, the amount of nitrogen emitted to water may be approximately 1–3 tonnes a year. The drainage water can be processed to reduce the amount of excess nitrogen.

6.3.3 Life cycle inventory

Sustainable management of natural resources is a key principle in the Swedish Environmental Code. The sustainability of the excavation methods have been assessed with a Life Cycle Inventory (LCI) method based on ISO TR14025 and ISO 14040-41. LCI is used to make a holistic view of the environmental impact from “the cradle to the grave”. The specific Swedish rules for Environmental Product Declarations (MSR1992:2) are prepared by the Swedish Environmental Management Council. In addition calculations have used the guidelines in Product-Specific Requirements 1998, rev 2, used for production of electricity and for district heating.

Several assumptions are of course used for the calculations. The life cycle here is limited to manufacturing steel and explosives, transporting these materials to the site, crushing and milling of blasted rock, pumping of water used in the tunnels to the surface level and the use of electricity for excavation and mucking. We here compare the energy usage, usage of steel for manufacturing and operation, emissions of carbon dioxide (CO₂) and pollutants (NO_x, SO₂, particles). Environmental impact of energy production is based on the present mix of electricity production in Sweden.

Table 6-3 is an outline compilation of resources needed to for different excavation methods.

Energy for the specific excavation work is based on supplier estimates, but only for the excavation work (drilling) and mucking. Energy for haulage or for ventilation etc is not included. Energy use for the most demanding method is 42 GWh. As comparison the yearly energy consumption (2003) for CLAB and SFR and Äspö HRL is altogether around 27 GWh.

The muck from the Drill & Blast would need milling to produce the preferred 0–5 mm particle grain size and this energy needed is estimated to 5 kWh/ton of rock. Steel usage includes the steel in the equipment, steel for consumables and replacements of equipments. Consumables of steel for Drill & Blast are in the order or 0.15 kg/m³ of rock (drill steel, drill bits etc), for raise-boring around 1.35 kg/m³ (drill string and cutters) and for TBM around 0.35 kg/m³ (cutter rings and cutter hubs).

The environmental impact is evaluated in a simplified way in Table 6-4.

Table 6-3. Overview of specific data of consumption of resources.

Issue	Excavation of deposition tunnels		
	Drill and Blast 800,000 m ³	Raise-bore machine 1,000,000 m ³	TBM 1,000,000 m ³
Energy for the specific excavation work and for mucking (GWh)	5	42	20
Energy for processing muck for backfill material (assume 2/3 of the backfill volume is crushed/milled rock) (GWh)	7	0	0
Comparative energy usage for excavation (GWh)	12	42	20
Steel usage, manufacturing, operation (tonnes)	240	2,010	800
Explosives (tonnes)	1,760		

Table 6-4. Environmental impact of different excavation methods based on Life Cycle Inventory. Evaluation chart. Alternatives are ranked 1–3 where 1 is the most environmental and 3 the least environmental alternative.

Excavation method	Greenhouse gases (94–97% CO ₂)	Pollutants (NO _x)	Pollutants (SO ₂ , particles)	Electricity use at excavation	Steel consumption
Drill & Blast	2	3	1	1	1
Raise-boring	3	2	3	3	3
TBM	1	1	1	2	2

For Drill & Blast around 40% of the greenhouse gases are tied to manufacturing the explosives and the gases generated at the blasting. Around 20% is due to production of steel and the remaining 40% mainly for production of electricity used for excavation, milling and for pumping the production water up to the ground surface. For the TBM and raise-drilling approximately half of the emissions of CO₂ are for the manufacturing of steel and the other 50% for due to production of electricity for excavation.

For any selection of excavation method, the excavation of the repository would be a minor part of the total production of greenhouse gases from the SKB operations as the operation of the repository, canister fabrication, transport of bentonite and transports with the SKB ship Sigyn would be the most important sources, /Setterwall, 2004/.

6.4 Schedules and costs

For Drill & Blast it is assumed that several tunnels faces are available at the same time. For other methods we assume that the equipment can be moved from one excavated tunnel to the next tunnel face without significant delays.

Based on several tunnel faces available the corollary is that the repository master schedule is independent of choice of excavation method as enough production capacity is created to meet the target capacity. Based on the typical performance for different technology it is assumed that Drill & Blast and TBM excavation need only one set of equipment to meet production target, but that horizontal reaming of deposition tunnels would necessitate three reaming units.

Direct costs for excavation and mucking are compared in Table 6-5 with the following comments:

- Direct excavation costs include costs of capital, manpower and consumables for each excavation method. Capital cost is calculated as investment cost for excavation equipment and one loader (except for TBM and horizontal reaming where the lucking is by the TBM or by slurry pumps) divided by expected durability of the equipment in years averaged over the target production. Manpower is calculated from the direct manpower needed to excavate and muck, but general management transport, grouting and rock support is not included (assumed quite similar for the deposition tunnels). Consumables include explosives (if applicable), drill rods, drill bits, cutters, wear parts etc.
- The circular tunnel area for horizontal reaming and TBM is larger than for the reference design so increased backfilling costs are to be included for these methods.
- The reaming alternative necessitates an extra service tunnel.
- The horizontal reaming and the TBM produce tunnel muck that after simple processing more or less can be directly used as backfilling material. The Drill & Blast operation does not produce enough of fines (0–5 mm) so the muck would probably need crushing/milling to be used as backfilling material.
- It is not possible to calculate at this stage if the general operating costs are different for different excavation methods.

Table 6-5. Comparative costs for different excavation technologies. As the comparison is illustrative only, Net Present Values are not calculated. With NPV calculation, backfill cost is much lower.

Issue	KBS-3V Deposition tunnel		
	Drill and Blast 800,000 m ³	Mechanical Excavation Raise-bore machine 1,000,000 m ³	TBM 1,000,000 m ³
DIRECT EXCAVATION COSTS	210 MSEK	860 MSEK	315 MSEK
Increased costs for backfill compared to reference design	–	200 MSEK	200 MSEK
Extra excavation and backfilling of service tunnels	–	400 MSEK	–
Cost differences for producing backfill material	80 MSEK	–	–
SUM OF COMPARATIVE COSTS (ROUNDED)	300 MSEK	1,500 MSEK	500 MSEK

A general observation is that the cost differences mostly relates to differences in layout rather than differences in direct excavation costs. The direct excavation costs for the horizontal reaming are quite high due to need for three equipments and high costs for consumables like cutters etc.

6.5 Flexibility, risks and opportunities

Flexibility is the ready capability to adapt to new, different, or changing requirements and here we mean that

- a previous decision can be reversed by simple means or a decision can be referred for a certain time,
- there are alternate options available during the decision-making or,
- that new insights and information which emerge during the implementation can be incorporated to develop the most appropriate technical solution.

Flexibility here is discussed from a range of aspects:

Adaptation of layout to the bedrock conditions

Adaptation of layout is seen in two perspectives 1) Preparation of a generic reference design that is fit for the excavation method and 2) a “real design” that is fit for the excavation method and the rock conditions at hand.

The generic layout of the reference design is achievable for most excavation methods (see Chapter 4.8), but would not be practical for all methods as TBM excavation would be more favourable if deposition tunnels are long >> 300 m rather than short (< 300 m). An example is shown in Figure 6-3, where around 800 m long deposition tunnels are excavated and later cut by a secondary main tunnel excavated to simplify deposition logistics.

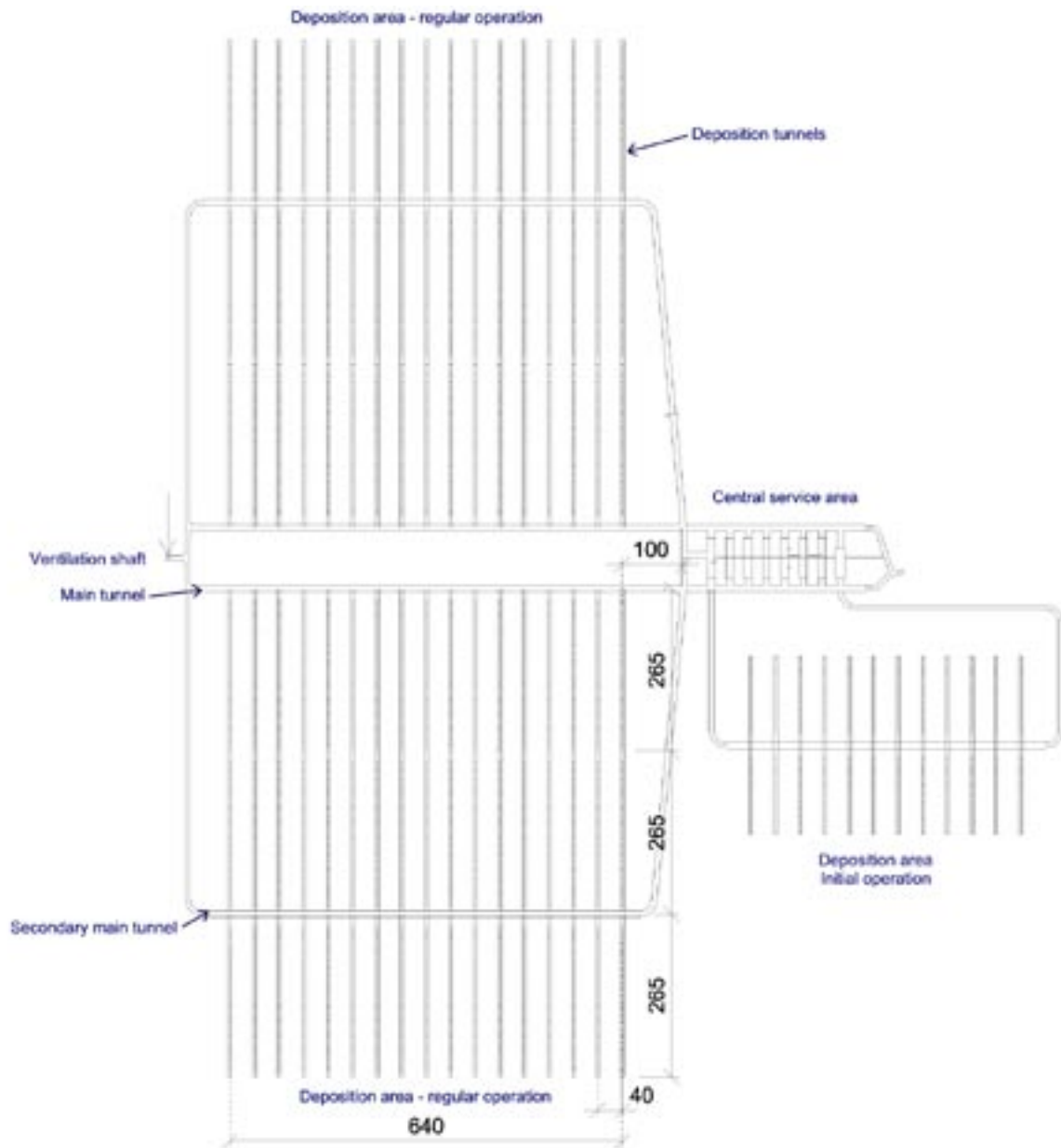


Figure 6-3. Long deposition tunnels designed for TBM excavation.

The “real design” should account for the rock conditions; fracture zones are to be avoided and there may even be a “respect distance” to the fracture zones that makes the available rock areas for deposition much patchier. An example from Finland is shown in Figure 6-4 based in on the surface site investigations. With more details from the underground characterisation it is likely additional minor fracture zones are included that would necessitate minor adjustments. The patchier design would mean shorter tunnels and need for more curves in the design and such a design is of course less suitable for e.g. TBM compared to Drill & Blast.

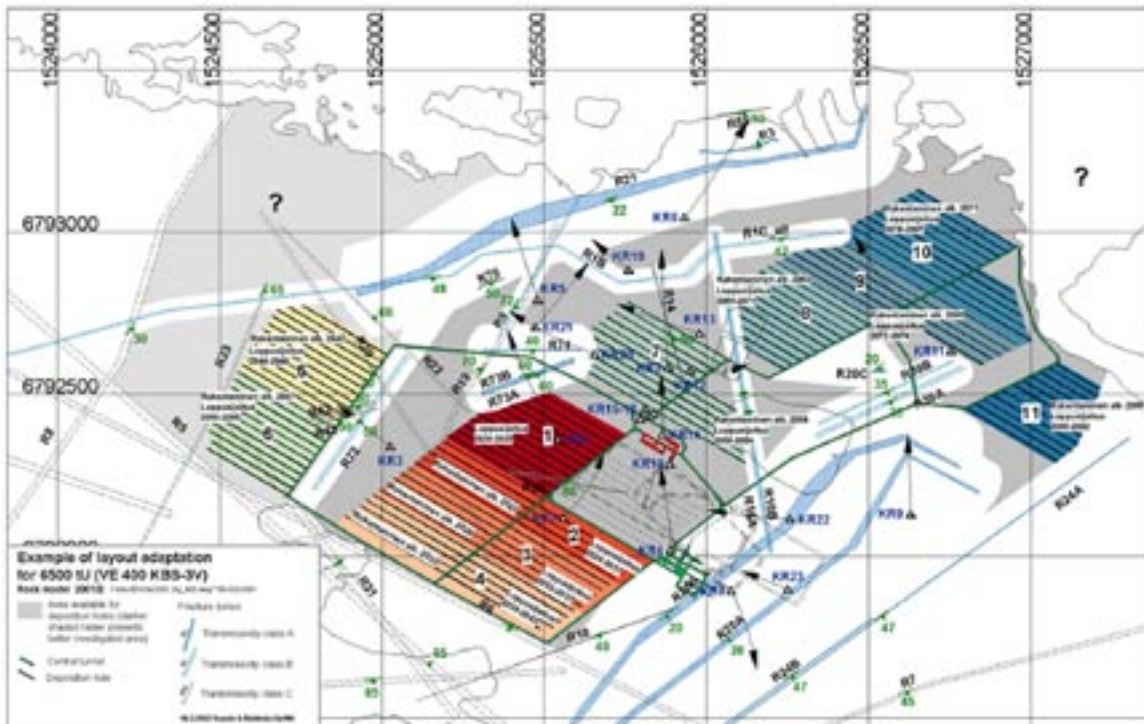


Figure 6-4. Example on layout adaptation of a KBS-3V repository to actual rock conditions. /Malmund et al. 2003/.

Investigations of rock conditions during construction

As described earlier in the report, rock characterisation would be a parallel activity to construction in general. For the deposition tunnel, it is assumed that at least one horizontally cored hole is drilled and investigated prior to construction. It is also foreseen that cross-hole measurements over several deposition tunnels will take place during construction. In connection with construction, mapping of geology in the tunnels will be made for reasons of occupational safety, planning construction and to decide most suitable locations to excavate the deposition tunnels. Neither the coring nor mapping will be much influenced by choice of excavation method. However there is a difference in flexibility in case investigations are needed at the tunnel face. For those instances would Drill & Blast be favourable to TBM and horizontal pull-reaming as access to the tunnel face is simplistic.

A concern is to what extent blasting may damage the monitoring systems and also disturb the ambient conditions.

Adverse conditions

Different excavation methods have different ability to cover a range of rock and water conditions. The rock conditions in the deposition tunnels will be favourable but minor zones may be found that need rock support and this would be standard practice for any choice of excavation methods. Water ingress in minor zones or in single open fractures must nevertheless be assumed. Drill & Blast would permit easy access to the tunnel for grouting operations, but the blasting may later impair the grouted zone around the tunnel perimeter. The suggested TBM design will permit a number of holes around the perimeter and at the tunnel face but it is deemed more difficult to reach possibly stiff requirements on maximum permissible water ingress to simplify deposition work and backfilling of the deposition tunnel. Grouting for horizontal pull-reaming would be in between Drill & Blast

and TBM in complexity, assuming that grouting is from behind the reamer head with access from the service tunnel.

Procurement

For any of the excavation methods it is assumed that special equipment will have to be designed and manufactured. For Drill & Blast, the drilling rig should be purpose-designed and manufactured and similar arrangements are necessary for TBM or reaming.

Independent of if SKB uses contractors or own employed work crew, training is essential. While the excavation of deposition tunnels is a repetitive work it is foreseen that efficiency would increase very much after some years of fine-tuning technology and organisation and human skill.

Change of requirements and preferences

Requirements are “must” and preferences “want” and both “must” and “want” can change over time due to new knowledge and/or new requirements. As excavation of each deposition tunnel rather is a singular event rather than linked to excavation of previous or later deposition tunnel it would be comparatively straightforward to adapt or switch technology. If tunnels are excavated by Drill & Blast it is no dramatic change to start excavation by TBM or reaming as all the methods are mature and technologically feasible. Such a switch would of course provide a new tunnel shape requiring adaptation of road beds, vehicles etc.

Technology development

As noted, Drill & Blast, TBM and reaming are all mature technologies and no major breakthroughs are anticipated within a 10-year period, but steady improvements in reliability, precision and industrial IT will extend current frontiers.

In summary: Selection of excavation method is a decision that may be changed in the future. If deposition tunnels initially are excavated with Drill & Blast it would be possible to change to TBM and vice versa, granted that any long term safety and other effects are re-evaluated before the new decision is taken. The present study shows there are several mature technologies available that would be technically and economically sound. However, Drill & Blast seems to be the most flexible method as the area, shape and excavation damaged zone can be varied round by round if required. The method is also flexible with respect to varying rock conditions. There are many industrial references to Drill & Blast operations meaning there are many suppliers that may be used during construction. The presence of world-leaders in machines and explosives in Sweden help to design and manufacture a system that is purpose-fit for the requirements for the deep repository.

6.6 Overall judgement

The factors used for evaluation are schematically summarised in Table 6-6 (see Section 3.3). The advantages for Drill & Blast are costs and flexibility. The human factor is more important for Drill & Blast than for mechanical excavation. The advantages for horizontal reaming would subjectively be higher occupational safety, but costs are high. The use of TBM is also more costly but less than for horizontal reaming.

As an overall judgement it is thought that Drill & Blast can remain the preferred excavation method for the deposition tunnels. However Drill & Blast is to a large extent dependent on the human factor and therefore is construction quality at higher risks for Drill & Blast. It is here assumed that SKB by additional practical tests corroborate that needed construction quality can be achieved in the practical field situation. In case mechanical excavation is favoured, both TBM and horizontal reaming are feasible methods, but TBM would be favoured due to less excavation/backfilling work needed compared to use of horizontal reaming methods.

Table 6-6. Comparison chart for different excavation methods for the deposition tunnels. “+”: the alternative is more likely to achieve the objectives. “-”: the alternative is less likely to meet the objectives. “0”: the alternative is neither better nor worse for meeting the objective.

Objective/Issue	KBS-3V Deposition tunnel			Comments
	Drill & Blast	Mechanical Excavation Raise-bore TBM machine		
Low radiation dose after closure (ALARA) ⁷	0	0	0	EDZ: With proper Drill & Blast operation excavation damage is very limited and mostly confined to the end of the drill holes. The shape of the tunnel may be more important to avoid axial connectivity than the EDZ created by excavation. The site-specific best shape is easier to implement by Drill & Blast.
• EDZ				Backfill: Optimal shape is dependent on backfill method selected. The possible differences in quality of backfilling operation are insignificant compared to other factors that would influence the final backfill quality.
• Backfill issues				Construction and stray materials: General experience is that less rock support is needed for mechanically excavated tunnels. While the rock quality is assumed to be good for the deposition tunnels this difference is not important and we can expect that types and amounts of rock support is method independent. For all excavation methods there will be stray materials as steel, hard metal hydraulic oils. For Drill & Blast we can also assume explosives and detonators. However most of the material will accumulate in the muck in the floor of the tunnel. As the tunnel is cleaned before backfilling and the dirty muck is not used to produce the backfilling material the difference between excavation methods in this respect is of no consequence.
• Construction and stray materials				
No accidents for employees and contractors during construction and operation (incl. ALARA)	0	+	0	Different excavation methods present different risks, which once understood can be mitigated by design, regulations, and education. A subjective evaluation is that horizontal reaming would be more favourable to other methods as basically no people are present in the tunnels during the excavation work.
Small environmental impact during construction and operation	0	0	0	All methods will produce muck that is suitable for backfill. However the market of muck from mechanical excavation is less obvious. Spillages of e.g. oil and explosives into the muck would likely need treatment before deposition of waste rock. The oil spillage would be similar for all methods when haulage is included. Processing of drainage water would be simpler for mechanical excavation as the NO _x content is of no concern.
Sustainable management of natural resources	0	-	0	Raise-boring is high in energy consumption and in steel consumption that results in overall high generation of greenhouse gases and pollutants.

⁷ ALARA is a common principle phrased as “keeping the radiation doses to humans As Low As Reasonably Achievable, economic and social factors taken into account” /IAEA, 2002/.

Objective/Issue	KBS-3V Deposition tunnel			Comments
	Drill & Blast	Mechanical Excavation Raise-bore machine	Excavation TBM	
Low Net Present Value cost of construction and operation and closure	0	-	-	The real costs are very much dependent on the layout of the repository. One factor relating to cost is the length of deposition tunnels and need for extra tunnels due to choice of excavation methods. Cost differences in operating costs as a result of excavation method for the repository are difficult to estimate.
Short duration of construction from start of excavation to start of initial operation	0	0	0	With the assumption that several tunnel faces are available for Drill & Blast, excavation of the tunnels are not on the critical path.
High flexibility	+	0	0	Drill & Blast can easily adapt to change in tunnel shape, layout and rock conditions. The deposition tunnels are however quite standardised so need for flexibility is not very high except for the length of tunnels. As each deposition tunnel is a separate entity choice of excavation method is not an overall critical decision as the decision can change without dramatic overall consequences for the repository implementation.
Low project risks	0	0	0	Mechanical excavation is less dependent on the human factor and therefore is construction quality at higher risks for Drill & Blast. The evaluation here is based on a successful education and motivation programme. It is further assumed that SKB by additional practical tests corroborate that needed construction quality can be achieved in the practical field situation.

7 Evaluation of alternatives for other underground openings in comparison to objectives

7.1 Access ramp

Long term safety after closure

In a previous study /Bäckblom et al. 2003/ the impact on long term safety due to shaft or ramp access was explored with no preferences for neither shaft, “straight ramp” nor “spiral ramp”. However it was thought that the location of access compared to the location of the repository in relation to the groundwater flow might have a small impact on long term safety if all engineered barriers and the backfill have impaired isolating function. For this assumption it would be preferable to locate the access “upstreams” the repository in a costal aquifer regime.

With the notion that most retardation of nuclides would take place in the near-field to the engineered barriers, selection of excavation method for the access ramp is more of a standard optimisation task. The reference design and implementation plan is to Drill & Blast the access ramp in parallel with sinking of a shaft as the concurrent ramp and shaft excavation shortens the construction period with 18 months.

Occupational safety during construction, operation and closure of the facility

In case time good rock conditions for the access ramp exist, TBM excavation would be much faster (65 to 80 m advance per week for utilisation factor 40% to 55%) than using Drill & Blast for the access ramp (around 30 m advance per week). The faster advance rate for the TBM could either be used to cut the overall construction time or to change excavation method for the skip shaft. The use of TBM would provide time enough to construct the first shaft by raise-boring instead of shaft sinking and still maintain the reference schedule. The excavation of the shaft by raise-boring rather than by shaft sinking is much preferred for reasons of occupational safety as shaft sinking would be the single most precarious operation during all the repository implementation; fatalities are common. One fatality has occurred during construction of the Gorleben shaft in Germany, the Bure shaft in France and the WIPP-shaft in the USA, all shafts related to nuclear waste management facilities.

For construction, operation and closure of the ramp itself, occupational safety would not be very much dependent on selection of excavation method, see also Section 6.2.

Environmental impact and sustainable management of natural resources

Evaluation of environmental impact and sustainable management can use the reasoning in Section 6.3. Change from Drill & Blast to TBM however may have local effects on environmental effects. The transport of rock muck would take place more or less continuously while the TBM is working, whereas haulage of muck from Drill & Blast would take place in batches. For the latter method it is also assumed that mucking and haulage would take place during the night shifts. The construction duration is also shorter for the TBM than for the Drill & Blast operation.

However due to logistics, see Figure 7-1 the muck from excavation of the central area and shafts and raises (in total around 200,000 m³) solid rock would be hauled through the TBM-ramp rather than through the skip shaft that is the planning for the reference design. If this would have any local environmental impact is a site-specific issue.

Comparison of sustainability etc (Table 6-4) for Drill & Blast and for the TBM is not significant.

Schedules and costs

A simplified schedule for comparison of schedules for the reference design and a TBM access is shown in Figure 7-1. The excavation work for the access ramp includes excavation of the open cut, excavation of the ramp and road bed. For both alternatives it is assumed that grouting would take 7 months of the total excavation duration.

The schedule also assumes that the TBM can start excavation around 7 months after start of construction (7 months after receiving the permit). This necessitates procurement of the TBM (cost of around 80 MSEK) before the permit is received. Delivery time of proposed TBM will be approximately 10–13 months after contract signature depending on lead time and it will depend upon lead time of certain critical components like main bearing and gear reducers, as well as the selected supplier’s commitments at the time of order. In addition to the 10–13 months we need to anticipate shipping time to nearest harbour and the system assembly time of up to 2 months, total around 12 to 18 months from contract to start of TBM excavation.

Cost comparison is not straightforward as the consequences of selection are several and that cost data is truly uncertain, but an outline is provided in Table 7-1. However TBM is a strong alternative for the access ramp with the additional benefit of overall increased occupational safety due to absence of shaft sinking. In the calculations we assume that Drill & Blast excavation is by standard good practice (Specific drilling around 3.5 drill meter/m³ and specific charging of around 2.2 kg explosives/m³.)

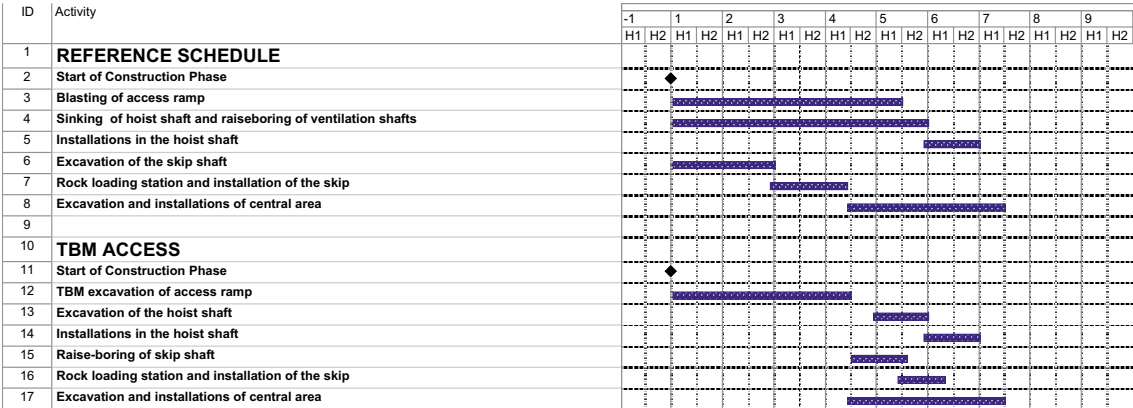


Figure 7-1. Comparison schedule for Drill & Blast of access ramp or TBM.

Table 7-1. Cost comparison for excavating the access ramp with Drill & Blast and TBM.

Issue	Access ramp		Comments
	Drill and Blast	TBM	
DIRECT EXCAVATION COSTS (see Section 6.4)	40 MSEK	45 MSEK	For the TBM it is assumed that the capital cost is proportional to its accrued usage. Assuming that market value is 50% of purchase price increases the direct excavation costs to 70 MSEK for the TBM
Increased costs for back-fill compared to reference design	–	70 MSEK	The backfill cost for the TBM would be lower for a Net Present Value calculation (Costs as NPV is in the range 15–30 MSEK)
Less costs for shaft excavation	70 MSEK	–	The skip shaft is excavated as a raise instead of a sink shaft for the TBM alternative
Shorter duration of construction	0	0	The TBM alternative may shorten the overall construction duration, but this is not accounted for here
SUM OF COMPARATIVE COSTS	110 MSEK	115 MSEK	

Flexibility, risks and opportunities

Drill & Blast can be used to excavate any layout, but the TBM is restricted to larger radii; we here assume that 200 m radius is achievable. However, even if a TBM is used, there is need for Drill & Blast to excavate niches where haulage trucks and vehicles can turn and meeting points be arranged as the TBM-ramp is too narrow for 2-lane traffic. It is expected that these niches are blasted at a distance of 200–300 m. However with a slightly larger tunnel (8.2 m diameter), no Drill & Blast niches would be needed.

A particular risk for the access ramp in addition to the other (see Section 6.5) is water ingress. As for any decline precautions must be taken regarding water inflow to prevent loss of life and inundation of machine and equipment. The system must be designed with drainage pumps and pipes to carry water out of tunnel to prevent water ponding at the heading, and standby generators must be provided for immediate engagement in case of electric power failure. To reduce water inflow in critical zones, the TBM would be equipped with equipment for pre-excavation probing and grouting in order to reduce or eliminate the water inflow problems. However, SKB will need to review the overall water inflow issue and maximum inflow possibilities in detail based on comprehensive site-specific geological and hydrological studies and the particular difficulties with grouting in connection with TBM excavation.

There are several industrial references for excavating a decline with TBM for example at SKB's Äspö Hard Rock Laboratory where 409 m of the access was constructed by a TBM. The TBM started at the depth of 420 m, drilled a decline in 14.5% for 200 m down to 450 m depth and then turned into an incline 1:100. Boreholes for probing and hydro tests were performed successfully from the TBM. Total 56% of the tunnel was grouted but in quite simple rock conditions where the maximum inflow in an open probe hole was 120 L/min. Mucking was with a LHD and haulage with standard trucks.

Another industrial reference is the Manapouri 9.6 km long 10 m diameter tail race tunnel in New Zealand constructed by a TBM in hard rock conditions with rock strengths up to 226 MPa. The tunnel passes through five sub-vertical fault zones and cover varies from 100 m to 1,200 m. The total water inflow into the tunnel during construction was up to 1,000 L/s with water pressures up to 7.2 MPa. About 20% of the tunnel is lined with concrete or shotcrete and 70% is left unlined. The around first 200 m of the tunnel was successfully drilled in a 12% decline using trucks for the muck.

The main opportunity for using a TBM for access is that the machine can smoothly continue to excavate main, transport and pilot tunnels. The diameter of the machine is in some sense flexible and can be adjusted ± 1 m if necessary for that case. However it is assumed that the TBM tunnel later is slashed into full profile with a flat floor in accordance with the present reference design.

Summary

Both Drill & Blast and TBM are feasible methods but the TBM would from an overall perspective be favoured if the site conditions allow for the preferred layout and that the special risks with water when excavating the decline are manageable. The main advantage would be that the skip and hoist shafts can be excavated by raise-boring rather than by shaft sinking (that is a much more risky operation from occupational safety point of view) and yet keep the master schedule.

7.2 Main – transport and pilot main tunnels

Main tunnels are the connecting tunnels between the deposition tunnels. Transport tunnels are the tunnels from the central area to the deposition area. Pilot tunnels are excavated to permit access to the site for detailed rock investigations prior to finalising the repository layout. These pilot tunnels are later converted to main tunnels by slashing the pilot tunnel to the shape of the main tunnel.

The general design requirements /SKB, 2002a/ specifically mention that the layouts of “other underground openings” like shafts, ramps, tunnels and central area should be arranged to minimise impact on the barrier function of the rock. As shown in Figure 6-4, the real layout can be quite different from a generic layout and it is then important that the connecting tunnels can adapt to the rock conditions at hand. In case the rock is more or less homogeneous it is likely that TBM is feasible. For a more patchy design, construction of curves (< 100 m) is essential and this would be achievable with a 5 m diameter machine but not likely with an 8 m diameter machine.

Reasoning of choice of excavation methods can follow the line of argumentation in Section 7.1 but it would be premature at this stage to make a firm recommendation as the site conditions are not yet known. The final selection is likely connected to selection of excavation method for the access ramp; it is more likely that the main – transport and pilot tunnels are excavated with TBM if the access ramp is successfully excavated with TBM.

Excavation of pilot tunnels with TBM would be especially attractive as the higher advance rate quite speedily would provide access to the whole site for detailed investigations (see Figure 2-10g). Also as this is a more or less continuous operation, other parts of the repository work would be disturbed neither by vibrations nor by gases. The TBM can be standardised and it is likely that no new machine need to be built but that second-hand machines can be used for this comparatively small construction work.

7.3 Central area

The only feasible choice for excavation of the central area is by Drill & Blast, both due to shapes and layouts. During excavation of the central area no spent fuel has been deposited so there is no need for coordinating construction and deposition work.

7.4 Deposition holes

The quality of the deposition holes is important as the rock in the deposition holes is the interface to the engineered barriers. Geometry is important both for deposition work but also for the long term function as the gap between the rock and the buffer should be narrow enough so that a design swelling pressure is reached when the buffer is saturated.

SKB and Posiva have jointly tested mechanical excavation of deposition holes with two different methods – the down-reamer at Olkiluoto and the shaft boring machine at Äspö and both methods fulfil requirements on geometry but not on efficiency as the direct excavation costs for the deposition holes are in the order of 3,500 SEK/m³. In this study an alternative design of the deposition holes has been explored where the upper grading of the deposition hole is made to allow for a smaller deposition tunnel. Such design would be difficult to arrange with a shaft boring machine.

For any of the methods available thorough development work would be needed to increase efficiency and a decision on what method to develop is recommended but it is likely that down-reaming is simpler to advance in technology than the shaft boring machine.

7.5 Deposition drifts for horizontal emplacement

The reasons for exploring horizontal emplacement of canisters is described in /SKB, 2001/. The report also contains a short overview with advantages and disadvantages of possible excavation methods for the around 300 m long 1.85 m drifts. The cluster drilling (Section 4.6.4) was deemed more favourable than TBM or horizontal push-reaming. The weaknesses of horizontal reaming identified were the mucking, grouting and the stabilizers for the drill pipes. A weakness of TBM is that it is not likely it can fulfil the geometrical tolerances on diameter with the tolerance of ± 5 mm over a 6 m length of the drift (SKB, unpublished report). However this conclusion is based on using disc cutters. Using button bit gage cutters, it is more likely that TBM would be technically feasible.

Based on the report /SKB, 2001/ and additional studies the cluster drilling was selected for development. Design, manufacturing and pilot tests have been carried through and it is most likely the technology would fulfil the geometrical requirements, especially if the pilot hole is preceded by a cored straight hole. However the technology has several drawbacks like water handling of around 65 m³ of water per m³ of rock excavated and high cost assumed to be more than two times higher than for horizontal push-reaming. The higher costs emanate from high costs for consumables and lower expected durable life for the equipment. In addition it would be much more complicated and costly to handle the flow of water/slurry for the cluster drilling. SKB has decided to make practical field tests at Äspö Hard Rock Laboratory with the horizontal push-reaming technology for future decisions on excavation methodology for a KBS-3H drift.

Horizontal pull-reaming necessitates an extra service tunnel and this would add some 300 MSEK to the construction costs.

Comparison of the Life Cycle Inventory for the different excavation methods for the deposition drifts was made. Some important input data is shown in Table 7-2 and from the table it is evident that the cluster technology would be comparatively lower rated with respect to environmental impact and sustainable management of natural resources, see the reasoning in Section 6.3.3. The water consumption for Cluster technology would also add additional cost for pumping the water to the surface in case it would not be possible to re-circulate the water. The energy for pumping the water to the surface would add an additional estimated 15 GWh to the energy for the Cluster technology.

Table 7-2. Overview of specific data for consumption of resources.

Issue	Excavation of deposition drifts		
	Cluster technology 120,000 m ³	Horizontal pull-reaming 120,000 m ³	Horizontal push-reaming 120,000 m ³
Energy for the specific excavation work and for mucking (GWh)	25	7	6
Steel usage, manufacturing, operation (tonnes)	750	450	450
Water consumption (m ³) per deposition tunnel of 285 m	49,000	15,000	15,000
Energy for pumping of water to the surface (GWh)	15	5	5

At the present stage it is assumed that both the cluster drilling and horizontal push-reaming are feasible, but that in the long run the latter would be a preferred technology. However, this assumption should be corroborated by the practical field tests in progress (autumn 2004).

8 Conclusions

The conclusions are here presented in view of the objectives stated for this particular study.

To investigate and compare principal technical solutions for rock excavation, both methods that are used at present but also methods that may be feasible 10 years from now

For all major mature technologies studied no major breakthrough are foreseen within the coming 10-year period. The basic technology will look very much as today but improved in several areas. For the deposition tunnels there is anyhow need for special design of equipment to ensure compliance with requirements and preference whether Drill & Blast or TBM is used. Methods for mechanical excavation of deposition hole exist but they are not efficient and technological development is needed to streamline the work. In case horizontal emplacement of canisters is favoured two methods would be feasible, but it is likely that horizontal push-reaming would show overall advantages.

The development stage for novel methods like plasma technology etc have been investigated, but no novel methods would be at a mature level for production within a 10-year period.

To assess how the selection of excavation method influences the design and operation of the deep repository

Selection of excavation method will for some instances change the generic layout of the repository. One example is horizontal pull-reaming that requires an extra service tunnels. TBM excavation of main and transport tunnels can not be excavated with small radii and therefore will not exactly adapt to the present generic reference design. In the real field situation where repository layout is patchier than the generic design, layout work will be hampered if restricted by the more inflexible TBM.

The selection of excavation method will influence tunnel shape that will present implications for design and operation of the repository. Drill & Blast can produce any tunnel shape, but TBM and reaming methods give a circular section that may not be optimal with respect to day-to-day operation of vehicles and for later backfilling work. From long term function, the shape should be stable and this optimal shape would be dependent on stress situation and rock properties.

Mechanical excavation is a more continuous operation compared to Drill & Blast and generates less vibrations and gases that would impact the repository in operation. For full mechanical excavation the overall logistics of the repository may be revisited as the need for a “construction” and a “deposition” side of the repository would be less obvious.

To present a definition of the Excavation Damaged/Disturbed Zone (EDZ) and practical methods for measurements of EDZ

The definitions suggested for the Excavation Damaged Zone is “the part of the rock mass closest to the underground opening that has suffered irreversible deformation where shearing of existing fractures as well as propagation and/or development of new fractures has occurred.” The definition of the Excavation Disturbed Zone is “the zone in which only reversible elastic deformation has occurred”.

Core mapping, geophysical measurements and direct visual observations of fractures and cracks are obvious methods, but measurement of acoustic emissions and microseisms are also methods that are feasible and useful.

While a main issue is the hydraulic conductivity of the Excavation Damaged Zone, interpretation of hydraulic tests in single boreholes are not straightforward due to partly unsaturated conditions and imprecise boundary conditions. The recourse is hydraulic tests in tunnel scale with backfilling in place and with saturated conditions. Such a test is in progress at the Äspö Hard Rock Laboratory (Plug & Backfill Test). Need for practical large-scale field tests based on best available technology for excavation cannot be precluded.

To present advantages and disadvantages with different excavation methods for the various tunnels and underground openings as a basis for selection of preferred excavation methods

Methods have been compared with respect to a set of factors – long term safety, occupational safety during construction, operation and closure of the facility, environmental impact and sustainable management of natural resources, schedules and costs and finally flexibility, risks and opportunities. While data are lacking for many instances, advantages and disadvantages are discussed in a qualitative rather than quantitative way.

The main advantage with Drill & Blast is flexibility and cost. The method can easily adapt to a range of rock conditions where tunnel shape and blasting design is adjusted to meet particular requirements. The technology is mature and efficient with resulting good overall economy both with respect to the excavation itself but also with respect to downstream costs of the repository.

The main advantage with mechanical excavation is that the operation is more or less continuous with a very constant and high excavation quality as the human factor cannot impact the quality to the extent possible with Drill & Blast. A disadvantage is that cost is higher, not necessarily due to excavation costs itself, but rather to downstream costs as the circular shape creates voids of no use but that anyhow need expensive backfilling.

Based on reasoning it is suggested that long term safety would not be impacted by choice of excavation method pending that Drill & Blast is executed to minimize the excavation damaged zone and would not create connected flow paths along the perimeter of the deposition tunnels. Also differences in backfill quality and remaining stray materials are not significant in between the methods.

There is no data to corroborate that Drill & Blast is safer/more hazardous than mechanical excavation. It is rather suggested that Drill & Blast and mechanical excavation present different risks, which once understood can be mitigated by design, regulations, and education.

The overall repository impact on environment and sustainable management of natural resources is to a smaller degree dependent on the selection of excavation methods. However from the study here, we conclude that raise-boring of deposition tunnels for KBS-3V (canisters deposited vertically) would be less favourable compared to the alternatives and that cluster technology would be less favourable for excavation of the deposition drifts in a KBS-3H design (canisters deposited horizontally).

To present the Design Justification Statement for the selection of particular excavation methods for the different tunnels and openings in the deep repository to underpin a decision on excavation method

Basic information has been compiled to compare excavation methods for the different tunnels and openings in a Best Available Technology (BAT) perspective. Any selection made on present premises and understanding would not prevent change of technology in the future. The excavation of the deposition tunnels is made over several decades and consequences of any future changes in technology would then be scrutinized.

The preliminary Design Justification Statements are as follows:

Access ramp

Drill & Blast is the reference design with TBM as a viable option. The TBM alternative however, can only be used for certain site conditions. The rock should be of good quality and the layout of the ramp feasible to construct with the TBM. In such circumstances the TBM is the preferred option, the main reason being reduction of cost and overall increase of occupational safety as shafts due to faster advance of the ramp can be constructed by raise-boring instead of by shaft sinking. The TBM-alternative assumes that procurement of the TBM is finalised around 12 months before receiving the permit for construction in order to start excavation in accordance with the current master schedule for the deep repository.

Shafts

With respect to schedules, it is assumed that the skip shaft is constructed as a shaft from the ground surface and downwards. Other shafts (hoist- and ventilation shafts) can be excavated by Drill & Blast and by raise-boring. The latter method is preferred in due consideration of costs and the decreased risks for accidents.

Central area

Drill & Blast is the only doable method in consideration of the large underground openings and the irregular shapes.

Pilot-transport and main tunnels

The reference design assumes conventional smooth blasting and this is still the recommended method. Construction of transport and main tunnels by TBM is less favourable as the layout flexibility is low. Pilot tunnels have smaller diameter than the ramp, which provides the opportunity to use a smaller TBM that more easily can construct curves with smaller radii. In case the ramp is constructed with a TBM, the possibility to use TBM for the pilot tunnels should be studied later.

Deposition tunnels for the KBS-3V alternative

All excavation methods studied (very smooth blasting, TBM and horizontal pull-reaming) would be technically feasible and possible to adapt to the requirements and preferences for the repository.

Drill & Blast can still be used in the reference design, but SKB will further study the integrated function tunnel/backfill, where the possibility of hydraulically connected flow paths along the tunnel floor is one of many parameters to consider.

In case mechanical excavation is needed, the TBM methodology would be selected before horizontal pull-reaming in consideration of the overall efficiency and economy. TBM is however a more complicated method for excavation than Drill & Blast.

Deposition holes

Drill & Blast is not a possible method due to requirements on final geometry like surface roughness etc. Two different types of mechanical excavation (down-reaming and shaft boring machine) are viable as both can fulfil the geometrical requirements, but neither of the methods are efficient and further studies are required before selection of method. It is assumed that down-reaming would be a more favourable method than using a shaft boring machine, but further studies are necessary.

Horizontal deposition drifts for the KBS-3H alternative

Cluster drilling technology and horizontal reaming are deemed to be viable methods. Horizontal push-reaming is preferred to pull-reaming as the latter requires an extra service tunnel. Also TBM using button bits gear cutters may be feasible.

SKB now has initiated practical field tests also with horizontal push-reaming to provide a firm basis for later decisions in case the alternative of horizontal emplacement is pursued.

To present background data that may be required for the evaluation of the long term safety of the deep repository

This study has compiled typical stray materials (types and amounts) for different types of excavation methods that may warrant detailed studies by the safety assessors. Data on excavation damages has been collected and evaluated. Possibilities for layout adaptation of the repository for different excavation methods have been explored.

9 Discussion

This study is conducted to possibly select excavation methods for the different tunnels and underground openings in the deep repository based on present understanding of relevant issues.

As a general comment there exist several technologies for excavating the different portions of the repository that all would be technically feasible and sound. Any selection made on present premises and understanding would not prevent change of technology in the future. The excavation of the deposition tunnels is made over several decades and consequences of any future changes in technology would then be scrutinized.

The present reference design for the deposition tunnels is based on Drill & Blast and the conclusion from this study is that the Drill & Blast option still should be the first priority for the deposition tunnels. The reasoning of this report is that excavation damage from very smooth blasting not is likely to produce axial flow along the deposition tunnels, a hypothesis that need to be underpinned by additional practical excavation tests. To achieve very smooth blasting, technology, the crew and the overall organisation in combination would be important. Drilling rigs should be especially designed for the deposition tunnels. The work needs strict quality plans and also a strong programme for motivation so each and every blasting round would meet the requirements. The importance of axial flow paths from long term safety point of view is also analysed in the safety assessments in progress (summer 2004). Independent of selection of excavation method additional integrated tunnel/backfill tests with relevant groundwater pressure and saturated conditions are necessary. The excavation for those tests should be done with Best Available Technology (BAT) and where backfilling is executed with the intended method and backfill material. In the unexpected future conclusion that Drill & Blast would not do for some reason, the TBM for deposition tunnels would be the second priority. With the present design proposal, there is not much need for layout flexibility for the deposition tunnels except c/c distance and length.

The possible advantages using a TBM for pilot-, transport- and main tunnels cannot be judged until the strategy for repository development and specific site conditions have been established. However in case a TBM is used for the access ramp, the option using TBM for other tunnels as well should be revisited.

Special considerations are worthwhile for excluding shaft sinking or for really ensuring the best practice for occupational safety as the shaft sinking likely is the most dangerous single activity at the site during the repository implementation.

An observation in this study is that all excavation technologies are mature; no major breakthroughs are foreseen within a 10-year period. It is likely that for any technology selected, SKB would specifically fine-tune the design of the equipment in view of requirements and site specific conditions.

The issue of EDZ warrant further studies as current tests are inconclusive with respect to the axial hydraulic conductivity and selection of excavation method. Concerning hydraulic measurements in boreholes for the ZEDEx-experiment the report by /Emsley et al. 1997/ op cit p. 133 states that "It is not clear whether the measurements reflect primarily the drift excavation effect (EDZ) or rather a wellbore effect. The results suggest that there is no well defined and significant increase in permeability of the rock mass in the damaged zone in

the vicinity of drift excavation which could be observed systematically in the analysed data even when taking the uncertainties into account. The data show that there is no significant distinction between the damage extent in the Drill & Blast and TBM boreholes". In view of these statements and other tests it is reasonable that the existence of a conductive EDZ is an assumption that has to be considered separately for the evaluation of the long term safety.

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