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Storage of nuclear waste in very deep boreholes:

Feasibility study and assessment of economic potential

Part I Geological considerations
Christopher Juhlin

Part II Overfall facility plan and cost analysis
Håkan Sandstedt

Vattenfall

December 1989

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This report concerns a study which was conducted for SKB. The conclusions and viewpoints presented in the report are those of the author(s) and do not necessarily coincide with those of the client.

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STORAGE OF NUCLEAR WASTE IN VERY DEEP BOREHOLES:

PART I GEOLOGICAL CONSIDERATIONS FOR THE VERY DEEP BOREHOLE CONCEPT

Vattenfall, December 1989

PREFACE

As part of SKB's (the Swedish Nuclear Fuel and Waste Management Co) research and development programme, alternative concepts for the permanent storage of nuclear waste are being studied.

The current study began in 1987 and two interim reports have been written

Stage A: Preliminary Review

Stage B: Outline Design and Quality Assurance Review

This report represents Part I of the final report which builds upon work done during Stage A and Stage B in addition to further studies carried out during 1989. Part II, which complements Part I, deals with the engineering aspects of a very deep borehole repository. The two reports are

Part I: Geological Considerations For the Very Deep Borehole Concept

Part II: Overall Facility Plan and Cost Analysis.

This report has been prepared for SKB by Vattenfall, Dep. BEL as a joint venture between different experts in a number of geological fields.

Stockholm, December 1989

Christopher Juhlin

Storage of Nuclear Waste in Very Deep Boreholes: Part I
Geological Considerations For the Very Deep Borehole Concept

Contents

SUMMARY AND CONCLUSIONS

Introduction

This report constitutes Part I of a feasibility study for storage of radioactive high-level waste in deep boreholes. The basic idea is to deploy the waste at such a great depth that the time for migration of radionuclides to the biosphere becomes so long that either adequate decay has taken place or sufficient dilution of the waste has occurred to eliminate any safety hazard. In the interim report, Stage A, it was concluded that disposal of radioactive waste in deep boreholes may be feasible and economical using today's technology. It was also concluded that considerable engineering and development work will be needed in a number of fields.

Part I of this feasibility study concentrates on quality assurance-related questions such as geological prerequisities at great depth.

Geological model for a Swedish rock column down to a depth of 6 km

The geological model for a Swedish rock column at great depth presented in this report is based on results from the Gravberg-1 deep borehole within the Siljan Ring area. The most important geological results from the borehole related to deep borehole storage are summarized below:

- The bedrock is highly fractured down to a depth of about 1200 m. Below this depth fracture zones, which typically extend over 2-20 m, occur at a frequency of about every 200-300 m.
- Hydraulic measurement between 1250 and 3200 m indicate a hydraulic conductivity within the interval $k = 10^{-9}$ - 10^{-10} m/s. This conductivity probably corresponds to the most permeable zones in the rock mass.
- Highly saline fluids (salinities of 10-15%) are present below 6 km.
- Isotope data on calcite indicate groundwater may percolate to great depth.
- A temperature gradient of 1.61°C/100 m was measured after a furlough period of 10 months.
- Data from different sources including the Gravberg-1 borehole indicate a stress field where the vertical stress is lithostatic, the minimum horizontal stress is 2/3 of the vertical stress and the maximum horizontal stress is somewhat larger than the vertical stress.

Review of scientific results from other deep boreholes

A review of geoscientific data available from deep boreholes in crystalline rock confirms in principle the geological model based on the Gravberg-1 borehole. On the basis of this review, the following model of the behaviour of crystalline rock in the upper crust can be proposed. The upper 1000 m (this depth can probably vary from 500-2000 m) contains a zone of heavily fractured rock with average permeabilities several orders of magnitude greater than that of the rock below. This zone also has a separate or distinct fluid system with generally lower salinities than the fluid deeper down. Below about 1000 m the rock is much more competent and its seismic velocity is dependent mainly upon its composition. However, there will be fracture zones within the competent rock which have considerably lower velocities and may have significantly higher permeability than the surrounding rock. These fracture zones within the competent rock may contain different fluid systems that are not in hydrological contact with one another over the timespan corresponding to the age of the fluids.

The results from Gravberg-1 and other deep boreholes show very clearly the necessity of a deep investigatory borehole in Sweden. Even if a mined concept for the final storage is chosen, a location below the upper heavily fractured zone (approx. 1000 m) needs to be considered. Today very little knowledge about Swedish bedrock is available below 1000 m, and a well designed investigatory borehole to say 3000 m will answer many questions discussed in this report. Results from shallow coreholes in Sweden, where the most permeable zones are encountered at the bottom of the boreholes, exemplify this viewpoint.

Geological investigation programme

Also discussed in this report are different methods for geological investigation of a rock mass at great depth, both from the surface and in deep boreholes. The localization of a deep borehole storage site will primarily be based on geophysical surface investigations. Some deep drillholes will also be necessary.

In comparison to normal core drilling investigations, deep boreholes will be more dependent on geophysical borehole logging. For studies of, for example, fracture density and orientation at great depth in situ measurements are preferable compared to normal core mapping. Stress release, core losses and drilling-induced fractures will make any core mapping interpretation very uncertain. To summarize, the geological prerequisities need to be evaluated from several different sources such as drill cuttings, drilling parameters (ROP, Torque etc.), wireline logging data, spot coring etc.

Recent advances in wireline logging, as well as in borehole seismic techniques, will permit investigation of the rock mass well away from the borehole itself. These techniques together with crosshole seismics, should make it possible to identify any major fracture zones running parallel to the borehole itself, as well as the more easy to identify ones that intersect the borehole.

Compared to investigations in shallow coreholes it will be much more difficult and expensive to perform hydraulic measurements in deep full sized boreholes. Data with the same accuracy should not be foreseen. Various techniques for coring in deep boreholes are commercially available on the market today and extensive development work is being carried out within several scientific deep drilling projects.

For future work in the development of a concept for storage of nuclear waste in deep boreholes it is important to recognize as a source of knowledge that several scientific deep drilling projects are in progress throughout the world. In addition to several holes in the Soviet Union, a 10 km deep borehole is just now going to be drilled in Germany and other deep boreholes are planned in Great Britain, Canada and Japan.

Rock mechanic considerations

Good knowledge of the rock stress state is vital when drilling deep boreholes in crystalline rock. Stress redistribution, in the formation of breakouts, is by far the most important source for the generation of a disturbed zone around a borehole. In the Gravberg borehole, large breakouts are also the main reason for the difficulty in maintaining good deviation control at depth during drilling.

The rock mechanic discussions presented in this report show that breakout and fracture generation in hard, granitic rock is most likely to be initiated at the borehole wall and propagate into the rock mass. Any movement in the rock mass away from the borehole wall is most likely to take place along existing joints. This is very important because it will always be possible to seal off the disturbed zone from the borehole with the chosen sealing material, i.e. bentonite clay. With a proper choice of mud weight and drilling fluid composition it may be possible to reduce the formation of breakouts.

The fact that breakouts and fractures are created at the borehole wall and not parallel to the borehole some distance from it is very important. If "onion fractures" were created as proposed in the literature this could create a flowpath for migrating fluid and have a very negative effect on the concept of deep borehole storage.

1. INTRODUCTION

This report constitutes Part I of a feasibility study for storage of radioactive high-level waste in deep boreholes. The basic idea is to deploy the waste at such a great depth that the time for migration of radionuclides to the biosphere becomes so long that either adequate decay has taken place or sufficient dilution of the waste has occurred to eliminate any safety hazard.

In the interim report (SKB report), Stage A: Preliminary Review, it was concluded that disposal of radioactive waste in deep boreholes may be feasible and economical using today's technology. It was also concluded that considerable engineering and development work will be needed in a number of fields. A summary of Stage A is presented in this report in Appendix 1.

Part I of this feasibility study focuses on quality assurance related questions such as geological prerequisites at great depth. Part I focuses on the following issues:

- Geological, hydrogeological and geotechnical description of a probable Swedish rock mass from the surface down to 6000 m.

- Development of a geological, hydrogeological and geotechnical investigation programme.

The geological model for Swedish bedrock put forward in this report is mainly based on the results from the Gravberg-1 deep borehole within the Siljan Ring area. The description is based on reports not yet published, and some revisions may arise after a more detailed evaluation. For comparison, a review of geological results from other deep boreholes is also presented.

This report was commissioned by SKB, Swedish Nuclear Fuel Waste Management Co, and written by the people below:

Christopher Juhlin	Project Management Geophysics Review of deep boreholes in crystalline rock
Göran Rissler-Åkesson	Geophysical borehole logging
Håkan Sandstedt	Project management Geological investigation programme
Ove Stephansson	Rock mechanics
Bill Wallin	Isotope geochemistry

The work has been carried out in very close cooperation with Anders Bergström, SKB.

2. GEOLOGICAL MODEL FOR A SWEDISH ROCK COLUMN DOWN TO A DEPTH OF 6 KM

2.1 Introduction

The geological model for a Swedish rock column presented in this chapter is based on results from the Gravberg-1 deep gas borehole located within the Siljan Ring area. All data is released with the permission of Dala Djupgas KB, which is responsible for the deep gas prospecting venture. All data should be treated as preliminary. When the drilling operations are finished, Dala Djupgas KB will present all data of scientific interest in a Summary Scientific report and in a number of sector reports covering different parts of the project.

The Siljan Ring structure was formed by a meteorite impact that occurred in Devonian time about 360 million years ago. As a result of the impact the bedrock probably became fractured to great depth. The fracturing consists of both microfracturing within the rock matrix and large scale fracturing.

The Gravberg-1 borehole is located just outside the excavated crater within the so-called Granån-Styggforsen line, one of the major lineaments in the area. The Granån-Styggforsen line is dated to be preimpact. Even though the impact event has changed the geological prerequisites in the area, there is a feeling among the scientists involved in the deep gas project that the geological results from Gravberg will not differ that much from other areas in Sweden. It is at least unlikely that fracture density, hydraulic conductivity, etc. are considerably less than at other sites in Sweden where storage of nuclear waste is being discussed.

2.2 Lithology

Rock types present at the surface in the Gravberg area are also encountered at depth. The two Dala granites present at the surface may be reclassified as seven separate types of granite in the borehole. At 3900 m severe drilling problems arose. Later interpretation has shown that the drilling problems may have been caused by a chilled contact between two different granite intrusions. Today no information is available about the hydrogeological conditions of this zone.

Aside from the granite, several sub-horizontal dolerite intrusions occur in conjunction with the strong seismic reflectors. Their thickness varies between 3 to 60 m. The granites surrounding the dolerites appear highly fractured and could constitute a reservoir or a flow path for migrating fluids. The length scale of the dolerite sills is more than 15 km. In addition to the dolerite intrusions, several fine grained granite intrusives are present below 5400 m. It should also be noted that the granite is substantially altered in some zones even at great depth.

Contact zones between different rock types and altered zones are possible pathways for migrating fluids. When choosing a site for deep borehole storage it is preferable to locate it within a stable block with few intrusions. It is not yet known which zones in the Gravberg borehole constitute pathways for migrating fluid. The feeling is, however, that most fractures and altered zones have been healed considerably with secondary mineralization.

2.3 Fractures

The Gravberg-1 borehole has penetrated numerous fracture zones while being drilled. These fracture zones typically extend over 2-20 m of borehole and occur at a frequency of about every 200-300 m, except in the upper 1200 m of the well, where they occur at a frequency of about every 50-100 m. Many of the fracture zones in the granite at depths below 1200 m are associated with basic dolerite sills or with intrusions of acidic rock characterized by high thorium content. There are, however, several fracture zones which are not associated with intrusions of any kind and may be related to impact events or post and pre-impact tectonic stress regimes.

There has been some debate as to whether or not it is possible to identify fracture zones in crystalline rock wells drilled by rotary drilling, particularly if the borehole is severely broken out, as is the case in the Gravberg-1 well. There has also been considerable debate as to whether these fracture zones contain open permeable fractures or whether they are all healed. It is the opinion of the authors that the answer to the first question is that fracture zones can be positively detected by using a variety of fracture indicators. It is, however, a more difficult problem to determine whether these zones contain open fractures without directly hydraulically testing them. The technique used to locate fracture zones in the Gravberg well has been to incorporate information from the drilling parameters, geological on-site analyses and wireline logs. The following changes in the data are characteristic for fractures zones below 1500 m.

1. The hole drills in-gauge (no breakout)
2. Drilling rate increases
3. Sonic travel times increase
4. Deep laterlog resistivity decreases
5. The amount of fracture-associated minerals increases
6. Fractures or effects of fracturing are directly observed in the cuttings

Surprisingly, point 2 appears to be the most inconsistent when studying the data since very high drilling rates have been observed when drilling in what appears to be tight intact granite. This phenomenon is probably related to the fact that the intact granite is capable of supporting high differential horizontal stresses and when it is penetrated by the drill bit unstable conditions arise which allow the rock to be drilled more efficiently. Analyses from the cuttings can provide good correlation with the wireline log and drilling data to confirm the presence of fracture zones. However, the absence of corrobarative cutting analyses should not dismiss a zone as not being fractured since there are several possible reasons for fracture indicators not being present in the cuttings. In general, it is felt that the wireline log data is the most reliable for determining the locations of fracture zones in the well. In addition, there are a number of wireline logs which could be included in the above list to increase the confidence in locating fracture zones. These are:

1. The variable density log format of the sonic waveforms
2. Tubewave processing of the sonic waveforms
3. FMS images of individual fractures
4. Dipmeter log

As of now, only information from the dipmeter has been incorporated into the fracture indicators log, an example of which is shown in Figure 2.3-1. The fracture zone located between 4656-4666 m (based on the wireline log data) is a good example of the response expected from the various data over a fracture zone.

1. The hole drills in-gauge, caliper ratio (CALR) = 1.

2. Drilling rate doubles through the zone ROP = 6 m/hr.

3. Sonic traveltime (DT) increases from about 50 to 58 microsec/ft.

4. Resistivity (LLD) decreases from over 40,000 Ohmm to about 1500 Ohmm.

5. Altered feldspars (ALTF) in the cuttings show a sharp increase as well as the presence of epidote.

6. The presence in the cuttings of open fractures (OFR), closed fractures (CFR), healed fractures (HFR), crushed zones (CRZO) and cataclastic grains (CATA) increases.

The offset of wireline log data as compared to the drilling and cuttings data is due to stretching of the logging cable which has not been completely accounted for. Proper shifting of the wireline data up 5 m would give much better correlation.

It is of great interest to know to what degree these fracture zones contain open fractures and microcracks. As of now, no quantitative methods exist for determining the porosity in crystalline rock based on wireline log data. However, qualitatively it can be argued that the decreases observed in the sonic velocities and resisitivities across the fracture zones must be, to a certain degree, due to open interconnected fluid-filled fracture systems. Calculations of porosity in the fracture zones, both from the sonic and resistivity data, indicate that porosities in the 3-5% range are possible. The resisitivity calculation implicitly implies that this porosity is interconnected. It is also possible that the low sonic velocities and low resistivities can be accounted for by the presence of clay minerals in the fracture systems. Results from processing the tubewave reflective energy from the sonic data support the possibility of open fractures at great depth in the Gravberg-1 well.

Current research efforts in quantifying the physical properties of crystalline rock from wireline logging measurements will improve the techniques for determining the porosity and permeability of fracture zones. However, in cases where breakouts are present, many of the techniques fail and the sonic, resistivity and caliper logs may be the only data which are useful for studying the porosity and permeability of the near wellbore rock.

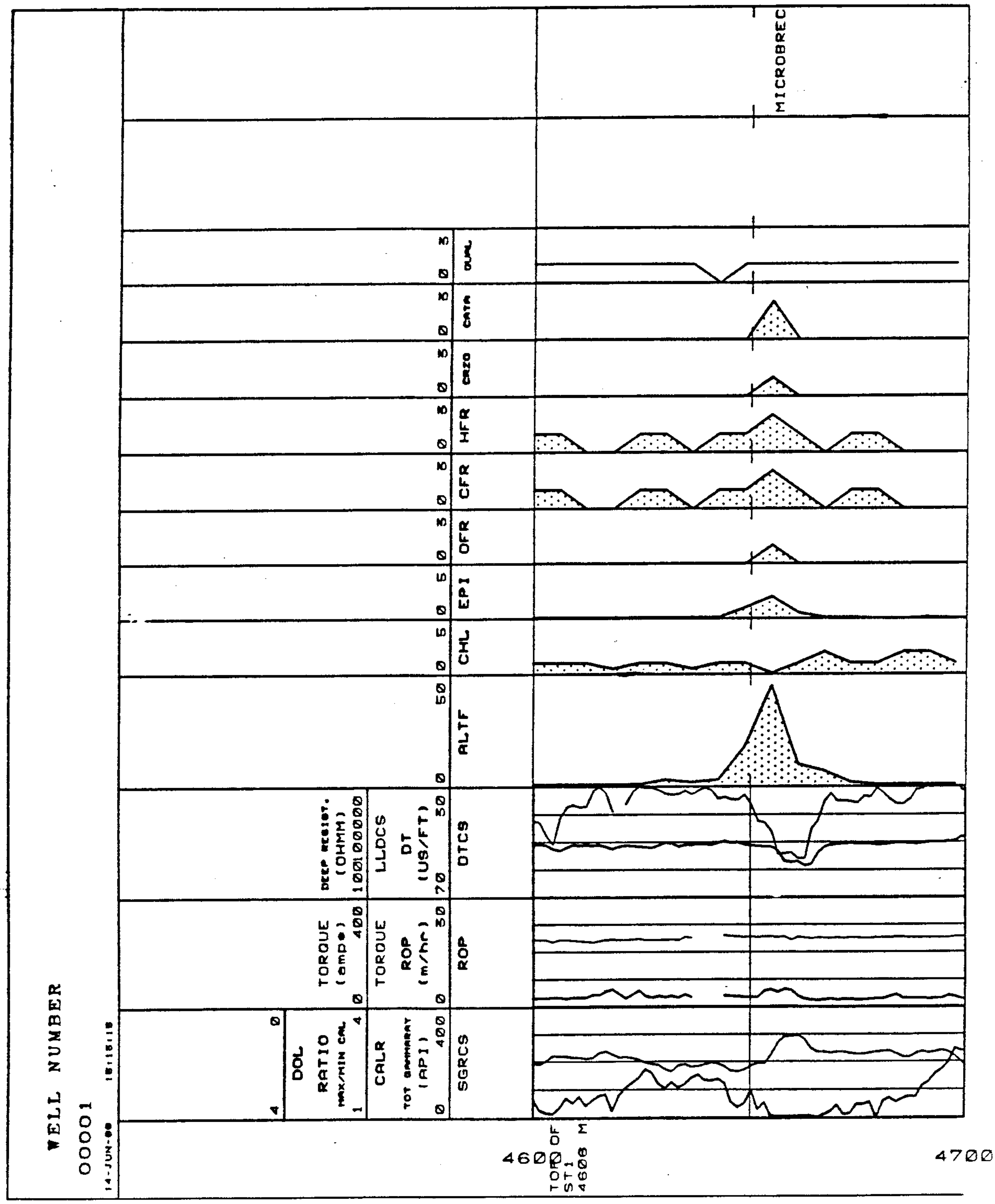


Figure 2.3-1 Fracture indicators log showing a fracture zone at 4656-4666 m

To summarize, the upper 1200 m of borehole in the Gravberg-1 well contains significantly more fracture zones than the borehole below 1200 m. These fracture zones in the upper part of the well are most probably open and able to conduct fluid. The zones that exist below 1200 m may contain open fractures and could also conduct fluids. However, they are not nearly as numerous as above 1200 m. Lack of reliable fluid conductivity tests over the most interesting fracture zones make it difficult to determine the quantity of open fractures at depth in these zones.

2.4 Hydraulic properties

Hydraulic measurements have been performed with fairly good accuracy in the interval between 1250 and 3200 m. Below this depth, data is influenced by different mud additives clogging permeable fissures during any pressure test. Below 3200 m only qualitative data is available, thus only relative values of permeability can be calculated.

Two types of tests have been performed: Leak Off Tests (LOT) and Drill Stem Tests (DST). LOT implies that drilling fluid is injected into the borehole at high pressure. During injection and a bleed-off period of 5-15 minutes, the pressure is recorded at one minute intervals. The bleed-off period is used for an evaluation of the permeability of the tested interval, which is always the whole open section of the borehole below the casing shoe. DST implies that formation fluid is extracted from the formation by pressure differences in the drillstem between packers or one packer and the bottom of the borehole. For depths above 2000 m, this implies that the drillstring is totally empty and the formation is treated with surface pressure at the beginning of the test. All pressure data is recorded with special equipment downhole.

Analyses of the hydraulic conductivities based on results from DST and LOT indicate hydraulic conductivities within the interval $k = 10^{-9}$ - 10^{-10} m/s. The results are presented in Figure 2.4-1. This permeability corresponds to the most permeable zones in the rock mass. The DSTs were carried out over zones which showed fractures and porosity. It should be noted that both tests are very crude and not at all compatible with falling head tests in shallow coreholes.

LOT below 4200 m indicated zones with higher permeability. This is in agreement with an increased density of fracture zones evaluated from other borehole data.

The LOT is performed with a high overpressure of about 10 MPa. This overpressure may open up some fractures resulting in an excessively high conductivity value. This hypothesis was tested at one point with two consecutive tests at different pressures. However, the results gave the same conductivity value.

The pore pressure in the rockmass down to 6 km is probably close to the hydrostatic pressure of fresh water. Due to higher temperature in the rock this pressure is slightly subhydrostatic compared to the hydrostatic column of fresh water in the borehole.

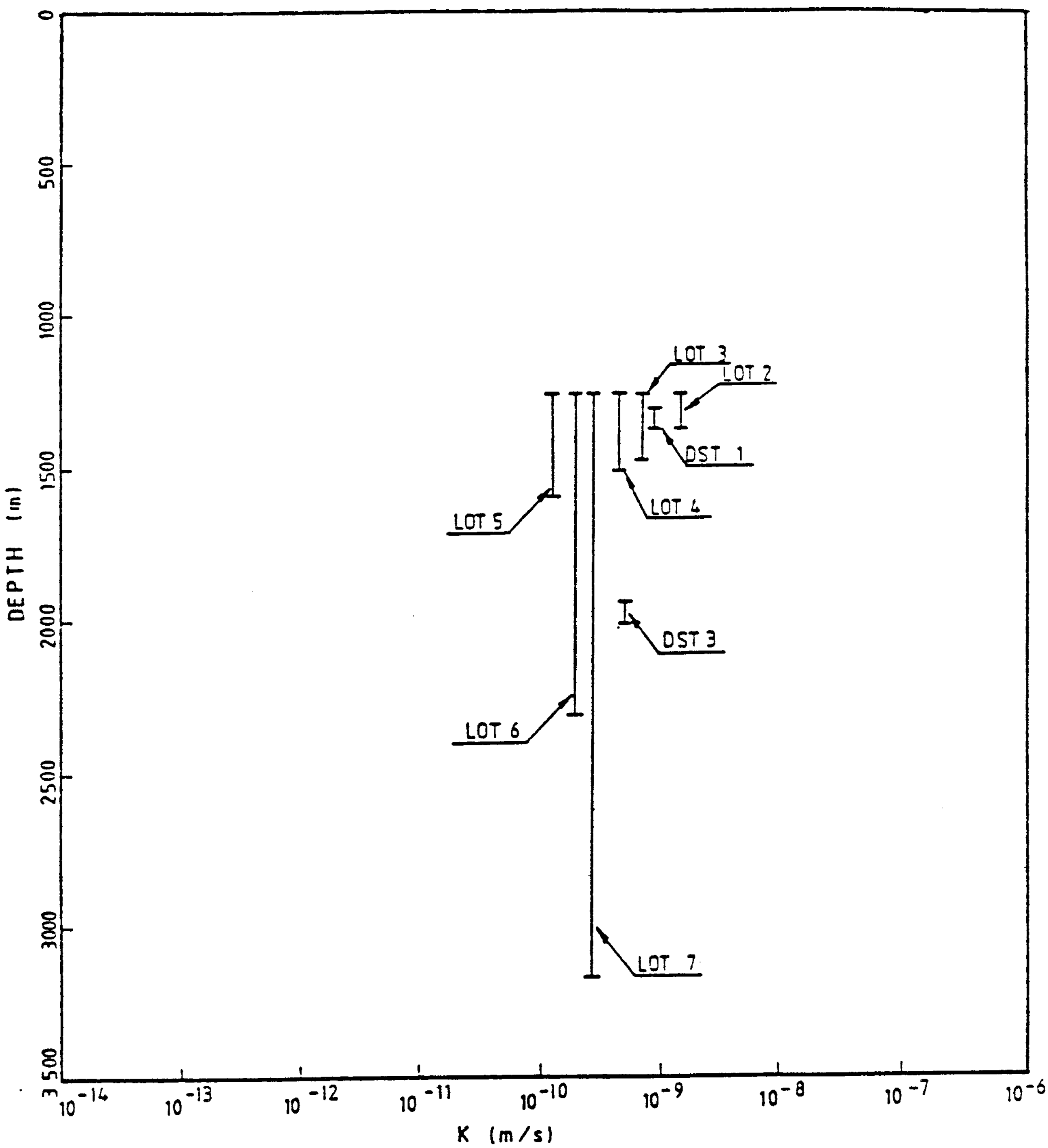


Figure 2.4-1 Hydraulic conductivities between depths of 1250 and 3200 m

Results from testing of the open hole interval 5452-6957 m after hydrofracturing give an average maximum conductivity of 6×10^{-12} m/s. This section of borehole was drilled with a high solids content oil-based drilling fluid which may have plugged a majority of the naturally open fractures. Even if plugging has occurred, the results show that very low conductivities surrounding the wellbore are present in this depth range. Analyses of the pressure vs. time response in the well while testing confirms that pore pressures are close to hydrostatic in this depth range.

2.5 Fluid systems

The drilling method used when drilling deep boreholes with oilfield equipment means that a large volume of drilling fluid has to be circulated in the borehole. The pressure from the fluid column is normally greater than the pore pressure in the rock formation, thus preventing any fluid from entering the borehole. Even when drilling with fresh water, the pressure in the borehole will be excessive due to an additional pump pressure and a subhydrostatic pore pressure in the formation.

Fluid sampling between packers is one method of obtaining fluid samples. During drillstem tests in the upper 3 km in the Gravberg-1 well attempts were made to sample fluids from the formation, but the samples were contaminated by drilling fluid. Good fluid samples can be retrieved from the formation, but an extensive flow period is needed. For a low permeability formation, the required time is very long.

2.5.1 Indirect information

Indirect measurements such as isotope data, ^{13}C and ^{18}O on calcite, and fluid inclusion studies give information on the fluid regimes present in a borehole. Such information is valuable, especially when no in-situ fluids can be sampled.

Studies of the light stable isotopes ^{13}C and ^{18}O have been made from the Gravberg-1 borehole (6 km depth) and six shallow (700 m) holes in the granite within the area of the Siljan Ring meteorite impact. The analyses are based on the calcite material found within the granite.

Physical features, including cracks and intrusions, have been recorded. A number of dolerite intrusions have been identified at different depths. The largest and the most important one for the groundwater percolation is situated at about 1500 m. This depth coincides with a drastic drop in calcite concentration, and this suggests that we might have a semi-active barrier for the groundwater, although judging from the isotope signatures it plays a minor role for the deeper circulation.

The overall isotopic data from the Gravberg-1 well and the shallower drilled holes in the vicinity suggest that the calcite mainly originates from a groundwater percolation system. The ^{18}O data reveal no dominant input of deep-seated high temperature fluids. In contrast, the data are typical for low temperature regimes. It would also seem extremely likely that there is one large source for the ^{18}O, although there is a wide range in the values. The explanation for this is the influence of the thermal gradient.

Since the fluid inclusion data within the granite indicate influence of thermal activity, we must be aware that the wide scatter in carbon most likely indicates that we are looking at different calcite generations in the granite.

The ^{13}C data display a very large range between -19.8 to +2.2 o/oo PDB (Pedee Formation Belemnite-standard) and the delta ^{18}O is +7.3 to 23 ^{o}/oo. SMOW (Standard Mean Ocean Water standard). Calcite concentrations from 0 to 1500 m vary between 0 and 2 wt %, and drop off dramatically below this depth to an average of about 0.2 wt %.

In conclusion:

The oxygen isotope data suggest a deep meteoric percolation system. There is most likely a mixing with input of deeper hydrothermal water at greater depth. However, the range in ^{18}O at a first approximation attributes this as an effect of the thermal gradient.

There is no striking evidence from the isotopic data that the dolerite intrusions in the granite have a major influence on the percolation of the meteoric water. However, the drop in calcite concentration at a depth of 1500 m, coinciding with the dolerite, indicates an influence. Usually major faults, cracks and lithological boundaries have a strong influence on the percolating fluids, although this seems not to be shown at Gravberg other than at the dolerite intrusion.

The ^{13}C-values indicate two sources of the carbon for the meteoric water: either organically derived carbon or input of high ^{13}C-values from the Palaeozoic carbonaceous sediments. This, together with the calculations of the thermal gradient influence of the ^{18}O, suggests that there is a deep circulated meteoric water in the granite with a slight mixing of a deep seated, possibly hydrothermally derived water.

Finally, it should be again stressed that we are looking at different generations of calcite in the granite cracks, each having its own precipitation history. We must also bear in mind that the values we observe are overprinted, mainly by the thermal gradient effect.

2.5.2 Fluids recovered from below 5452 m

During the testing operation in August of 1989 at least 200 m^3 of formation fluids were pumped from depths greater than about 5.5 km. Due to operational problems it was not possible to determine the exact depth of their origin. Since the well was hydrofractured downward below its total depth it is possible that these fluids originate from as deep as 7.5 km. Factors supporting a relatively deep origin are their extremely high helium concentrations compared to what was observed while drilling and the high salinity. The total dissolved gas concentration was estimated to about 0.25-1.0 m^3/m^3 of water which is below saturation at pressures of 60-80 MPa. Aside from helium, the primary gases consisted of nitrogen, hydrogen and argon. The highly saline water contains about 40,000 ppm Ca^+ and 12,000 ppm Na^+ cations with Cl^- being the major anion. Tritium analyses of the water showed no indications of any contamination with surface waters.

2.6 Temperature gradient

A temperature measurement was carried out in the Gravberg-1 borehole by the University of Århus in Denmark after a furlough period of about 10 months. This time frame should be sufficient for a steady state condition to be reached in the borehole. Two measurements were taken at an interval of two weeks. Both measurements give a gradient of 1.61°C/100 m.

Temperature measurements in several shallow coreholes down to a maximum depth of 600 m show a temperature gradient in the range of 1.32 to 1.45°C/100 m. The difference probably depends on the cooling effect from the ice cover during the last period of glaciation or on circulation of surface waters. The temperature data is currently being analyzed in order to determine the depths to which the effects of the glaciation are seen and for correlations between temperature anomalies and fracture zones.

2.7 State of stress at depth

The severe borehole breakouts observed in the Gravberg-1 deep well show the importance of rock stresses in the drilling performance and the stability of a well for nuclear waste storage. Since no stress measurements have been conducted in deep boreholes in Fennoscandia, an accurate estimate has to be made on the basis of other sources of information. For this study the following sources are used:

- Results from shallow rock stress measurements in the Siljan district.
- Data from the Fennoscandian Rock Stress Data Base.
- Focal plane solutions from deep earthquakes in Fennoscandia.
- Deep stress measurements in the western part of the European intraplate.

2.7.1 Stress measurements in the Siljan District

Rock stress measurements have been conducted in a 500 m deep borehole within the Siljan impact structure. The results show that the stress magnitudes are moderate to low compared to rock stresses measured at other sites in Fennoscandia and no extreme stress values are recorded, see Figure 2.7-1. Close to the surface, both the maximum and minimum horizontal stresses, S_H and S_h respectively, are larger than the assumed vertical stress, where the vertical stress is calculated from the weight of the overburden. The ratios of horizontal stresses to the vertical stress S_H/S_v and S_h/S_v decrease rapidly with depth, and at 500 m the stress state is approximately isotropic.

The orientation of the maximum horizontal stress at Stajsås as inferred from strikes of vertical hydrofractures in the borehole wall is approximately E-W from the surface down to a depth of 200 m, but rotates to about N63°W from 200 to 500 m.

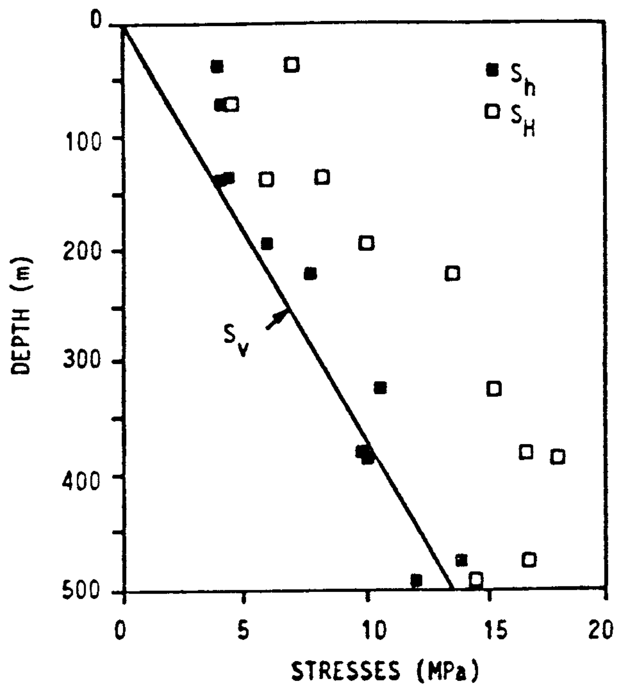


Figure 2.7-1 Rock stresses as a function of depth in the Stajsås borehole, Siljan district, after Bjarnason, 1987.

2.7.2 Data from the Fennoscandian Rock Stress Data Base

The data covering rock stresses in Fennoscandia (Finland, Norway and Sweden) have been compiled in a data base referred to as the Fennoscandian Rock Stress Data Base (FRSDB), Stephansson et al. (1987). A gross compilation and regression analysis of all existing stress data gives the following equations for horizontal stresses versus depth where z is in metres:

$$S_h = 0.028z + 12 \text{ MPa} \qquad (2.1)$$

$$S_H = 0.043z + 18 \text{ MPa} \qquad (2.2)$$

From this compilation it is evident that the vertical stress becomes the intermediate stress at a depth where the state of stress will favour strike-slip faulting. There is considerable scatter in the magnitudes and directions of the stresses for the uppermost 300-500 m. Below 400 m there is a clear tendency for the NW-SE direction to be the azimuth of the maximum horizontal stress in central Sweden.

2.7.3 Fault mechanisms of deep earthquakes in Fennoscandia

Focal plane solutions for deep earthquakes give valuable information about stress orientations in the crust. Slunga (1981) analyzed the fault mechanisms of 11 earthquakes in Fennoscandia. Nine of these were located in Central Sweden and in Southern Norway, not far from the Siljan district. Seven of these nine earthquakes showed strike-slip displacements on the fault plane. The mean azimuth for the largest horizontal compression estimated from the seven earthquakes is N55°W.

Thus, stress orientations from three independent sources coincide closely in the Siljan district. To estimate the stress orientations at great depth we prefer to use the information from deep earthquakes in the area, with the maximum horizontal stress, S_H, oriented N55°W.

Breakout orientations and borehole deviation in the Gravberg-1 deep well provide an additional independent source of information about horizontal stress directions at Siljan. This will be analysed in more detail in Chapter 5.

2.7.4 Results from worldwide deep stress measurements in continental crustal regions.

Rummel (1986) reviewed the stresses and tectonics of the upper continental crust using both the constraint imposed by experimental rock mechanics as well as existing in-situ stress data from deep hydraulic fracturing borehole profiles. He concludes that the maximum shear stress in the upper crust is determined by the two horizontal principal stresses, except from very shallow depths where the maximum shear stress is given by the stress difference S_H-S_V. Thus, strike-slip faulting would be the dominant fault mechanism in a randomly fractured crust. By using the stress ratios S_H/S_V and S_h/S_V as a function of depth for existing deep hydrofracture data in the continental crust, he obtained the following relationship between horizontal and vertical stresses:

$$S_H/S_V = (250/z) + 0.98 \text{ (MPa)} \qquad (2.3)$$

$$S_h/S_V = (150/z) + 0.65 \text{ (MPa)} \qquad (2.4)$$

where z is in metres.

In making a compilation of the state of stress in the Earth's crust it is of great importance to use data from the same type of tectonic regimes. A compilation of stresses from an extensional tectonic regime will show a completely different stress profile versus depth compared to one from a compressional tectonic regime. The western part of the Eurasian plate where the Siljan district is located belongs to a compressional tectonic regime. Figure 2.7.2 shows the stress ratios S_H/S_V and S_h/S_V versus depth according to equations 2.3 and 2.4.

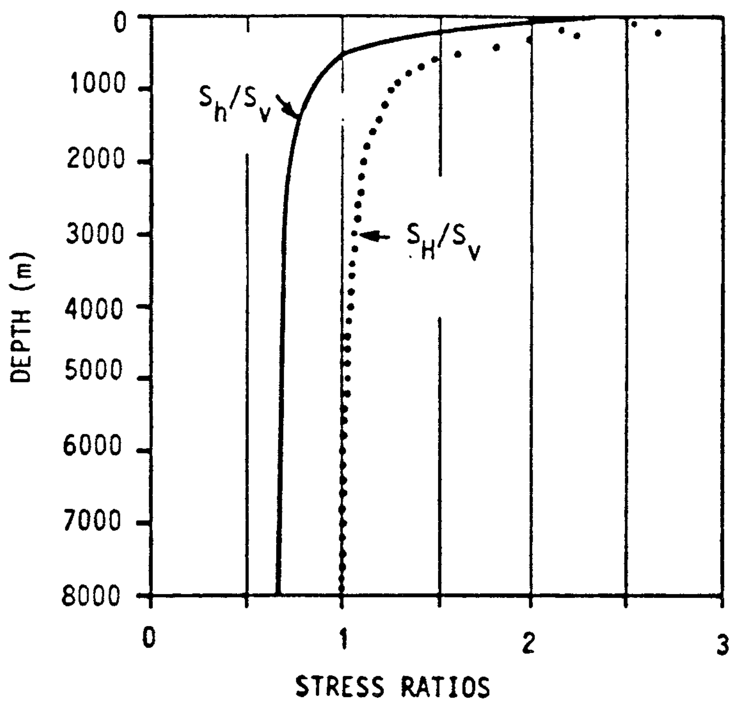


Figure 2.7-2 Stress ratios as a function of depth from deep hydrofracturing profiles in continental crust, and in accordance with eqs. (2.3, 2.4).

2.7.5 The estimated stress field in the Siljan district

The compilations and conclusions of Rummel (1986) and Stephansson et al. (1987) about stresses versus depth, correlate well with the situation at Siljan as inferred both from the shallow hydrofracturing results at Stajsås and from focal plane solutions of earthquakes in the area. In the absence of more reliable data, our estimate on stress magnitudes as a function of depth in the Siljan district is therefore based on equations (2.3) and (2.4). The vertical stress is assumed to be equal to the weight of the overburden at all depths as defined by:

$$S_v = 0.027\ z\ (\text{MPa}) \qquad (2.6)$$

where z is in metres.

Figure 2.7-3 shows the estimated stress profiles for the Siljan area according to equations (2.3), (2.4) and (2.6). At the surface, both horizontal stresses are larger than the vertical stress. Below an approximate depth of 500 m, S_h becomes smaller than the vertical stress, approaching a value of 0.65 times S_v for extreme depths. The maximum horizontal stress is larger than S_v at all depths but the difference is very small, see Figure 2.7-3. This stress state is most likely to be applicable to major parts of the Precambrian rocks of Fennoscandia.

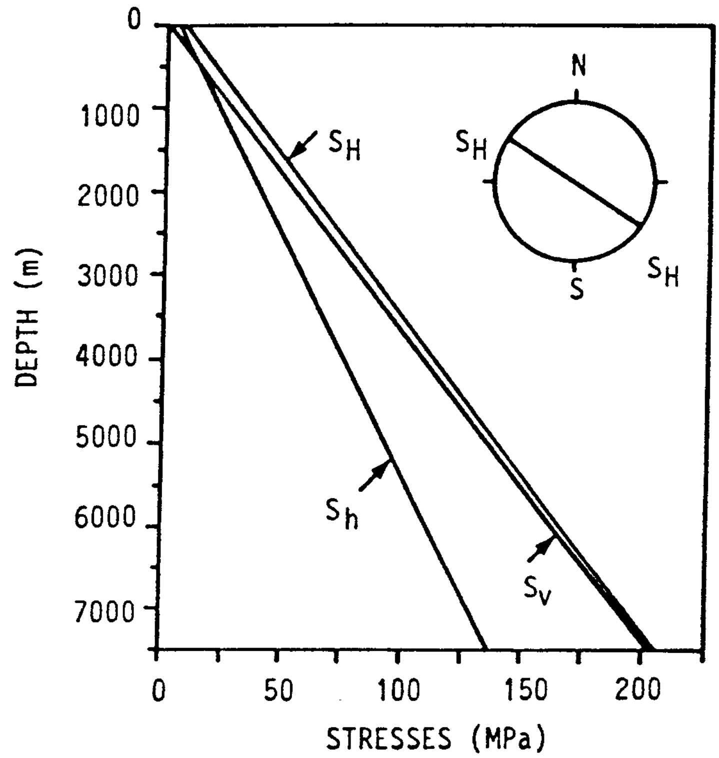


Figure 2.7-3 An estimate of stress profiles in the Siljan district. Orientation of S_H is shown in the upper right corner of the figure. This stress state is most likely to be applicable to large areas of the Precambrian rocks of central Sweden.

2.8 Rock strength

From the compilation and analysis of the stress data we find that at, or near, the Earth's surface, the horizontal stresses tend to exceed the vertical stress. However, at greater depth, the vertical stress becomes the more dominant stress component. The stress magnitude a rock mass can sustain depends on its rock type. Strong, brittle rocks like granites maintain stresses that are close to the failure strength of the intact rock and the discontinuities.

From the knowledge of strength versus depth for an average type of granite we can construct the ratio of S_h/S_v that the granite can sustain at failure. Here the maximum stress σ_1 is equivalent to S_v and the minimum stress σ_3 equivalent to S_h. The far left line in Figure 2.8-1 A represents the strength of intact granite tested to failure under conventional triaxial stress conditions and strain rates. At larger depths in the Earth's crust the strength increases. Hence, the S_h/S_v ratio, as shown in Figure 2.8-1, will increase and reach a value of about 0.15 at a depth of 7.5 km.

The frictional strength of pre-existing faults and fractures is governed by the coefficient of sliding friction, m, and the major and minor principal stresses, σ_1 and σ_3, are determined from the well known equation:

$$\sigma_1 = q\,\sigma_3 = (\,(\mu^2 + 1)^{1/2} + \mu)^2\,\sigma_3$$

where μ = tan ϕ and ϕ is the residual friction angle. From a large number of shear tests on rocks, Byerlee (1978) found the coefficient of friction to be $\mu = 0.85$. Instead, in eq. (2.5) we obtain q = 4.68, and the ratio σ_{o3}/σ_{o1}, corresponding to S_h/S_v is equal to 0.21. The ratio is shown in Figure 2.8-1 A and B as the second line from the left in the diagrams. Hence, any state of virgin stress and/or external loading of the granitic crust, causing a state of stress where $S_h/S_v < 0.2$, will cause failure.

When the limits of failure for intact rocks and friction of faults, joints and fractures are extended to greater depths, the two lines will cross each other at a depth of about 12 km. For a depth down to 12 km the failure of virgin rocks is most likely to be a slip failure and will appear along existing faults, joints and fractures. Below 12 km, failure is most likely to appear in intact rock.

Figure 2.8-1 also shows the change in S_h/S_v-ratio with depth according to eq. (2.4). The left part of the figure (A) is for dry rock and the right (B) is for wet rock. For the latter case the effective stress ratio, i.e. $(S_h-p)/(S_v-p)$, is plotted, where p is the pore pressure.

Irrespective of depth down to 12 km and whether we consider dry or wet granitic rocks, the curves for stress versus depth and strength versus depth never intersect. The gap between the limiting curves for stress and strength, Figure 2.8-1, tells us the excess loading the rock might take before failure. For dry and wet granitic rocks these ratios are S_h/S_v = 0.67-0.21 = 0.46 and $(S_h-p)/(S_v-p)$ = 0.48-0.21 = 0.27.

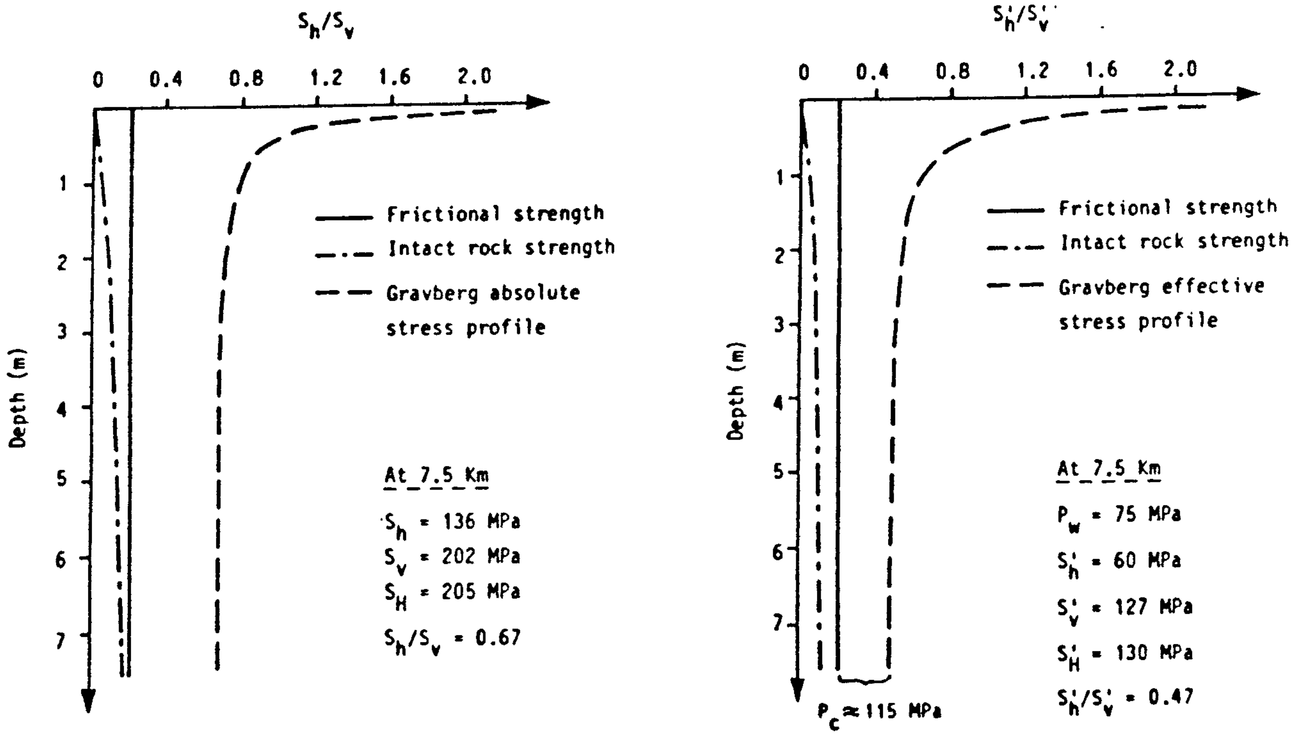


Figure 2.8-1 Ratio of minimum horizontal to vertical stress in crystalline rock as a function of depth. Strength of intact granitic rock and frictional strength for crystalline rocks are also shown. A, dry rocks and total stresses. B, wet rocks and effective stresses.

3. REVIEW OF GEOSCIENTIFIC RESULTS FROM OTHER DEEP BOREHOLES IN CRYSTALLINE ROCK

3.1 Introduction

The purpose of this chapter is to give a brief overview of the geoscientific information available from deep boreholes in crystalline rock. The list of boreholes in Table 3.1 included in the survey is by no means meant to be exhaustive, and the list can be updated as more information is obtained and previously unsurveyed wells are added to the list. This means that the matrix in the section on data availability will gradually grow in length and possibly in breadth. The term deep boreholes in crystalline rock refers to holes that have been drilled on land to a depth of at least 1500 m in primarily crystalline rock. The limit of at least 1500 m was chosen somewhat arbitrarily. However, it will be shown in the following sections that the intensive fracturing present in the near surface of crystalline bedrock is reduced significantly by this depth in all cases. This is consistent with the statement made by Professor Markus Båth which is referred to in the SKB report "Kärnkraftavfallets behandling och slutförvaring. Alternativa slutförvaringsmetoder, September 1986".

The data which are felt to be most important for quantifying the hydrogeological regime as a function of depth are the following:

1) Permeability
2) State of stress
3) Pore pressure
4) Fracture intensity
5) Fluid composition
6) Temperature

The ideal borehole is one in which all these data are available on as fine a sampling rate as required. Obviously, the ideal borehole does not exist, but some scientific boreholes do contain a large proportion of the above parameters as function of depth. It is also important to be able to tie results from boreholes back to surface investigations. Table 1 in Wynn and Rosenboom (1987) gives a summary of what geophysical surface and borehole techniques may help to define the hydrogeological regime. A number of other techniques can be added to this list, including drill stem and leak off tests, vertical seismic profiling and borehole imaging tools. It may be useful to keep these techniques in mind when reading the review in order to obtain an impression of what can be resolved from the surface and away from the borehole.

3.2 Known deep boreholes in crystalline rock

The list in Table 3.1 is based on our present knowledge. It can be updated continually as more information is gathered. However, it is felt that in its present form it includes most of the important boreholes where a reasonable amount of data is available concerning the properties of the rock as a function of depth. The boreholes are grouped by country and only those that were drilled on land, penetrated deeper than 1500 m and were drilled primarily in crystalline rock are included. The wells are coded with the following format which refers to the purpose of the drilling of the well (similar to the grouping used by Rowley and Schuh, 1988).

P - Petroleum exploration
G - Geothermal - Hydrothermal
H - Hot dry rock
S - Scientific

In those cases where several wells have been drilled in approximately the same location, such as in the hot dry rock projects in the USA and Great Britain, only one is listed.

Table 3.1 List of boreholes drilled in crystalline rock to a depth of 1500 m or greater.

Well no	Name	Spud Date	Measured Depth (m)	Crystalline Portion	Type of Well
USA-1	Mobil 1-A, Nevada	790909	5962	2440? - TD	P
USA-2	Nellie-1, Texas	831213	5822	1748 - TD	P
USA-3	Pinal County A-1, Arizona	800306	5490	1180 - TD	P
USA-4	Paul-Gibbs-1, Montana	830908	5418	1980? - TD	P
USA-5	Haraway 1-27, Oklahoma	810909	3810	520 - TD	P
USA-6	1-12 Boulder, Wyoming	840302	3506	? - TD	P
USA-7	TXO Henley F-1, Oklahoma	840509	3366	1000? - TD	P
USA-8	Fenton Hill, New Mexico	790304	4663	730 - TD	H
USA-9	Roosevelt Hot Springs, Utah	770330	3854	174 - TD	G
USA-10	Cajon Pass, California	870101	3472	500 - TD	S
USA-11	Wind River, Wyoming	?	3050	0 - TD	?
USA-12	South Hamilton, Mass	87	1829	O - TD	S
FRG-1	Urach-3, Swabia	790310	3334	1602 - TD	H
FRG-2	KTB, Bavaria	8709	4001	0 - TD	S
FRA-1	Sancerre-Couy	?	3500	940 - TD	S
SWT-1	Nagra, Böttstein	8310	1501	315 - TD	S
UK-1	Rosemanowes, CSM	811102	2800	0 - - TD	H
CAN-1	Measer MT, BC	820417	3500	0 - - TD	G
JAP-1	Higrori, Tohoku	790927	1804	1300 - TD	G
URS-1	SG-3, Kola	70	12060	0 - TD	S
URS-2	DB-3000, Ukraine	?	3500	0 - TD	S
URS-3	Ural SG-4	?	4008	?	S
URS-4	Krivoy Rog SG-8	?	3508	?	S
URS-5	Saatly	?	8300	?	S
URS-6	Central Asia	?	4000	?	S
URS-7	Caucaus	?	3700	?	S
SWE-1	Gravberg-1, Orsa	860701	6600	0 - TD	P/S
ITA-1	Sasso - 22, Lardello	?	4094	1450 - TD	G

3.3 Data availability

Deep boreholes are of limited value unless they have been investigated through sampling (cores, cuttings, drilling fluid) or downhole hole instrumentation (logs, fluid pumping tests, surface to borehole methods). In this section an attempt is made to give an overview of the data available from the boreholes listed in the previous section. The matrix in Table 3.2 is designed for this purpose. As stated in the introduction, its configuration is not static and it can be expanded as more information is obtained. In addition to the parameters mentioned in the introduction, borehole breakout and deviation have been included. For the drilling of deep holes these factors become extremely important, and it is of significance to know if breakouts occurred and to what degree, as well as any deviation, especially if attempts were made to drill vertical holes. The following codes have been used to describe the status of the data for each parameter for the particular borehole in question:

O - Data exists
⊗ - Data exists and is presented in this report
⊕ - Data exists but is not obtainable
ND - No data covering this parameter was collected
F - Future investigation
Blank - Status unknown

Table 3.2 Status of data on rock properties of the wells listed in Table 3.1.

	Permeability	State of Stress	Pore Pressure	Fracture Intensity	Fluid Composition	Temperature	Breakout/ Deviation
USA-1							
USA-2							
USA-3							
USA-4							
USA-5							
USA-6							
USA-7							
USA-8		⊗				O	O
USA-9				⊗		O	
USA-10	⊗	O	⊗	⊗	⊗	⊗	⊗
USA-11				⊗		⊗	
USA-12							
FRG-1	⊗	⊗	⊗	O	⊗	O	O
FRG-2	O	O	⊗	⊗	⊗	⊗	⊗
FRA-1	⊗		O	⊗	⊗	O	O
SWT-1	⊗		⊗	⊗	⊗	⊗	
UK-1	⊗	⊗	O	O	O	O	⊗
CAN-1							
JAP-1							
URS-1	⊗		⊗	⊗	⊗	⊗	O
URS-2							
URS-3							
URS-4							
URS-5							
URS-6							
URS-7							
SWE-1	⊗		⊗	⊗	⊗	⊗	⊗
ITA-1				O		O	⊗

It is necessary to make a subjective judgement as to whether data exists in sufficient quantity and high enough quality as to be of any value. Due to the limited amount of any data whatsoever on crystalline rock, the judgement has been generous because under the present circumstances any new data is good data. As one example of this philosophy, the matrix position corresponding to permeability for the Gravberg-1 well has been marked as "data exists" and is presented even though it is of a limited extent. In order to gain a full understanding of the nature of the permeability in the well much more testing is, in fact, needed.

It should be noted that several of the wells in Table 3.2 contain only blanks regarding the status of the data on the rock parameters. Enquiries have been made to the relevant institutions or persons concerning the status, but as yet no response has been obtained.

3.4 Results from various boreholes

A short review of the most important results pertaining to the hydrogeological regime of the rock near the borehole is presented on each borehole where data is available.

3.4.1 USA-8; Fenton Hill, New Mexico

The boreholes at Fenton Hill are part of the Los Alamos Hot Dry Rock Geothermal Energy Program and three of them extend to a depth of below 4 km. As of now, only limited data is available from these boreholes. However, some important results from stress measurements made in the boreholes can be noted. The minimum stress gradient is approximately 18 MPa/km (Brown, et al, 1985) which is somewhat greater than current estimates at Gravberg of about 17 MPa/km (Stephansson, et al, 1988). Most important, however, is the apparent stress discontinuity between 2.6 and 3.2 km which appears to be related to a change in lithology at these depths. Results from breakout studies in the well EE-3 (Barton, et al, 1988) show only minor breakouts indicating that the maximum horizontal stress is less than that in the Siljan area.

Important crosshole seismic investigations have been carried out between two of the wells where it was found that the seismic velocities decreased as the reservoir was tapped (Pearson, et al, 1983). This is explained by the opening of new cracks and fissures due to thermal stresses, which is consistent with the data concerning Vp/Vs ratios explained by O'Connell and Budiansky's (1974) theoretical models for fluid-filled cracked solids.

3.4.2 USA-9; Roosevelt Hot Springs, Utah

At present only standard wireline log data are available from Utah State Geothermal Well 15-21 (TD 2287 m, bottom hole temperature 204^{o}). The logs run include temperature, 1-arm caliper, sonic, neutron, density, resistivity, natural gamma and spontaneous potential. The rock below a depth of 174 m consists mainly of granites and gneisses of both Tertiary and Precambrian ages. The rocks contain significant quantities of mafic minerals and this is reflected in the average high velocities of about 6500 m/s below 817 m. At the top of the crystalline sequence at 174 m the average velocity is about 5300 m/s and gradually increases down to 6500 m/s at 817 m, remaining fairly constant below this depth.

Another notable result is the effect of the hole size and the concentration of mafic minerals on the neutron porosity, increases in which cause the neutron porosity to rise. The increase in neutron porosity over more mafic zones is attributed to the hydrous mafic minerals present in the these zones.

Wireline logs from the three other wells in the area have been requested from the University of Utah.

3.4.3 USA-10; Cajon Pass, California

The Cajon Pass scientific borehole has now been drilled to 3510 m (Andrews, 1989). The people involved in the project would like to deepen the hole to about 5 km, but lack funds at the present time. In general, the funding of scientific drilling in the USA is quite low at the present time (Elders, 1989) and it is not foreseeable that any scientific boreholes deeper 1 or 2 km will be drilled in the near future in that country. The shear stress and heat flow appear to be quite low along the San Andreas fault (Zoback, 1989). The low stress and heat flow values imply that the fault is quite weak mechanically and that current theories about how transform faults work need to be reevaluated. A fracture zone at 2400 m resulted in loss of the drillstring and subsequent sidetracking. The zone is interesting since it was observed on the VSP run at 1829 m and the drilling team was made aware on its existence prior to drilling it, however, it still created drilling difficulties. The borehole showed breakouts in the interval of 1750 m to 3510 m.

The primary goal of the Cajon Pass borehole was to determine the thermal and stress regimes in the vicinity of the San Andreas fault. Measurements in an adjacent exploratory well indicate low shear stresses in the area (Healy, 1987). Heat flow data show anomalously high heat flow in the vicinity of the borehole (Lachenbruch, et al, 1987) in apparent inconsistency with the stress measurements. The high heat flow may be due to recent erosion, and if adjustments for this are made it is possible to reconcile the two measurements. Temperature logs also show that the heat flow in the area can be completely accounted for by heat conduction and that circulating groundwater is not important. This is consistent with the chemical analyses of fluid recovered from seperate fracture zones where salinities can be high (2.15 g/l) in one zone and low (0.7 g/l) in another nearby fracture zone (Kharaka, et al, 1987).

Extensive vertical and offset seismic profiling in the borehole have enabled the rock to be characterized mechanically on a gross scale. The use of shear-wave vibrators as sources allowed shear-wave splitting to be observed in the borehole (Leary, et al, 1988), and from this an estimate of the direction of maximum horizontal stress could be made. This estimate was consistent with that made from the borehole breakouts that were first observed below 1830 m. Correlation of the sonic log with the VSP shows several reflecting horizons having their origin from fracture zones. The sonic log also shows a rapid increase in the crystalline rock velocity from about 4000 m/s at the sediment/basement interface (500 m depth) to about 5500 m/s at about 900 m. The velocity then rises gradually with increasing depth below 900 m and approaches 5800 m/s at 2000 m.

The 1830-1905 m interval was drill stem tested (Coyle and Zoback, 1988) over an interval containing at least two fractures zones. Permeability estimates over the zone are of the order of 10^{-18} m^2 and the pore pressure is estimated to be 5 % greater than the hydrostatic of fresh water.

During visits by members from Vattenfall to Cajon Pass, contact was established with the drilling side of the programme. At the symposium Deep Drilling in Crystalline Bedrock in Mora held in September 1987, good contacts were also made with the scientific side of the project.

3.4.4 USA-11; Wind River Mountains, East Pinedale, Wyoming

At present, it is not clear why this well was drilled. However, the Precambrian crystalline environment and the bottom hole temperature of 62°C at 3003 m (Smithson and Ebens, 1971) are very analogous to the Gravberg well. Cores of about 1.2 m in length were recovered from 19 intervals and are all of granitic composition ranging from quartz monzonites to quartz diorite gneiss. Wireline log data include sonic and density logs as well as a check-shot velocity survey. The check-shot survey shows a relatively low average velocity (5180 m/s) in the upper 460 m of crystalline rock. Below this depth the average velocity has increased to 5750 m/s. The authors have attributed the low velocities in the upper 460 m to intense near-surface fracturing and the low velocity zones observed deeper down in the well to fracture zones. In their opinion, "fractures are the most important factor in determining the velocities of crystalline rocks in place on a large scale".

3.4.5 USA-12; South Hamilton, Mass.

An interesting drilling project is the South Hamilton well which has been drilled in granitic and gneissic rock down to a depth of 1828 m north of Boston, Massachussets. The well has been drilled by the Reiss Foundation which is a non-profit organization whose purpose is to explore for and develop deep-seated water resources. The well is located in what is believed to be a major shear zone in the area. The interval from 914 m to 1828 m was drilled with a truck mounted rig from Tonto Drilling Services which allowed continuous fullsize core to be retrieved throughout the interval. A major fracture zone was encountered at 1219 m depth which turned out to have communication with surface waters since drilling mud turned up in one of the shallow water wells in the area. No permeabilities have been measured across the zone, but it is apparent that they must be extremely high for the transport to occur so quickly. No fluids from the fracture zone were obtained. One aspect of the project that is of special interest is the data on the gases which have been observed in the well. Methane has been found in the well in both the granitic sections and in higher concentrations in association with mafic intrusions. This is a pattern similar to that found in the Gravberg-1 well. The isotopic signature of the methane is also very similar that of the methane from Gravberg. In addition, there are several felsic intrusions which intersect the wellbore. Petrographic studies of these intrusions show that some of the samples have up to 15% porosity, even from the deeper levels. These intrusions could possibly form quite permeable pathways which allow fluid to be transported at depth. The Reiss Foundation would like to deepen the well to 3048 m, but lack funds at the present time. The cost for the deepening of the well with the same drilling rig as mentioned above and with continuous recovery of 70 mm core has been estimated to be $600,000. This is quite cheap and implies that a 3 km exploratory borehole can be drilled for about $1,000,000 with negigible mobilization costs since the rig is truck mounted.

3.4.6 USA-3; Pinal County A-1, Arizona

The well was originally drilled for petroleum prospecting purposes in the overthrust belt of Arizona. Seismic data showed a highly reflective crust beginning at about 4 km which was thought to represent sedimentary rocks which had been overthrust by Precambrian granites (Reif and Robinson, 1981). Upon drilling it was found that the reflectivity correlated to a change from a granitic lithology to a gneissic lithology (Goodwin and Thompson, 1988). At present it is not clear whether the source of the reflectivity is due to lithology contrasts within the gneiss or to fracture zones in that unit. One particularly strong reflection can be correlated with a zone of intense hydrothermal alteration at about 3.7 km, which represents the top of the reflective crust. However, this zone is within a young intrusive granite (47 Ma) in contrast to the older granites (1.39 Ga) above it or the even older greisses (> 1.5 Ga) below it. The upper 1183 m of the well consisted basically of granite wash and a study of a velocity increase within this lithology is not feasible. The Precambrian granite, which extends from 1183 m to about 3200 m, is considerably more fractured above 2.5 km than below this depth. Thus, the upper 1300 m of granite below the Tertiary granite wash may be of a similar nature to the upper heavily fractured interval at Gravberg.

3.4.7 FRG-1; Urach-3, Swabia

The crystalline rocks below 1602 m encountered in the geothermal well, Urach-3, consist of medium-coarse grained gneisses and diatectic rocks of plutonic character. All the rock types are occasionally severly altered in zones due to hydrothermal circulation (Dietrich, 1982a). These zones of strong hydrothermal alteration below 2000 m correlate with lower sonic velocities and lower resistivities. Since the crystalline sequence does not begin before a depth of 1602 m, it is not possible to monitor the important increase in velocity with depth in the upper 1500 m of the crystalline rock. However, below 1602 m the velocity remains nearly constant on the check-shot data at about 5600 m/s (Wohlenberg, 1982).

Results from the open hole fracture experiment at 3320-34 m (Schädel and Dietrich, 1982) indicate a minimum horizontal stress at this depth of 50 MPa, corresponding to a gradient of 15 MPa/km which is somewhat less than the estimate at Gravberg. Estimates of permeability over the same zone are on the order of about 3×10^{-18} m^2. It is interesting to note that communication was obtained between the interval of the propped frac No 4 (3296-3301) in the casing and the open hole below 3320 m. This communication was observed through use of uranin tracer and probably occurred along pre-existing natural fractures. That there exists vertical communication in the area is evidenced by results from chemical analyses of the fluid recovered. Only the fluid from the sedimentary rock above 700 m showed a fluid composition differing from the deeper NaCl mineral water. Fluid from below 700 m in the sedimentary rock down to the sediment-crystalline interface at 1602 m showed similar compositions. The fact that similar water types were encountered both in the sedimentary rock below 700 m and in the crystalline rock points to communication between the two rock types. Heat flow measurements also confirm the presence of fluid transport by convection with estimated velocities of 10^{-10} m/s. It is somewhat puzzling that in this regime measured pore pressures are about 10% below hydrostatic throughout the borehole.

3.4.8 FRG-2; KTB Bavaria

The KTB pilot well has been drilled down to 4000 m with approximately 3500 m of it cored with 98% core recovery (Rischmuller, 1989). It has been necessary to sidetrack the well on two occasions, once at 2000 m and once at 3677 m. The first sidetrack was the result of losing part of the drill-string when drilling through a fractured interval. The second sidetrack was due to oxygen pitted corrosion which attacked the drillstring from the inside of the pipe and resulted in loss of part of the drillstring downhole. Down to 3677 m the borehole had been drilled with a specially designed coring system and a flush drillstring. Below 3677 m the hole was drilled with standard rotary drilling equipment since the specially designed drill-string was unique and could not be replaced on short notice. Wellbore breakouts became a problem at about 2000 m depth in the well and so did deviation problems. It is deemed necessary to carry out extensive R & D in these two areas in order to be able to drill the deep well to the now targeted depth of 10 km. The deep well was originally targeted to 14 km, but an unexpectedly high temperature gradient makes this goal unattain-able with current drilling technology.

There have been a number of very interesting observations in the pilot well concerning fracturing. In the interval from 500 - 3500 m, virtually no open fractures were observed. However, below 3500 m open fractures became prevalent as evidenced by the influx of a highly saline fluid at 3900 m with an overpressure of about 14 bar (Emmerman and Behr, 1989). Gases such as methane, helium and hydrogen were also detected in the fracture zones below 3500 m with a 30 fold increase in the methane concentration in some intervals. The source of the methane is believed to be abiogenic, but whether it is of mantle origin is unknown. The uppermost zone of open fractures at 3500 m also correlates with a prominent seismic reflection observed on the VSP. As of now no virgin fluids have been recovered from these fracture zones and no permeability data have been obtained, how-ever, an extensive sampling program, started in November of 1989, should provide information on these parameters.

Other interesting results from the well are that there is a gradual increase in the seismic velocity with depth similar to the trend observed at Siljan. This is indicative of closing of microcracks due to higher pressures. The trend is obviously broken below 3500 m where open fractures become prevalent. Strain recovery tests on core recovered indicate the minimum horizontal stress to be approximately half that of the maximum horizontal and the maximum horizontal to be somewhat greater than or equal to the vertical stress. The observed temperature gradient in the borehole is about 27°/km which is greater than the maximum predicted gradient. The reason for the discrepancy has been attributed to surface circulation of ground-waters in the upper 500 m which distort the temperature profile. This upper 500 m interval is also apparent on the sonic log where it can be deduced that fracture density is much greater than in the rock below. The KTB group has also been able to post-orient all their core samples by matching fractures observed on the core with those seen on electrical images of the borehole in-situ.

An important point to note is that breakouts are nearly non-existent in the interval 2500 - 3500 m. Whether this is due to geological factors or drilling techniques is not clear at the moment.

3.4.9 FRA-1; Sancerre-Couy

The French have a program for investigating the continental crust by chosing specific targets where scientific problems can be solved through deep drilling. They have chosen to focus on comparatively shallow wells (depths less than 5 km) so that a variety of areas can be studied rather than spend all their resources on one super-deep well. At present, three wells have been drilled under the program and one of them penetrated crystalline rock, the Sancerre-Couy borehole, from 940 m to 3500 m.

The main purpose for drilling the Sancerre-Couy borehole (Megnien, 1989) was to investigate the source of the Paris Basin magnetic anomaly which has been studied by surface techniques for centuries. The borehole did not provide the answer to the source of the anomaly and it has been concluded that its source must lie deeper than the current 3.5 km depth of the borehole. The borehole did, however, give some interesting hydrological information about the rock at depth which consists primarily of 500 ma. gneisses. Below 3200 m several fracture zones where penetrated which were able to produce water to the surface. The water consisted of a saline solution (32 g/l) with about 1.3 m^3 of dissolved gases per m^3 of water. A breakdown of the gases shows that the main components are 70% nitrogen, 26% methane and 1.4% helium. The well produced approximately 0.5 m^3 of water per hour from these fracture zones for one month before the well had to be shut down due to budgetary constraints. The permeability in the interval from 3200 - 3500 m has been estimated to be about 10^{-15} m^2.

3.4.10 SWT-1; Nagra Project, Bottstein well

Nagra (National Cooperative for the Storage of Radioactive Waste) has been carrying out an extensive investigation of crystalline rock since 1980 for the expressed purpose of storage of nuclear waste. The borehole investigation programme has included continuous coring, sampling of formation fluids, hydraulic testing and geophysical logging. The crystalline sequence, beginning at 315 m in the Bottstein well, consists of mainly biotite granite with intrusions of thin pegmatitic and rhyolitic dykes.

Since the goal of the project was to investigate an area for its suitability as a nuclear waste repository, extensive hydrological testing has been performed. Results from the crystalline portion of the well are that the permeability is generally less than 10^{-18} m^2 except for a number of zones where it ranges from 10^{-18}-10^{-16} m^2. Below 1000 m the maximum permeability recorded during short duration tests was 40 x 10^{-18} m^2. However, in longer duration tests the highest value was 8 x 10^{-20} m^2 indicating that the fracture zones have limited lateral extent. Recovery of formation fluids points to three distinct water systems in the well. The first one is in the upper sedimentary rocks with a salinity of 6.4 g/l and an age of 13,000-20,000 years. The second extends from the sedimentary/basement interface down to 792 m and has a salinity of about 1.2 g/l and is younger in age than those waters above it. Below 800 m only one zone could be sampled at 1326 m due to the low permeabilities, and the fluid from this zone had a significantly higher salinity of 13 g/l and appears to belong to a different fluid system from those above. Pore pressures appear to be somewhat (3-4%) above hydrostatic throughout the well.

The decrease in permeability below 1000 m noted in the hydrological tests correlates with the sonic log. From 315 m to 1050 m the velocity increases from about 5000 m/s to about 5800 m/s. Below 1050 m the average velocity is about 5800 m/s which probably corresponds to the intact granite velocity. At the bottom of the well there is a marked low velocity zone, although this zone showed virtually no permeability when tested. The zone is not present on the check-shot survey either, which indicates that it may only have a limited lateral extent.

The geothermal gradient is about 34^{o}/km and the bottom hole temperature was 68^{o}.

3.4.11 UK-1; Rosemanowes Quarry, CSM

Three boreholes, RH11, RH12 and RH15, have been drilled in the Carnmenellis granite to depths of greater than 2000 m as part of the Camborne School of Mines Geothermal Energy Project. Most of the effort and resources associated with the drilling have been focused on determining the state of stress in the rock, stimulation, and characterization of the reservoir, especially below 2000 m. This has meant that only limited information has been made available concerning the properties of the rock in the upper 1500 m of the boreholes. For instance, the collection of sonic log data and vertical seismic profiling have almost exclusively focused on the rock below 1500 m and, therefore, no velocity profile versus depth is available. However, the data which have been obtained from the hydrofracture experiments is of great value, as are associated seismic studies below 1500 m.

The stress testing gives some of the most reliable estimates that are available on the state of stress down to 2500 m. In addition to the minimum horizontal stress, the maximum horizontal stress and its orientation have been determined (Batchelor and Pine, 1984 and 1986). Estimates of the minimum and maximum stress gradients are 12 MPa/km and 28 MPa/km, respectively. The minimum stress gradient is significantly less than that estimated in Gravberg of 16 MPa/km by Stephansson (1988). It has also been shown when hydrofracturing at Cornwall that the fracturing migrates downwards, and that it is primarily shearing along existing joints which is responsible for fracture. This was done by monitoring the fracturing experiments on the surface with a network of seismic stations. Analysis of shear wave splitting at these stations (Roberts and Crampin, 1986) gave stress directions consistent with those from other analyses. Crosshole seismic surveying before and after viscous stimulation shows decreases in frequency content, velocity and amplitude of seismic waves that cross the zone of stimulation (CSM, Report 2B-33, 1984-86). This indicates that the effects of stimulation, such as shearing and opening of joints, are permanent features. Hydraulic data (Pine and Ledingham, 1983) indicate that the pre-stimulation permeability at 2000 m is about $+10^{-17}$ m^2 at low differential pressures (0.7 MPa) and increases to $+6.10^{-17}$ m^2 at higher differential pressures (5 MPa).

There are numerous internal reports covering various aspects of the project. The following have been obtained by Vattenfall.

- Heat flow and temperature prediction in the vicinity of Carnmenellis pluton.
- Vertical seismic profiling 1984-1986.
- Crosshole seismic results 1984-1986.
- Viscous stimulation of well TH15.
- Microseismic results 1984-1986.
- Seismic velocity structure.
- Seismic reflection survey field report.
- Phase 2C technical progress review. October 1986 - September 1987.

3.4.12 URS-1; SG-3, Kola Peninsula

The 12 km deep Kola hole has the potential to provide a wealth of information concerning the physical properties of crystalline rock as a function of depth. Unfortunately, the literature available on the borehole in English, essentially the book edited by Kozlovsky (1987), is written in such a way that extracting real data from it is almost impossible. Perhaps the best overview of the drilling is found in the Swedish synthesis of the original Russian book conducted by Finkel (1985). This synthesis is at least as useful as the Kola book and much more understandable. In spite of the problems associated with obtaining accurate information from the Kola hole, much can be learned from the well.

The geology in the well is complex, but can roughly be devided up into three sequences:

0 - 4500 m	- Diabase, Gabbros, Volcanics and Meta-Sediments (Proterozoic)
4500 - 6835	- Metamorphosed mafics and Schists (Proterozonic)
6835 - TD	- Gneiss, Amphibolite and Ultrametamorphite (Archean)

The boundary at 4500 m is of great interest since it is at this depth a major influx of fluid into the well occurred corresponding to pore pressures a few per cent above hydrostatic (although higher pore pressures are also reported) and demonstrating unequivocally that it is possible to have permeable zones at depth in crystalline rock. This zone at a depth of 4.5 km is also associated with an observed low velocity zone on both the VSP and the sonic log. It is also at this depth where the temperature gradient has stabilized at about 23°C/km, having increased from 13°C/km at depths of 2-3 km (Moiseyenko, 1986). The water present in the rocks can be grouped into four zones

0 - 800 m	- low salinity (0.5 g/l) free water
800 - 4500 m	- higher salinity (20-25 g/l) chemically bound water
4500 - 9200 m	- high salinity free water
9200 - TD	- chemically bound water

The upper 800 m corresponds to what is called the exogenic fissured zone and contains the water that has recently been in contact with the atmosphere.

A sharp increase was noted in the upper 150 m in the seismic velocity in VSP surveys from 5200 m/s to 6400 m/s. However, no further increases were noted in the rest of the exogenic zone down to 800 m. In the remainder of the well, the velocities appear to be consistent with lithologies drilled. Early reported extremely low velocities, as well as high porosities and permeabilities, were based on studies carried out on cores and are thus uncertain due to the decompression which the cores have undergone and the impact from the drilling. The one in-situ measurement of permeability reported, giving a value of about 10^{-19} m^2 at 6370 m, is still relatively high considering the depth in the well. This measurement is from the interval 4.3-9 km were a continuous influx of water was observed while drilling. This influx probably comes from one zone with a fairly high water flow. No stress measurements have been reported in the literature and it is not known whether any were carried out. Borehole breakouts were observed throughout the well and were in some cases extreme.

3.4.13 URS-2; DB-3000, Ukraine

The DB-3000 well has been drilled to a depth of about 3500 m and penetrates primarily granitic rocks. The only literature obtained so far comprises results of velocity and density measurements on cores from the well (Levedev, et al, 1984). No information is available on what other investigations have been carried out in the borehole.

3.4.14 SWE-1, Gravberg-1

A summary of results from Gravberg-1 is included in Chapter 2 of this report. The main results were a rapid increase in velocity down to 1200 m (5000 m/s to 5800 m/s), permeabilities estimated to be of the order of 10^{-17} - 10^{-16} m^2, pore pressures of about hydrostatic, a temperature gradient of about -16.1°C/km, severe breakouts below 1500 m and severe deviation of the borehole below 4500 m. In addition, numerous fracture zones were encountered throughout the wellbore. In the final testing operation in August of 1989 formation waters from below 5.5 km were recovered. These waters had salinities of about 10-15% and a gas content of about 0.25-1.0 m^3/m^3 at the surface with the major gases being helium and nitrogen.

3.4.15 ITA-1, Sasso-22, Lardello, Italy

As of 1981 over 800 geothermal wells had been drilled in Italy (Baron and Ungemach, 1981), most of which were located in sedimentary environments. A number of them have, however, penetrated the crystalline basement and some of them penetrated deep into the basement rock in environments where the temperatures are in excess of 400°C. The Sasso-22 well was drilled to 4094 m penetrating Palaeozoic and Precambrian metamorphic rocks below 1450 m. The well is of interest since hard gneissic rocks below 3000 m caused severe deviation problems. The drillstring had to be fished out frequently and the hole had to be sidetracked three times. Only limited log data was obtained from the well, but it may be of interest to look at the sonic and resistivity logs over known permeable zones. As of now no data has been requested from this well.

3.5 National programs

3.5.1 Soviet Union

It is still difficult to obtain detailed results from boreholes drilled in the Soviet Union. However, in discussing results with some Soviet workers it is apparent that they have encountered fracture zones at depth which have both caused drilling problems and have contained open fluid filled fractures. Perhaps as the country opens its doors, more specific information concerning these fracture zones will become available. There are currently 6 deep wells being drilled in the Soviet Union in crystalline rock (table 3.3).

Well	Current depth
Kola peninsula	12066 m
Ural SG-4	4008 m
Krivoy Rog SG-8	3508 m
Saatly	8300 m
Central Asia	4000 m
Caucaus	3700 m

Table 3.3 List of deep wells in crystalline rock in the Soviet Union. The Ural well and the Krivoy Rog well have bottomhole deviations of 20° and 29°, respectively, and are currently being sidetracked in attempts to get more vertical wells.

The Soviets report that they have solved the problem of hole deviation in elliptical holes which would be a major breakthrough in deep drilling. It is not clear how this has been done and no details have been given. Apparently the technique works since they now report a deviation in the sidetrack in the Kola well of 2° where it previously was 15-20°. Although it is difficult to obtain details from Soviet workers attending conferences, they appear eager to work on a consultancy basis and it may be necessary to employ them as such in the future if detailed information is desired.

3.5.2 Canada

Canada is in the process of starting a national deep drilling program (Hall, 1989) similar to the French program where they focus their efforts on shallower targets (maximum depth is 5 km). The initial drilling project will consist of four 1-2 km deep holes in the Kapuskasing formation in Ontario. This formation is believed to represent lower crustal rocks and it is hoped the boreholes will provide clues as to the source of the lower crustal reflectivity which is commonly observed on seismic surveys.

3.5.3 United Kingdom

Three locations have been chosen in the United Kingdom as possible deep drilling sites for the British program (Whittaker and Pharaoh, 1989). All three of the holes would penetrate crystalline rock in the 3-6 km depth range. As of now, it is not clear when or if drilling will actually begin. In addition, it is possible that a 6 km deep hole will be drilled in Cornwall as part of the Hot Dry Rock project there. Again, it is not clear when drilling would start.

3.6 Comments on the review

While reviewing the literature on other experience gained in connection with drilling in crystalline rock a number similarities between the wells became apparent with, of course, some exceptions, the most striking being:

1. The variability in the composition of the crystalline rock in the boreholes.
2. The rapid increase in velocity in the crystalline rock with depth within the upper 1000 m and a much more stable average velocity below this depth.

3. The composition of the rock appears to be the controlling factor in determining the average velocity, while fracture zones are responsible for the low velocity zones encountered over shorter intervals.

4. The presence of distinct fluid systems where fluid has been recovered and an apparent separate fluid circulation system in the upper 700-1000 m.

5. The borehole breakouts in the Gravberg-1 well are extreme, but similar to observations in the Soviet Union.

6. The importance of in-situ measurements and the discrepancies that can exist between core data and these measurements.

The most important results from the wells reviewed are presented in Table 3.4.

	SC	HS	MHSG	K	TG	BP	PP
USA-8			18		80	minor	
USA-9	817				90		
USA-10	900	1800		0.1	30	1750-3510	+5
USA-11	460				17		
FRG-1			15	0.3	35		+
FRG-2	500	3500			27	0-2500	-10
FRA-1		3200		100			+
SWT-1	1050	1326		.001-10	34		+4
UK-1			12	0.1-6	34	no	
URS-1	800	1200		0.01	13/23	major	+4?
SWE-1	1200	>6000	17	1-10	16	1500-TD	0

SC - Depth to which surface water circulation appears to be dominant (metres).

HS - Depth to higher salinity water in the crystalline rock (metres).

MHSG - Minimum horizontal stress gradient (MPa/km).

K - Permeability (10^{-17} m^2) below 1000 m.

TG - Temperature gradient (^{o}C/km).

BP - Depth interval where breakouts are present (metres).

PP - Pore pressure (% above hydrostatic of water column).

Table 3.4 Overview of results from the review. In cases where circulation depths of surface waters have not been explicitly stated in the reviewed material, the depth is inferred from velocity information.

The permeabilities K have been measured over fairly extensive intervals (approx. 25 m) and the rock within these may be highly variable. Individual joints within these intervals can have permeabilities two orders of magnitude greater. On the other hand, as demonstrated at Nagra, if the tests are not of sufficiently long duration the permeability measured may be too high of an estimate. It must be borne in mind that the long tests were carried out with a lower hydrostatic head compared to the short duration tests. Tests over apparently intact sections of rock at Nagra resulted in permeabilities of as low as 10^{-20} m^2.

Based upon the review, the following model of the behaviour of crystalline rock in the upper crust can be proposed. The upper 1000 m (this depth can probably vary from 500-2000 m) contains a zone of heavily fractured rock with average permeabilities several orders of magnitude greater than that of the rock below. This zone also has a seperate or distinct fluid system with generally lower salinities than the fluid deeper down. Below about 1000 m the rock is much more competent and its velocity is dependent mainly upon its composition. However, there will exist fracture zones within the competent rock which have considerably lower velocities and may have considerably higher permeability than the surrounding rock. These fracture zones within the competent rock may contain different fluid systems that are not in hydrological contact with one another over the time span corresponding to the age of the fluids.

3.7 Comparison of other boreholes with Gravberg-1

Figures 3.7-1 shows the permeability data for the deep wells surveyed where these data were available. There is a trend towards lower permeabilities with depth (fig. 3.7-1) with a few exceptions. These exceptions are the one estimate at Cajon Pass of the permeability of a single fracture ($k = 1.8 \times 10^{-16}$ m^2) at 2070 m and the estimate in the Sancerre-Couy over the interval ($k = 1.0 \times 10^{-15}$ m^2) which produced formation water at the surface. These two intervals may be viewed as being representative of permeabilities for fracture zones while the other values may be viewed as being representative for the matrix. These estimates over fracture zones are 2-3 orders of mangitude greater than the matrix permeability which is in agreement with that was stated earlier. Another point to note in figure 3.7-1 is that the permeabilities measured at Siljan appear to typical ones for crystalline rock at those depths and may be regarded as representative while the extremely low permeabilities at Cajon Pass may be an exception. All the data should be taken with some caution, however, since the permeability measured may vary with the differential pressure applied (Pine and Ledingham, 1983).

Figure 3.7-2 shows the salinity data where available. Here, there is also a trend towards increasing salinity with depth, particularly in the Kola hole. The 75 g/liter salinity shown in this figure for the Kola in the interval 0.8-4.5 km differs from the 25 g/liter reported in section 3.4.12 since this is an estimate from the hole itself while the previous estimate was from adjacent holes. The lower salinities (2 g/liter) reported from the Nagra project and Cajon Pass wells are probably meteoric in origin whereas the others have a different source. However, analyses of the log data from the Gravberg-1 well, where porosity has been derived from the density log and the resistivity of the formation water calculated from Archie's formula (Juhlin, 1990 b), show no trends in decreasing resistivity with depth in the upper 4 km. This is indicative of meteoric water, which is known to have a high resistivity in the area, being present down to these depths. This is also

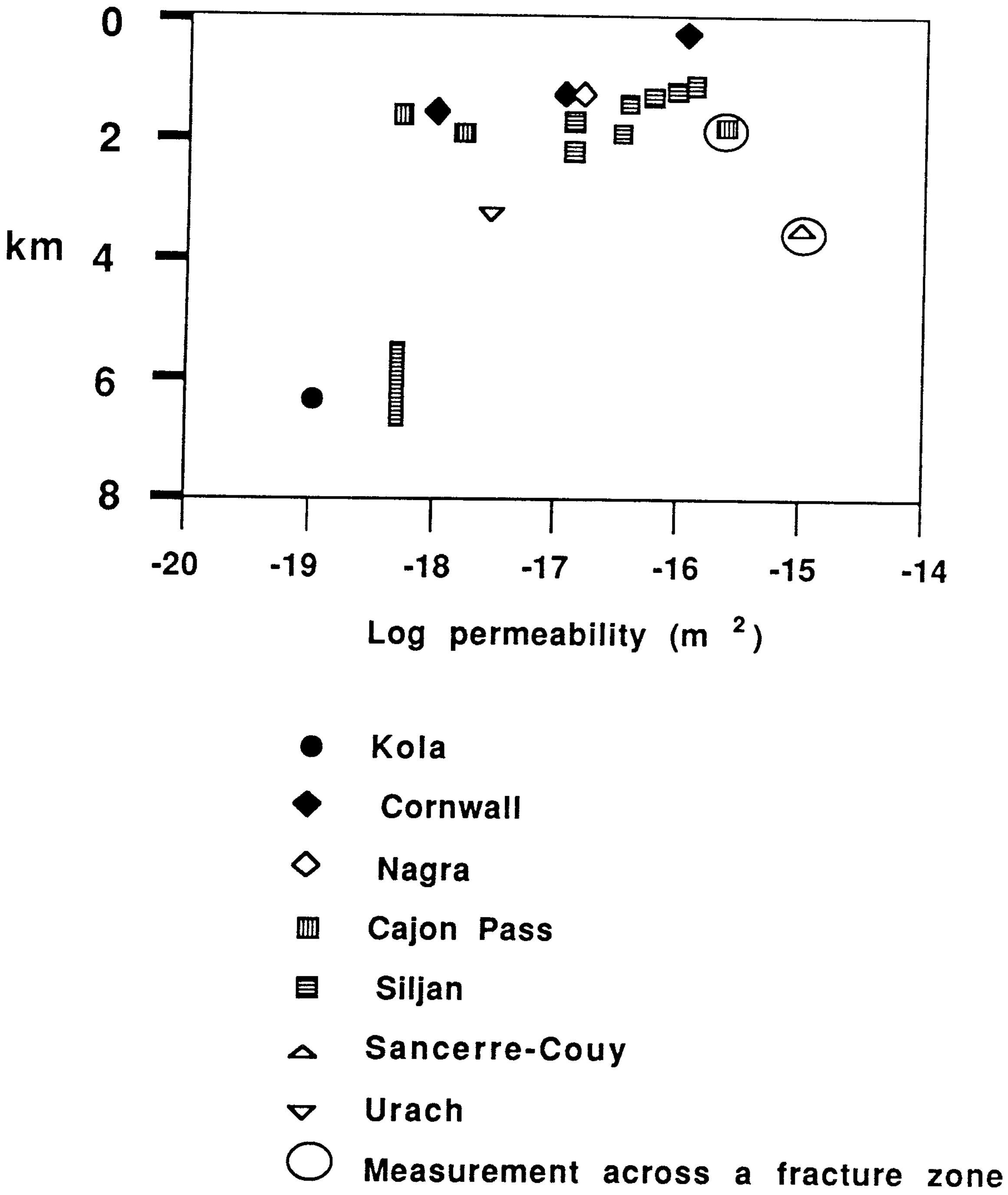


Figure 3.7-1 Permeability estimates in deep boreholes

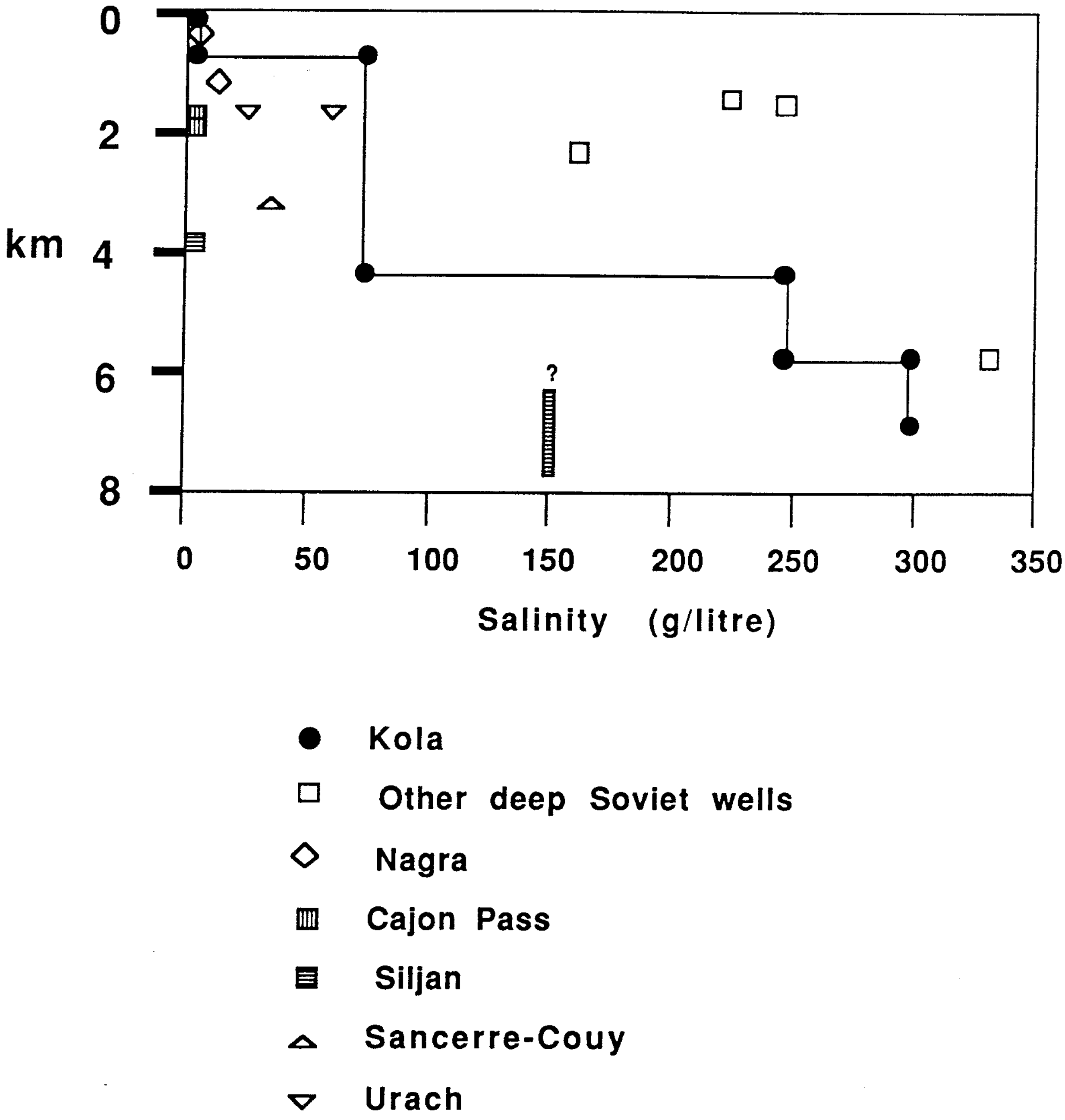


Figure 3.7-2 Salinities of formation waters from deep crystalline boreholes

in agreement with the geochemical interpretations of the calcite isotopic data reported on in Chapter 2. There appears to be a tendency for the salinity to increase in crystalline rock dramatically with depth in most areas once the depth to which meteoric waters do not percolate has been reached. This is illustrated in data reported by Vovk (1987) on waters from the East European Platform where the most saline water samples where from depths below 1500 m with salinities of 228, 246, 164 and 333 g/liter at depths of 1612, 1758, 2535 and 5099 m, respectively, in four different wells.

In summary, the expected trends of decreasing permeability with depth and increasing salinity appear to be verified by drilling, however, data is sparse. In addition, results from several boreholes (KTB, Sancerre-Couy and Soviet boreholes) prove that open fracture zones can exist at depths below 2000 m and that they can produce fluid. These zones must be taken into consideration when planning a very deep borehole repository.

An important question concerning the engineering aspects of a very deep borehole repository is the presence of breakout in the boreholes. At the present time the phenomenon is not completely understood. Breakouts became severe abruptly at about 1500 m in the Gravberg-1 borehole (Fig. 3.7-3) while in the KTB pilot hole they gradually decreased in magnitude with depth down to at least 3600 m (Fig. 3.7-4).

3.8 Comparative study of geophysical results from Cajon Pass and Gravberg

3.8.1 Background

As part of the review of deep boreholes program of this report a trip was sponsored by SKB for Christopher Juhlin to visit the University of Southern California and compare results between the Gravberg-1 borehole and the Cajon Pass borehole. It is hoped that the visit will result in that a scientific paper in an international journal can be published where the two boreholes are compared. Prior to any such publication, recently collected VSP data down to 3500 m in the Cajon Pass will need to be analyzed.

3.8.2 Preliminary comparison

The drilling of deep wells in crystalline rock provides an opportunity to study numerous geological processes which are poorly understood. Some examples are; what is the nature of the pore space at depth in crystalline environments, what are the fluids or gases occupying this pore space, what are the circulation patterns of these fluids and how does their presence affect geophysical measurements. Two wells drilled in contrasting crystalline environments provide some insight into the above processes, although many questions still remain. The Cajon Pass well (Geophys Res Lett. Special issue, 1988) in southern California has been drilled with the purpose of investigating the stress and heat flow regimes in the vicinity of the San Andreas fault. The well is located about 4 km from the fault and has penetrated mostly phanerozoic granitoids and gneisses down to 3.5 km in what is considered to be a highly tectionically active area. The Gravberg-1 well (Bodén and Eriksson, 1988) has been drilled in central Sweden in a 360 ma. old impact crater with the main purpose to explore for abiogenic methane. This well is located in the southern Baltic Shield and has penetrated primarily Proterozoic granite down to 6.8 km in what is normally considered to be a relatively inactive tectonic area.

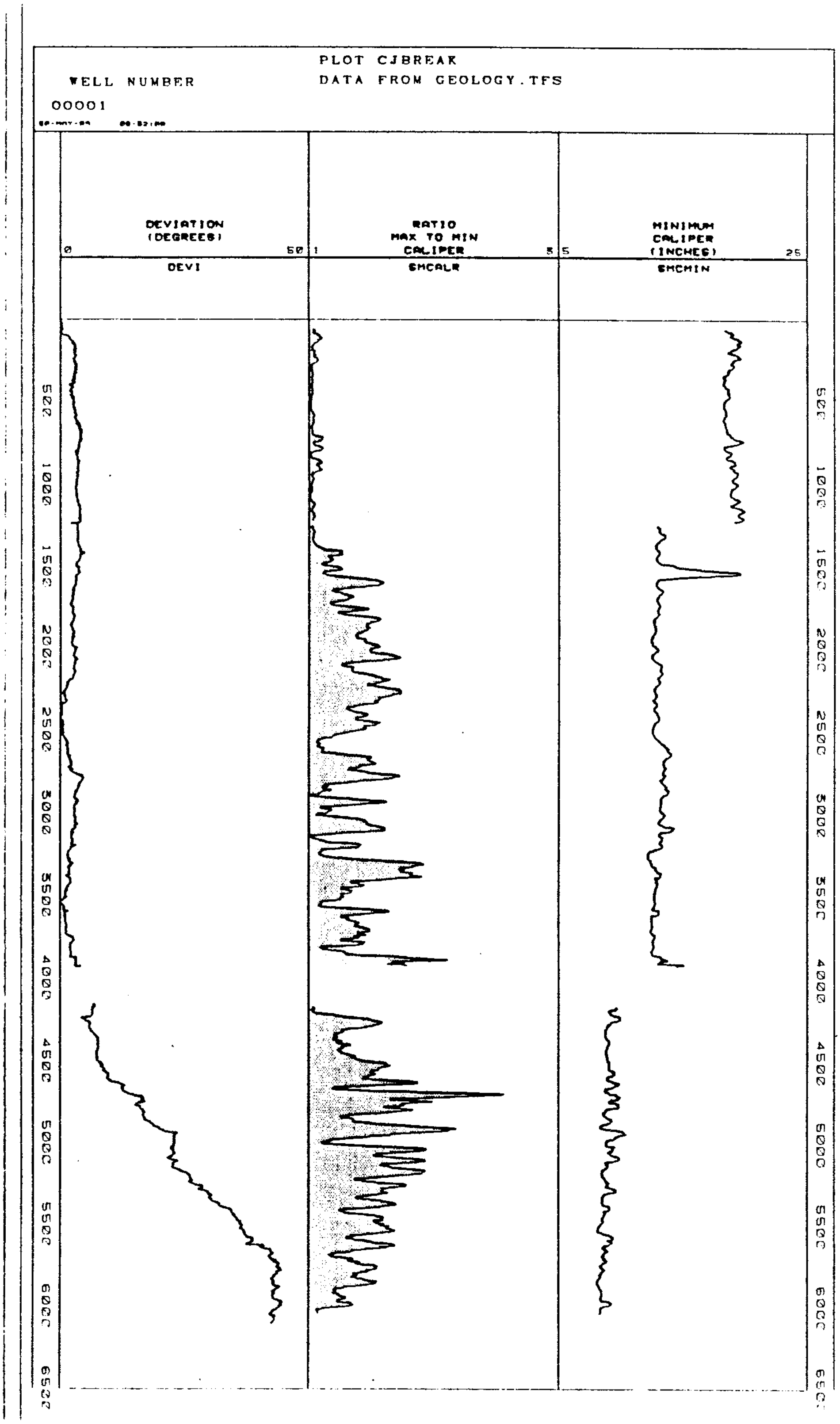


Figure 3.7-3 Breakout log at Gravberg. Shaded regions indicate zones where breakout is present.

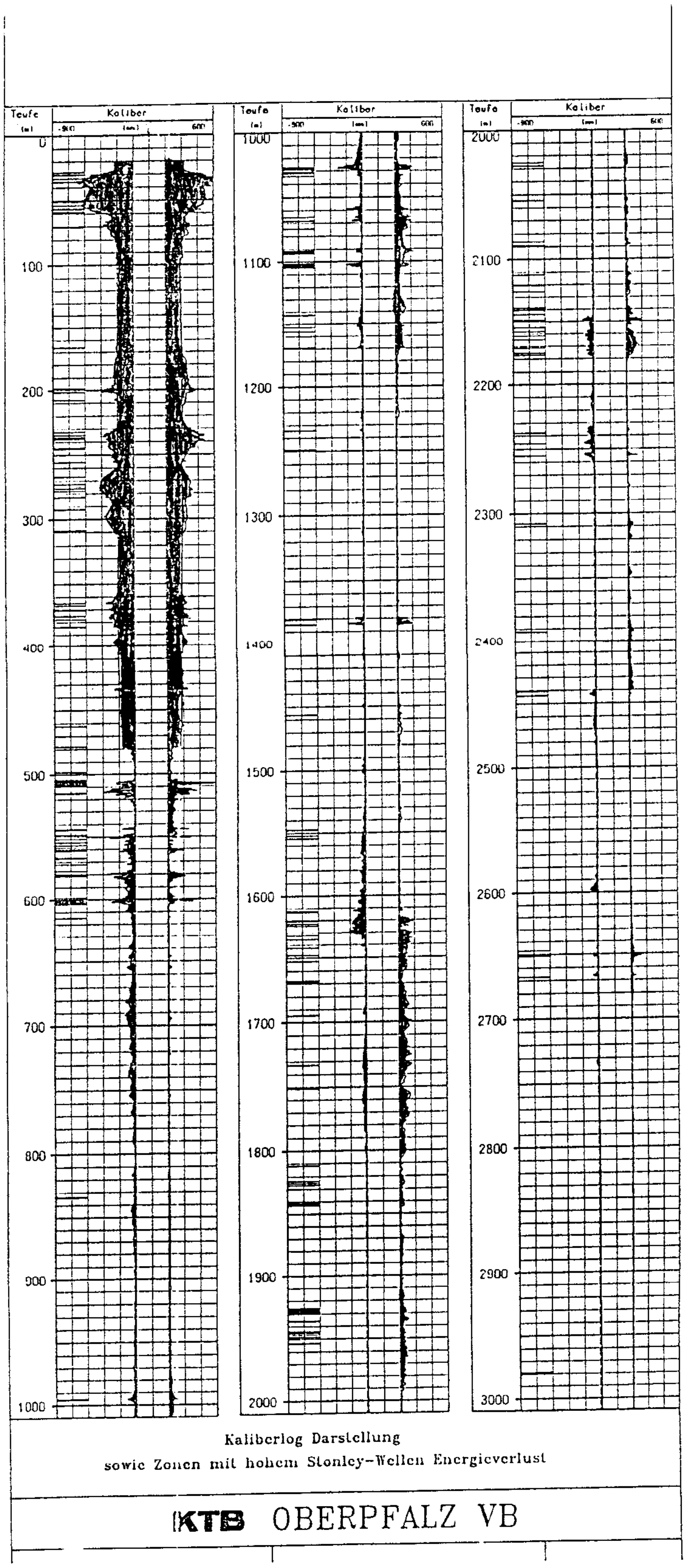


Figure 3.7-4 Breakout log from the KTB pilot borehole down to 3000 m. Shaded regions indicate zones where breakout is present. Data from KTB report 89-1.

There are several interesting results from the two wells up to now. Measurements of the temperature and stress in the Cajon Pass borehole indicate that the San Andreas fault is weak with low shear stresses and low heat flow (Zoback et al, 1989). Hydraulic tests in the well indicate that the permeability is low (on the order of 10^{-18} m^2) at 2 km with distinctly different fluid compositions in adjacent fracture systems (Kharka, 1988). The latter observation is evidence for that convection is of minor importance as a heat transport mechanism at the Cajon Pass site. In spite of the lack in connectivity in the fracture systems the rock behaves in a seismically anisotropic manner (Li et al, 1988) with shear wave velocities differing by up to 10% depending upon their polarization. The fracture systems are also responsible for most of the reflections seen on the VSP that has been run the borehole (Leary et al, 1988). Although the Gravberg-1 well is not strictly a scientific well it has provided some scientifically interesting results. The source of the high amplitude reflections observed on surface seismics in the area (Juhlin and Pedersen, 1987) have been identified as dolerite sills which have intruded into the surrounding granite (Juhlin, 1988 and 1990a). The borehole shows extreme breakout below 1500 m with the direction of the breakout approximately parallel to the minimum stress direction and with the long axis being about 70% greater than the short axis.

The goals of the study are to present some comparative results from the two wells which are related to fracturing. Inspection of the resistivity, density and sonic logs indicate that the rock in the Cajon Pass well is significantly more fractured than that from the Gravberg-1 well. This is, perhaps to be expected since the Cajon Pass well is being drilled in a much more tectonically active environment. However, analyses of seismic attenuation (Juhlin, 1990b) indicate that at least the upper 1500 m in the Gravberg-1 well has a higher attenuation than the Cajon Pass well. This is somewhat surprising and may be related to fundamental differences in the pore structures of the rock. In fact, porosity measurements on core from shallow boreholes in the vicinity of the Gravberg-1 well show that some granite cores have up to 9% porosity. Hydraulic tests in the Gravberg-1 well indicate the permeability to be on the oreder of 10^{-16} m^2 in the interval from about 1.3 to 3.2 km which is 1-2 orders of magnitude greater than that measured at Cajon Pass. This may also be related to differences in the pore structure or to the material occupying the pore space.

Both the Cajon Pass and the Gravberg-1 boreholes had natural gamma ray, deep reading resistivity, sonic and density logs run in them with instruments supplied by the same service company. This allows a direct comparison of some of the basic log responses of the two boreholes and provides a quick overview of how some rock properties change with depth.

The natural gamma ray log is indicative of the lithology of the formation and is primarily sensitive to the concentrations of potassium, uranium and thorium in the rock. In the case of the Gravberg-1 well (fig. 3.8-1) the moderately high gamma readings (at 0-1200 m, 3100-3500 m and 3850-4800 m) correspond to the more felsic varieties of the Dala granites which were identified by Karlsson (1988) as being higher in SiO_2 content. The moderately low gamma readings (at 1200-3100 m, 3500-3850 m and 4800-6000 m) correspond to the more mafic varieties as identified by Karlsson (1988). The extremely low values (about 25 API units) indicate intervals where dolerite sills have intruded into the granite which are the source of the high amplitude reflections observed in the area (Juhlin, 1988 and 1989). The low gamma reading at 3900 m is not due to the presence of dolerite since it is a zone in the well where the borehole was severely out of gauge,

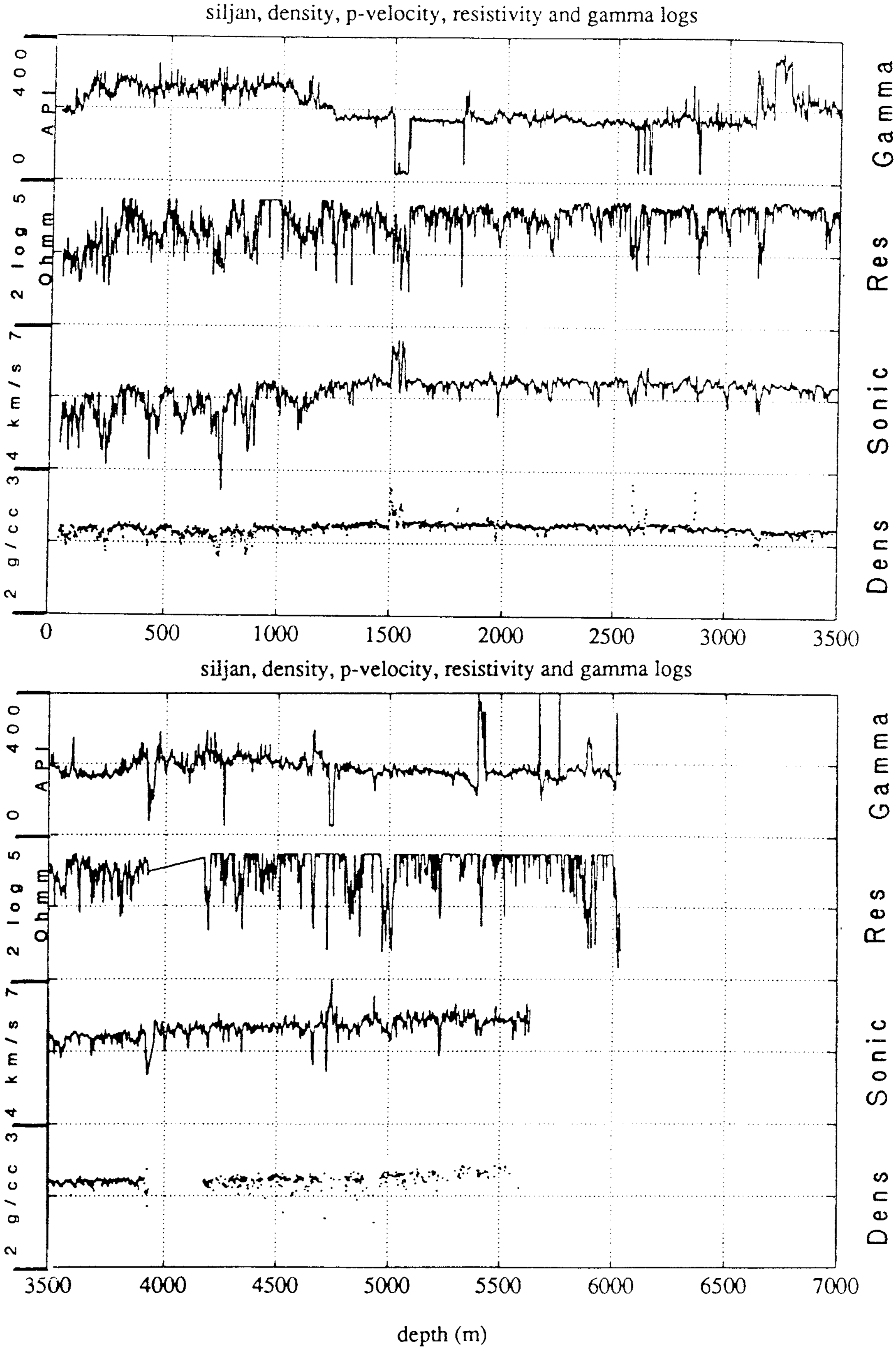


Figure 3.8-1 Overview of some of the wireline logs run in the Gravberg-1 borehole

otherwise all the gamma lows are due to dolerites. The extremely high gamma readings (about 400 API units) are indicative of intrusions of quartz monozonites which are fine grained granitic rocks (Collini, 1988) rich in thorium. The natural gamma log can, thus, be used to identify the basic lithology in the Gravberg-1 well. There are, of course, some exceptions to the pattern discussed above, for instance, in some fractured intervals the gamma levels will increase due to the presence of uranium or even thorium in the fracture system, but these cases are rare. In the Cajon Pass well the natural gamma ray log (fig. 3.8-2) can also be used to identify lithological units as discussed by Moos (1988) in the crystalline portion of the well. The low gamma readings (1050-1300 m and 1470-1830 m) correspond to granitic rock and the higher ones (500-1050 m and 1300-1470 m) to gneissic rock in the upper part of the borehole. This pattern continues in the lower part of the borehole as well (Broglia, personal communication). It is clear from the natural gamma logs that the rocks in the Cajon Pass well are significantly more mafic than the rocks in the Gravberg-1 well as indicated by the lower gamma readings.

The deep reading resistivity log is designed to read the true resistivity of the surrounding formation and is not generally affected by borehole conditions. It is considered to give accurate resistivities up to 40,000 Ohmm (Pezard et al, 1988). In the Gravberg-1 well the resistivity is quite variable in the upper 1200 m (fig. 3.8-1). Below 1200 m, the resistivity is generally close to 40,000 Ohmm or greater except over certain intervals where it is considerably lower. These intervals correspond to fracture zones. There appears to be a trend in the resistivity data where the resistivity in these fracture zones decreases with depth. The lowest resistivity recorded approaches 100 Ohmm at about 6000 m. The resistivity in the Cajon Pass well (fig. 3.8-2) is generally lower than that in the Gravberg-1 well, but there is a trend towards increasing average resitivities with depth. Some of the lower resistivity intervals in the Cajon Pass well also correspond to fracture zones as discussed by Moos (1988) particularly the ones at 850 m and at 1440 m. The marked resistivity low beginning at about 2500 m correlates well with a prominent seismic reflector on the VSP (Leary, personal communication) and is the zone where the drillstring was lost when drilling stage 2 of the well (Andrews et al, 1989).

The sonic log has provided data on P and S wave sonic velocities in the two wells. In the Gravberg-1 well there is a gradual increase in the sonic P wave velocity in the upper 1500 m (fig. 3.8-3) from about 4.5 km/s to about 5.8 km/s, although if fluctuates greatly. Below 1500 m the sonic velocity is relatively constant at about 5.85 km/s down to about 3100 m. Between 3100 and 3800 m the P wave velocity is lower than this and from about 4000 m down to 5650 m there is an increasing trend with velocities of about 6.15 km/s in the deeper portion of the interval. Superimposed upon these average trends are discrete intervals with significantly lower velocities. These zones of lower velocity generally correspond to zones of lower resistivity (fig. 3.8-1) which have been identified as fracture zones. Sonic S wave data is only available below 1250 m in the Gravberg-1 well with the S wave velocities fluctuating in a similar manner as the P wave velocities. However, there is a trend towards comparatively higher S wave velocities with depth as can be seen in the V_p/V_s (ratio (fig. 3.8-3) which decreases from about 1.8 at 1250 m to about 1.73 at 5650 m. Note that the V_p/V_s ratio in the dolerite sills are distinctly different from that in the granites with values of 2.0 being typical. In the Cajon Pass well (fig. 3.8-4) the average sonic P wave velocity increases rapidly from 500 to 700 m from less than 4.0 km/s to about 5.5 km/s. It then increases gradually in the interval from 700 m to about 1800 m where it reaches a value of 5.85 km/s which is comparable to that found in the Gravberg-1 well at similar depths. The sonic S wave velocity in the Cajon Pass well also tends to fluctuate in step with the P wave velocity and there is similar trend of a decreasing V_p/V_s ratio depth (fig. 3.8-4). Note the marked decrease in sonic velocities at 2500 m which is the source of the seismic reflector mentioned earlier.

The density log is generally the poorest quality log of the ones shown in figures 3.8-1 and 3.8-2. This is because it is dependent upon good contact between the tool and the borehole wall while the others are not. If the contact is poor then the log will tend to read too low a density compared to the true reading if contact was satisfactory. For this reason the density readings have been plotted as points in these figures instead of using lines as was done with the other logs. In the Gravberg-1 case, points where the density was obviously in error have been omitted. This has not been done for the Cajon Pass well and the low values in the lower part of the well are generally due to poor contact (Broglia, personal communication). Below 1000 m in the Cajon Pass well the density log generally reads higher than the density log in the Gravberg-1 well where the log quality is good. This is consistent with the rock in the Cajon Pass well being more mafic than in the Gravberg-1 well. The peaks in the Gravberg-1 well where the density approaches 3.0 g/cc correspond to the dolerite sills and troughs where the density approaches 2.5 g/cc corresponds to the fracture zones. There is a noteable increasing trend in the density in the interval from 5000 m to 5650 m in the Gravberg-1 well where the density increases up to about 2.7 g/cc at the lowermost point. Due to the poor quality of the density log in Cajon Pass well it is difficult to determine if any trends exist.

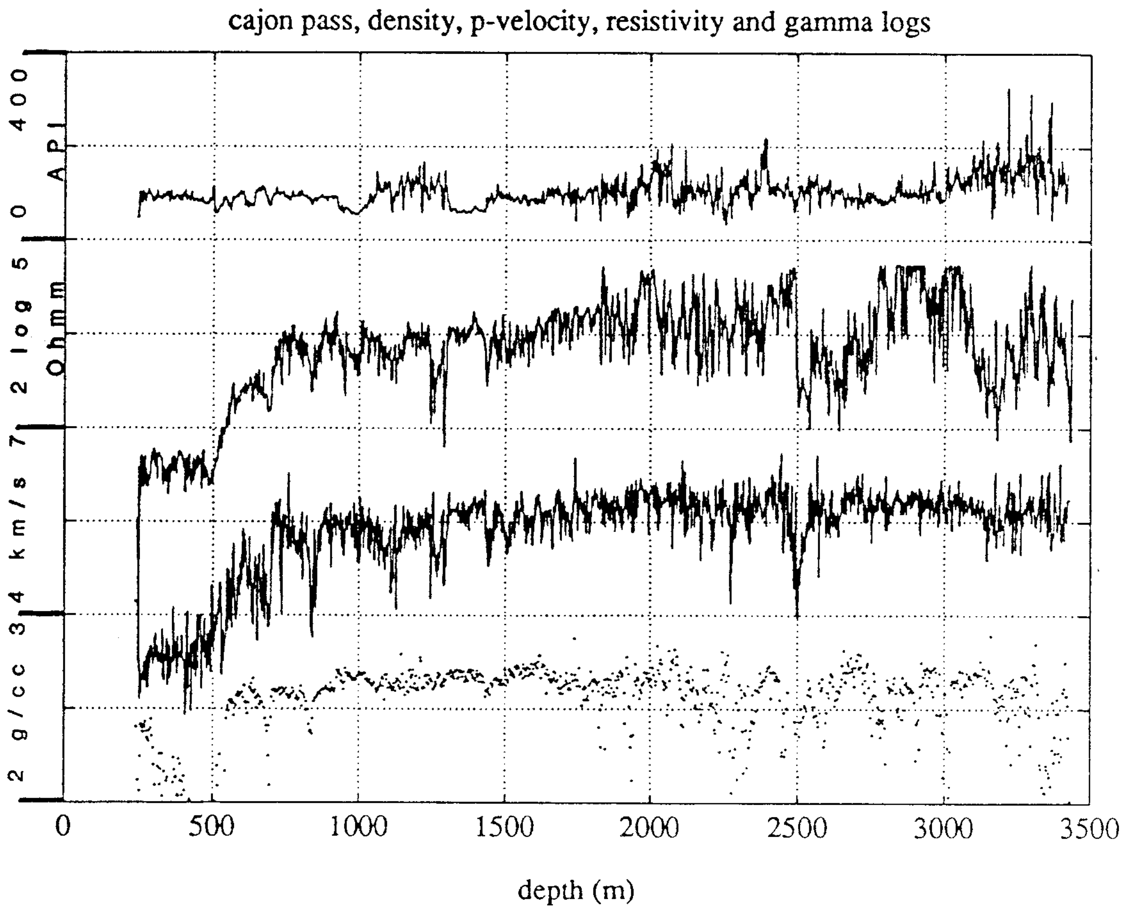


Figure 3.8-2 Overview of some of the wireline logs run in the Cajon Pass borehole

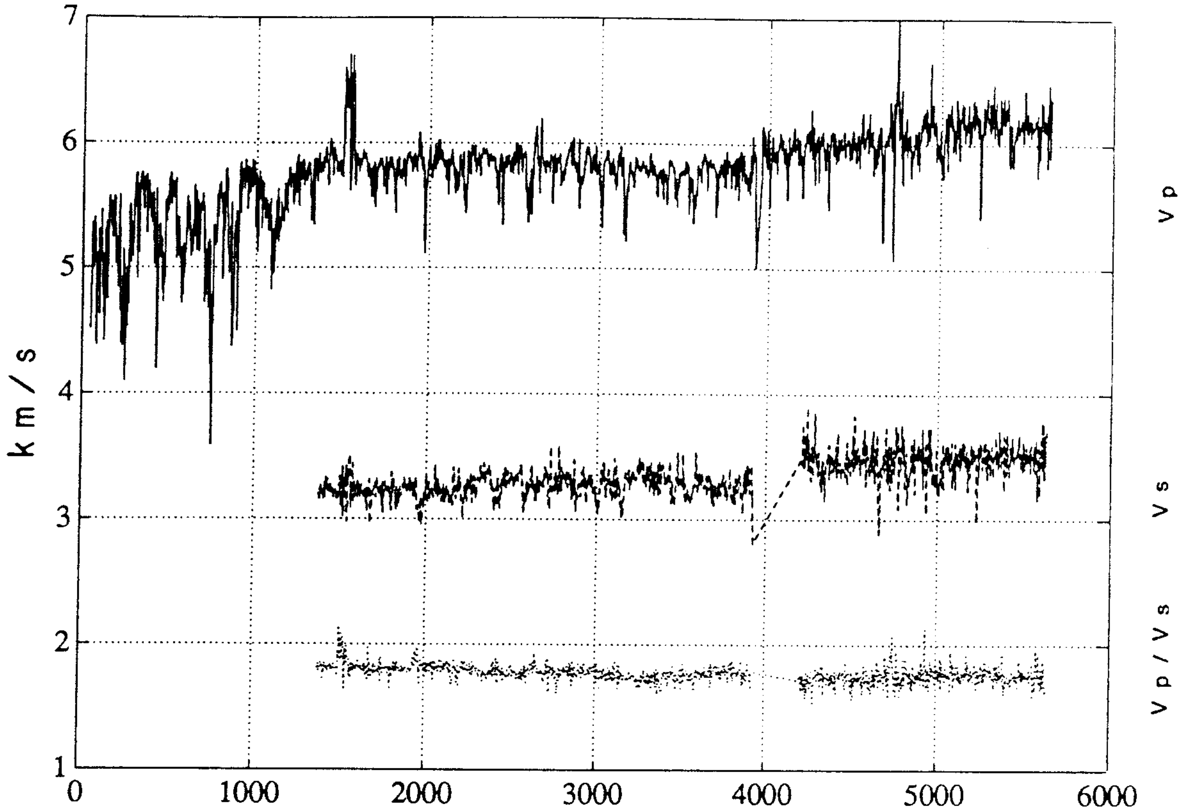


Figure 3.8-3 Variation in V_p, V_s and V_p/V_s with depth in the Gravberg-1 borehole

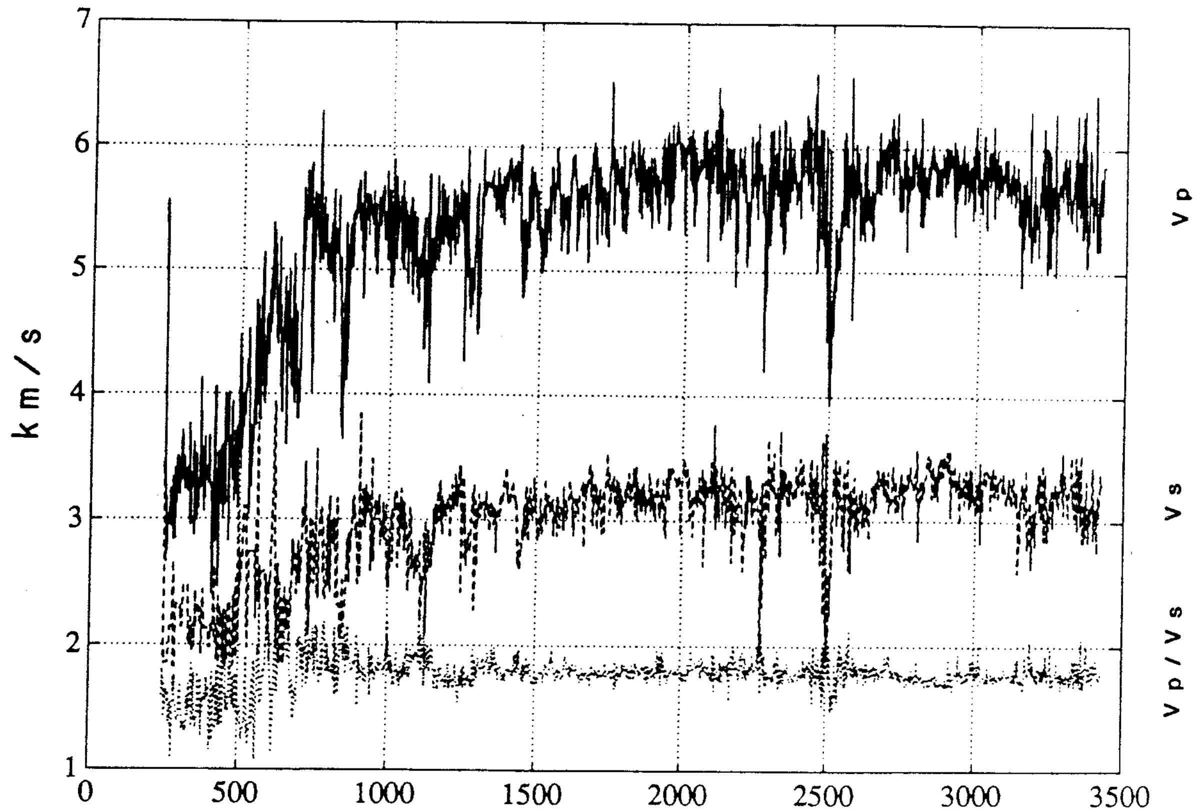


Figure 3.8-4 Variation in V_p, V_s and V_p/V_s with depth in the Cajon Pass borehole

4. GEOLOGICAL INVESTIGATION PROGRAMME

4.1 Surface investigations

The necessary geological surface investigation programme for the localization of a deep borehole storage site will not in principle differ from what has already been used within the KBS-project. Good knowledge of the geological prerequisities at the surface is vital, since many geological features are deep seated. It is important to link together surface information with features monitored at depth with various geophysical techniques.

The geological investigation programme presented in this report concentrates on different geophysical surface investigation techniques and various methods for investigations in deep boreholes.

4.2 Surface geophysical investigations

Relevant surface geophysical methods for investigating the rock below a depth of 1 km are:

1) Gravity
2) Magnetics
3) Magnetotellurics (MT)
4) Seismic refraction profiling
5) Seismic reflection profiling
6) Seismicity monitoring

Although their depth of penetration is less than 1 km, the following geophysical techniques are considered useful for determining the quantity of fracturing at or near the surface:

7) Remote sensing.
8) VLF measurements.

4.2.1 Gravity

The resolution of the gravity survey is dependent upon the spacing of the stations. At most, a high resolution survey can detect features on a scale of 50 m. In spite of the relatively poor resolution, gravity surveys can provide information on a regional scale, such as the extent of the granitic pluton or large-scale tectonic features at depth.

4.2.2 Magnetics

The resolution of magnetic surveys is better than that of gravity surveys, and the cost of acquiring high quality data over a large area is considerably less since the data can be collected using airborne magnetometers. The magnetic data can be very useful for detecting both linear and circular features which are not directly observable at the surface, such as dolerite dykes.

4.2.3 Magnetotellurics (MT)

MT surveys also have rather poor resolution, especially since the medium being investigated is not horizontally stratified. However, they can give qualitative information on the degree of fracturing present in the upper crust. They also have an advantage over DC methods in that they detect conductive layers in the crust, which are probably related to fracturing, rather than resistive layers which are representative of intact rock. MT surveys can also give important information about the structure of the crust in the area under investigation. For more site-specific investigations controlled source MT surveys (CSMT) are prefarable to those that use the natural sources to study the conductivity structure of the crust. MT surveys may be very usual for estimating the depth to higher salinity waters in an area.

4.2.4 Seismic refraction profiling

Through seismic refraction profiling, the velocity structure in the upper crust can be determined and the depths to the more intact rock which is expected to be encountered at 1000-1500 m can be more accurately estimated. As an example, the refraction profiling over the Gravberg well prior to drilling accurately predicts the observed velocity increase in the upper 1500 m. Refraction profiling will also assist in locating near-surface inhomogenities.

4.2.5 Seismic reflection profiling

Seismic reflection profiling is an important part of any surface investigation for site characterization. The experience from Gravberg shows that the method is capable of detecting anomalous zones 5 m in thickness at a depth of 3-4 km, provided that these zones have lateral continuity. Current commercially available techniques focus mainly on imaging horizontal reflectors, and in order to image dipping reflectors, such as fracture zones, both field procedures and processing techniques have to be adapted to this goal.

4.2.6 Seismicity monitoring

Monitoring of seismic activity in an area can be conducted passively, and considerable data already exists covering the whole of Sweden. However, it may be worthwhile setting up a dense network of stations in the area of investigation to improve sensitivity and resolution. This is something which should be done at an early stage in the site characterization programme.

4.2.7 Remote sensing

Remote sensing is a useful reconnaissance technique in the early stages of site characterization. It should be possible to detect large fracture zones which extend to the surface, and these areas should be avoided immediately. Depending upon the satellites being used, resolutioin is on the order of 30 m. However, there is virtually no vertical penetration into the ground.

4.2.8 VLF measurements

VLF techniques are useful for locating fracture zones at the surface, but current technology is highly dependent on the location of the transmitter, and zones which are oriented at certain azimuths go undetected. If new techniques are developed, as has been proposed, this biasing problem will be solved by using two transmitters to locate fracture zones. In addition, the new techniques would allow for determination of the strike and dip of the fracture zones, as well as to some degree, a qualitative estimate of the intensity of fracturing.

4.3 Investigations in deep boreholes

4.3.1 Geophysical borehole logging

Geophysical borehole logging or electric wireline logging is a collective name for a very large number of different measurements that can be made in a well. Logging can be defined as:

> The Continuous Measurement of various Physical Properties related to the environment around the borehole recorded at Downhole Conditions using electronic sensors lowered into the well by an electric wireline.

The gathering of information related to the rock environment in the borehole becomes increasingly difficult with increasing depth of the well. Standard coring at great depth is an expensive and sometimes hazardous operation with the risk of unnecessary drilling problems, and the recovery is very often far from satisfactory.

Cutting studies become increasingly difficult with selected minerals having a larger probability of being ground to such fine fractions that they are lost in the mud-cleaning system. Different drag coefficients for various minerals and cutting sizes as the mud is circulated while drilling causes problems in describing the mineralogy drilled through. Many important thin features have a tendency not to be seen in the cuttings.

It is important to realize that many of the required properties cannot be measured in any other way than using wireline logs. It is a common misunderstanding that the best way of obtaining reliable information on the rock is by taking core samples. This may be true for detailed studies on the mineralogy, but as far as other information on the rock environment is concerned, this is not the case. As an example, mention can be made of porosity, where core can give very good information on the spot porosity measured at surface conditions, but this may not be representative for the downhole conditions with much higher pressure and the investigated rock sample represents a very small portion of the rock. In fractured rock, the cores tend to break in the fracture plane, and the problem is to decide if the crack was original or induced by drilling. Furthermore, it is very often the case that the core losses occur in the fractured sections. Another area where the information can be extracted only by logging is the pore fluid. The logging measurements are primarily indirect and based on evaluation of measurements of the fluid conductivity, density, dielectric constant, speed of sound, etc. Only a successful full-scale production test can give better information on the movable pore fluid. Many other types of information can also only be obtained by wireline logging.

4.3.1.1 Major requirements for the storage well investigation

Many different requirements will have to be considered when designing a logging programme for a deep borehole for potential nuclear waste storage. This discussion will only highlight some of the most important areas where wireline logging may help to evaluate the geological conditions.

Fractures, fracture zones and zones of weakness

For the purpose of the well it is of the utmost importance to identify the fracture zones, zones of weakness and individual joints. It must be possible to localize the fractures in order to gain a better understanding of the potential fluid migration paths that may exist. Orientation of the fractures is also of importance for the rock mechanics studies. It must be possible to distinguish the natural or virgin fractures from drilling-induced fracturing near the well bore.

Permeable zones

It is of great importance to be able to identify permeable sections of the well. With the concept of a deep well it is not practical, and most probably not possible, partly due to drilling fluid plugging and borehole conditions, to perform selective production tests over the very large interval. It may be possible to gain an overview of potential permeable zones by the use of continuous logging.

Lithology

A good understanding of the lithology penetrated by the well is important in its own right, and it is vital to be able to recognize changes in lithology, since the boundaries are often associated with zones of weakness. Special minerals may have a relevance to the present and past fluid migration system and are therefore important to identify.

The mineralogy must also be known in order to be able to carry out the porosity calculations. The reason for this is that all measurements related to porosity are also more or less affected by the properties of the minerals forming the rock matrix.

Fluid properties

The fluid environment of the rock has an impact on the possible design of the storage and must be well known.

Rock mechanical properties

For the understanding of the rock behaviour under different stress conditions, it is of importance to know the mechanical properties of the rock.

Porosity

The porosity distribution, type and size in combination with the fractures, have a very great impact on the fluid migration process through the rock.

Borehole properties and damages

The drilling of a well through the rock column at great depth will affect the rock and change the condition of the rock in the vicinity of the borehole. The large breakouts, considered to be related to horizontal stress differences, that have occurred in the Gravberg well and in many other wells drilled in hard rock environment, is a very obvious example of change in condition of the rock. When drilling wells to great depths in hard rocks it is most likely to have the conditions that result in breakouts, and proper measures have to be taken to be able to handle this. Other changes of the rock may not be as dramatic in its immediate appearance, but may have considerably more serious effects on a potential nuclear waste storage situation.

Large scale structural information

The extent of fracture zones and their distribution on the large scale play an important role in safety assurance of the storage site.

4.3.1.2 Logging techniques applicable to the problem

The following is a discussion on some logging techniques that have a special application to the investigation of borehole for nuclear waste storage. The instruments discussed are not to be considered as the only ones required or of importance to the problem, but only as a presentation of some new instruments and evaluation techniques that have recently been introduced and some potential interesting areas where new developments in the near future could lead to interesting results.

It is important to realize that almost all logging measurements are indirect in the way they solve a problem, and only in a combined well-designed logging programme can the full use of an individual logging instrument be made.

FMS Formation Microelectric Scanner

This instrument is equipped with 4 hydraulicly operated arms each of which has a pad mounted on the arm. The pads are forced against the borehole wall and pulled up the hole by the wireline at a constant speed. Two of the pads each have 2-button electrodes which are used to send out an electric current into the formation. The current leaving the button is focused by a current leaving the pad itself. The button current is measured and can be related to the formation resistivity just in front of the button. These two pads are identical to the pads used by the SHDT Stratigrafic High-resolution Dipmeter Tool with which the FMS has major parts in common. The two remaining arms hold the FMS pads. These pads have a multitude of electrodes, mounted closely together and each recording a resistivity-related curve. (A new version of this tool being field tested at present uses 4 FMS pads for better coverage of the borehole wall.) These resistivity curves are used to build a resistivity-related image of parts of the borehole wall. Using various enhancement techniques different features can be studied. The two images created in this way each represent a 7 cm wide strip spaced 90 deg from each other.

The instrument is also equipped with a 3-component magnetometer as well as a 3-component accelerometer. These are used to orient the features in a north-south-vertical coordinate system.

The instrument is mainly used in the oil industry for fracture identification, sedimentation environment studies and information on the rock structure in general. This tool is particularly well suited to the crystalline environment, with the normally high contrast between anomalies and the crystalline country rock. In fact, an early prototype of the tool was first designed for use in the Nagra project in Switzerland.

Open fractures are often easy to identify in the image. This is due to Ace fact that the fractures are filled with drilling mud or mud-filtrate that has invaded the fracture during the drilling process. This fluid has a very large conductivity in comparison to the virgin rock. Other zones of disturbance are often also seen by the instrument as contrasting features. This could be due to alteration processes causing the altered material to be more conductive or alternatively healing processes, where the healing mineral is less conductive than the country rock.

The detailed interpretation of the images is normally done using an interactive workstation, where the image can be enhanced in different ways and the features observed can be oriented by interactive correlation point picking. Fully automatic evaluation programmes are being developed, but are not available commercially at the present time.

The main advantage of this instrument is that it eliminates a lot of the guesswork done previously when other conventional logs were used for fracture indication. Furthermore, the tool is not as dependent on the drilling fluid and the size of the borehole as the other alternative image tool: the Borehole Televiewer.

Figure 4.3-1 represents an example of an FMS image from the Gravberg-1 borehole.

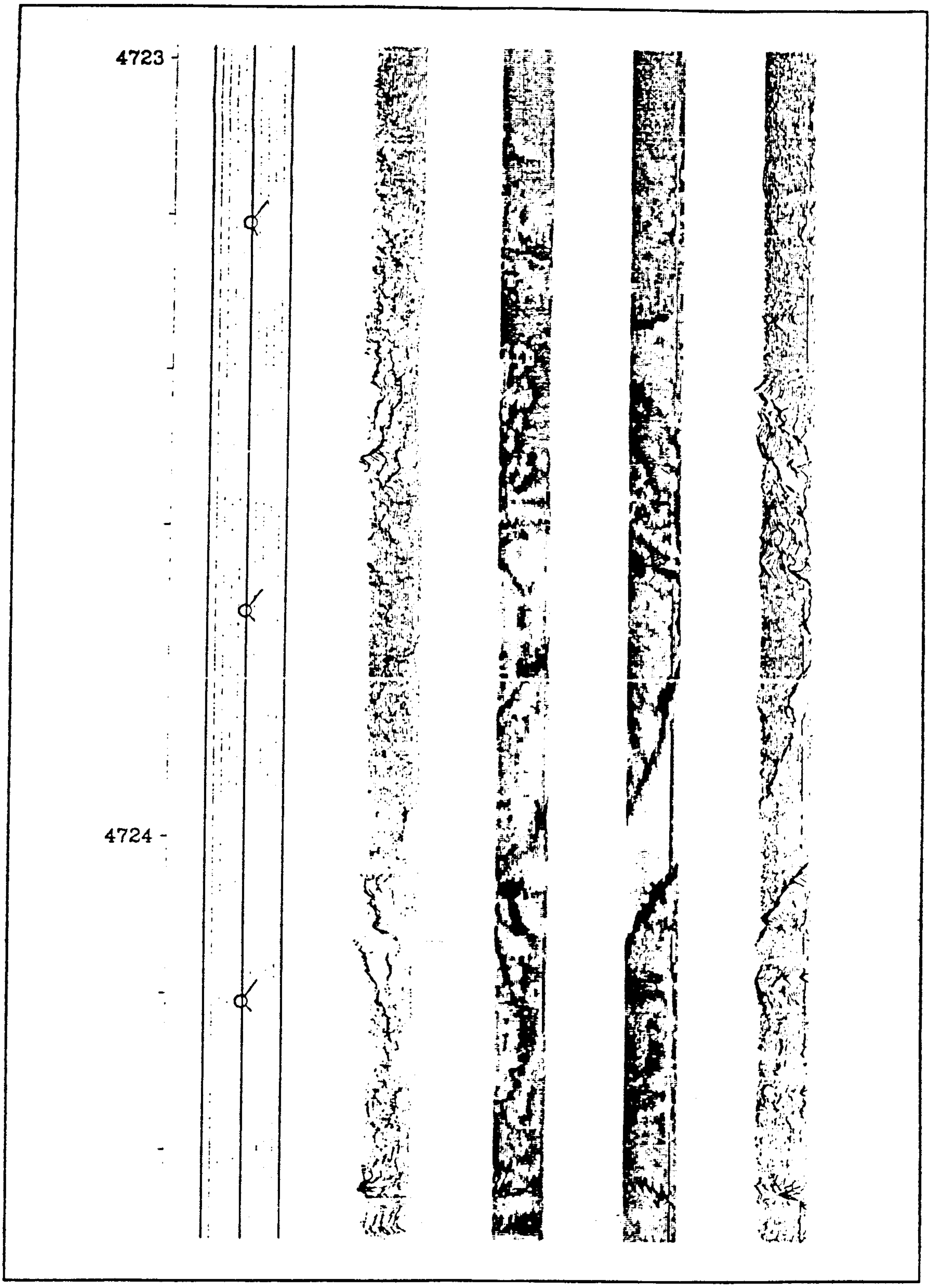


Figure 4.3-1 FMS image from the Gravberg-1 borehole with clear, strong fracture patterns.

Digital Sonic Tool

This instrument is the latest improvement of one of the basic techniques used in the logging industry, acoustic or sonic logging. This instrument uses two transmitters and an array of receivers to register acoustic wave trains, that are generated by transmitters.

Techniques the compressional, shear and Stoneley waves can be isolated By different filtering.

The speed of the compressional wave is the traditional measurement and has been used for porosity, lithology and fracture evaluations.

The shear and compressional waves in combination with the rock density are used for deriving the mechanical properties of the rock. It has been shown that the Stoneley wave loses energy across permeable zones, and this has been used to indicate permeability. The reflection of Stoneley energy has been observed in open fractures. Other techniques for permeability indication based on sonic data use the ratio between the three first events of the compressional wave to derive a permeability log.

The geochemical logging tool

This is a combination of instruments that have recently been improved to be able to register elemental yields of certain important rock-forming elements. The present set up of the tool measures the abundance of Ca, Cl, Fe, H, Si, S, Ti, Gd, U, Th, K, Al and Mn. The abundance of Mg can be derived indirectly from another logging source. In a stationary mode the tool can also measure the contents of C and O. The different yields can be used to derive the rock-forming oxides, and from there it is possible to reconstruct the mineralogy using a so-called normative mineral approach. The instrument is based on a neutron activation principle in which the resulting gamma ray spectrum is measured by the instrument. This is combined with the information given by the measurement of the natural gamma ray activity from U, Th and K bearing minerals.

The germanium spectroscopy tool

This is an instrument based on the same principle of neutron activation as the instrument described previously. However, this instrument uses, a greatly improved detection system with a liquid nitrogen-cooled germanium detector. This detector has a much better spectrum resolution, which can be used to detect elements at trace levels. In the literature (Hallenburg, Geophysical Logging for mineral and engineering application) it is reported that gold was detected at 5 ppm levels with a $\pm 20\%$ accuracy. The detection level for various trace elements is a function of the response to neutron activation, and has to be simulated for each case. Due ot statistical requirements, measurements with this instrument are made at stations, rather than during continuous movement.

The instrument is not commercially available at present and no development work is in progress owing to its limited commercial application to the oil industry. Scintrex and Princeton Gamma Tech. are reported to have worked in this area. Other instruments may exist as university research prototypes.

Borehole radar

For detection of fracture zones at a distance from the borehole, the radar principle has an interesting application. The limitation of the directional information may be overcome by evaluating the formation scanner log borehole images in combination with the borehole radar data, if multiple wells are not available.

4.3.1.3 Limitation in the use of logging instruments

Hole size

Normal logging instruments have been designed for accurate use in borehole sizes from 12.5" down to 6.5" diameter. All instruments are affected to a greater or lesser degree by the size and conditions of the borehole. The virgin rock condition is found by correcting for the influence of the known borehole conditions. In larger borehole sizes, the influence of the borehole becomes large and the uncertainty in the correction increases. Hole sizes above 17" can normally not be logged with any accuracy. If a large hole size is required over sections where detailed information is also an important requirement, it may be necessary to drill a smaller hole, which is later reamed to final size.

Drilling fluid

The mud properties may have a destructive influence on the logging results. Examples of this are large fractions of barite additions or oil based mud systems. This has to be considered when the mud programme is designed.

Hole conditions

The breakout enlargement of the hole and the associated borehole rugosity create problems with regard to registering data primarily with the so-called pad tools. The effect can partly be overcome with the short axis logging kits that turn the tools to log on the short axis of the hole, where the wall conditions are normally better. For deep reading tools the breakouts are less of a problem.

The rugosity of the hole can also create serious problems, such as when running instruments down. This problem can be overcome by redesigning the bottom hole finder on the instruments to be used.

The logging instruments can alternatively be pushed down using a drill pipe-conveyed system. This is, however, a very time-consuming method and not realistic for long logging intervals.

Evaluation principle

Log evaluation in the sedimentary environment is well-known and established principles can be used with good results. Log evaluation in the crystalline environment is, however, still not very well-known and work has to be done to improve the evaluation of data from logging. However, the NAGRA project and other scientific projects, such as Cajon Pass and the

KTB drilling, will result in valuable new experience for evaluation of logging data in crystalline rock. It is the opinion of the author of this report that the knowledge of wireline logging in crystalline environments will increase substantially in the near future.

4.3.2 Drilling-induced fracture detection

The drilling of a well through a rock sequence can have many different effects on the virgin situation of the rock.

- Release of anisotropic stress causing breakouts and spalling in the least horizontal stress direction and potential tensile fractures in the maximum horizontal stress direction.
- Mechanical cracking of zones of weakness or healed fracture systems improving the fracture porosity locally around the borehole.
- Hydraulic fracturing due to heavy muds.
- Invasion of drilling fluids into the existing pore space, replacing or mixing with the virgin fluids.

An important requirement of an investigation programme is to be able to see beyond the drilling disturbed zone. A description is given below of some of the methods that can be used for this purpose. It is important to realize that the numerous different situations that can be found in a well cannot be analysed by a single method only, but that only a comprehensive evaluation of a combined logging programme can give a good picture of the situation.

Breakout detection

The breakouts and resulting spalling of the rock are normally detected by the use of multiple arm calipers. The 4-arm calipers of the Borehole Geometry Tool (BGT), the Stratigrafic High Resolution Dipmeter (SHDT) or the Formation Microelectric Scanner (FMS) measures the diameter of the hole in two directions perpendicular to each other. One of the arm pairs will have a tendency to follow the larger diameter of the hole, forcing the other pair to measure on the short axis of the hole. With the aid of the tool orientation system, the maximum diameter, normally associated with the minimum horizontal stress direction, can be oriented. Multiple caliper arm instruments are also available but these instruments are normally only used for cased hole logging.

An alternataive method for detecting breakouts is to use the Acoustic Televiewer instrument. This instrument measures the radius of the hole in more than 300 directions around the instrument by a scanning system. These data allow for more detailed analysis of the breakouts. However, it is not possible to use the standard tools that are available in heavy muds or large diameter boreholes.

Drilling induced fracture porosity

Most logging instruments measure the conditions along the borehole wall or in the rock close to the borehole. This means that porosity-reading devices like the density of the neutron porosity instruments will mainly read in the damaged zone if drilling damage exists. The first step in eliminating the effect of the drilling damage is to be able to detect that such damage exists. This can be done by the use of a multiple spacing sonic instrument. This tool records the travel time of an acoustic pulse through the rock close to the borehole by the use of a transmitter and a pair of receivers. By increasing the spacing between the transmitter and the receiver pairs, the depth of investigation can be made deeper and deeper. If the speed of sound measured by the shorter spacings is slower than the longer spacings, then drilling damage exists. In a drilling-damaged zone, the measured speed of sound should be faster and faster the longer the spacing until the depth of the damaged zone is such that the measured signal has followed the faster undamaged rock for the two signals used to compute the speed of sound. By ray-tracing and processing similar to refraction processing, the sound speed in the virgin rock can be calculated at the same time as a measurement is given of the drilling damage depth and the magnitude of the drilling-induced apparent porosity in the near borehole zone. This data can then be used to correct other porosity-related measurements.

The deep reading resistivity (LLD) measurement of the dual laterolog tool is a measurement that is affected only very slightly by the borehole and the near borehole zone. This instrument can be considered to measure the formation in a cylindrical fashion where the cylindrical shells of varying resistivity are measured in series. This means that a low resistivity shell (damaged zone) close to the borehole will be virtually neglected if a high resistivity formation (virgin unfractured rock) exists behind the damaged zone. A large difference between the LLD and more shallow reading resistivity measurement (LLS) may mean drilling damage, althought it can also be due to vertical fractures or large differences in the drilling fluid and the formation fluid conductivity.

Fractures due to hydraulic fracturing

These types of fractures are considered to be close to vertical and found in the direction of the maximum horizontal stress in an otherwise unfractured rock. These types of features can be found by the use of imaging instruments . A large difference in the deep and the shallow reading resistivity, made by the dual laterolog tool, is expected in this situation. In those cases in which the hydraulic fracturing has improved, existing fracture systems will be difficult to detect unless measurements have been taken in before the hydraulic fracturing and comparative studies can be made on the data before and after fracturing. Experience from different scientific projects shows that it is difficult to detect fractures created by hydraulic fracturing.

Fluid invasion

In normal overbalanced drilling, a loss of drilling fluid will always occur in the porous and permeable zones. This fluid will be flushed into the formation pores replacing and mixing with the virgin fluid. When exploring for hydrocarbons, it is of the utmost importance to be able to determine the pore fluid beyond the flushed zone. This is normally done by the use of the different resistivity measurements with different depths of investigation. In this case the fluid conductivity contrast is used. In the crystalline environment with a large proportion of sub-vertical fractures, which have different effects on the different focusing methods, the fluid composition evaluation becomes more complex.

4.3.3 Borehole seismic investigations

Borehole seismic operations generally consist of activating a source at the surface and recording the signal in the borehole. Three common source-receiver configurations are vertical seismic profile (VSP), offset seismic profile (OSP) and walkaway seismic profile (WSP). The VSP involves employment of a source near the wellhead and recording the signal at some pre-set interval in the borehole (generally at 5-25 m spacing between recording positions). The incoming seismic waves travel vertically and the traveltimes allow calibration of the wireline sonic travel times as well as a time to depth conversion of the surface seismic. In addition, VSP permits identification of reflecting horizons on the surface seismic as well as identification of reflecting events not present on the surface seismic. VSP also enables detection of possible reflecting events below the current depth of the borehole that is being drilled. In the case of OSP, the field procedure is the same as for VSP, except that the source is placed at a considerable distance from the borehole, which gives lateral information away from the borehole, such as dips of reflecting horizons. In WSP, numerous source positions are used on the surface while recording is limited to a few levels in the borehole. WSP also gives information away from the borehole.

The primary goals of VSP will be:

1) To allow a depth to time conversion in order to correlate events in the borehole with events on the surface seismic.

2) To confirm that events observed on the surface seismics truly reflect the subsurface geology.

3) To look ahead of the bit and possibly image reflections below the current depth of the well that could not be imaged from the surface.

4) To image horizontal features in the vicinity of the borehole which would be missed using crosshole techniques.

From the experience gained at Gravberg-1, it is obvious that dynamite is a preferable source to airguns for generating high quality signals into the bedrock. No studies have been carried out regarding the capability for dynamite to function as a repeatable source which is necessary for goals 2-4 to be accomplished. However, through the use of shotholes drilled fairly deep into the bedrock (20-30 m) the source signature should be consistent enough to allow imaging of events by current processing techniques. The drilling of deep shotholes will also enable the same shothole to be used for numerous shots and thereby reduce the cost of using dynamite in comparison to airgun sources. It is also possible that vibrator sources can be used to generate the signals. However, no tests have been made regarding the frequency content of their signals and how they compare with dynamite sources.

4.3.4 Cuttings Studies

It is much more difficult to take cores during drilling of a deep well in igneous or metamorphic rock than to use normal slim hole coring with mining techniques. To overcome the problems of limited cores, and especially from those intervals that are of most importance, the fracture zones, it is possible to use other sources in order to acquire bedrock information. Rotary drilling produces cuttings which, with proper analytical treatment, will give vital information on the bedrock, especially if these studies are combined with wireline logging data.

The type of information to be acquired from cuttings is of both mineralogical and geochemical nature.

The mineralogical information is obtained from modal type analyses with a reflecting light microscope of etched and stained samples made as thin sections. This can normally be done on the drillsite during a drilling operation. The on-site analyses can be supported by more sophisticated analyses in lab facilities, such as electron microscope, microprobe and Mössbauer-analyses. The modal analyses give information on the mineralogical composition of the bedrock, while the lab analyses give information on alterations in cuttings, different types of mineralizations, type of accessory minerals, fracture filling minerals and cataclastic fragments (indicating brecciation in fracture zones).

If the sampling grid is short enough, at least one sample every 5 m, these analyses give such useful information that full-size cores are only needed for spot calibrations of the cuttings and for rock mechanical tests, especially if the cutting analyses are combined with sidewall coring. The sidewall coring is then used for sorting out unresolved questions or for confirmation of features seen in the cuttings.

The geochemical analyses are used to obtain information on the variations in major and trace elements in the bedrock.

During the deep drilling at Gravberg-1, this type of cuttings information was used to evaluate different lithologies and rock properties with encouraging results. To give an example; one interval that showed altered cuttings and on the wireline logs porosity, was sidewall cored and had a porosity in the core of almost 4%.

With proper evaluation, continuous coring from surface to TD is not necessary to make a good interpretation of the bedrock and fracturing. A combination of selected cuttings analyses and wireline logs, combined with sidewall cores, will give satisfactory information on the bedrock at a far lower cost than conventional continuous coring.

4.3.5 Coring

A borehole for storage of nuclear waste will probably be drilled with oilfield equipment. Normally when drilling deep boreholes for oil and gas only spot cores are taken to solve special problems such as reservoir and source rock properties and for stratigraphic control. The choice of coring spots is normally based on careful interpretation of rock cuttings and a seismic investigation giving a fairly good prediction of the depth to geological strata of interest. Continuous coring as at the Kola Super Deep borehole, is possible, but will be extremely expensive. When drilling deep boreholes for storage of nuclear waste, continuous coring may not be used. Geological data with enough accuracy can be acquired from a combination of spot coring, drill cuttings and wireline logging information. As discussed before in this report, it is probably an advantage to detect fractures and evaluate fracture density in-situ in the borehole.

If continuous coring is required, different systems for slim hole core drilling is available in the mining industry. In South Africa, boreholes down to depths of below 4 km are drilled continuously with wireline coring techniques. However, the cost is high and a change to oilfield equipment with spot coring is forseen. As a first step in the German super deep drilling (KTB), a pilot hole with continuous coring was drilled down to 4 km. One large disadvantage of slim holes is that they can only be used for geological investigations and not for storage of the waste. If necessary, one approach would be to drill one corehole or as many as required within an area for a deep storage repository for geological investigations. The Kola concept could also be used, but full-size coring all the way down to TD will be very expensive with both coring and reaming afterwards to achieve required borehole size.

Coring in crystalline rock with a full-size drilling rig is normally difficult. The coring results from Gravberg-1 are very discouraging with extremely low core recovery. 14 coring runs have been performed, 8 with no recovery and the others with a recovery of between 1-40%. 12 of the coring runs were performed with roller type bits. In an attempt to improve core recovery, two cores have also been drilled with diamond bits but both with no recovery. The reason for the bad coring result is probably a combination of a rough elliptic borehole and the wrong choice of coring equipment. Some improvement in the mechanical systems such as core catchers is also required. It is also noted that the worst results are obtained in fracture zones which are probably of greatest interest when investigating a rock mass for the possibility of storing nuclear waste.

The problem of coring is also depth-dependent. At deeper depths the rock stress is higher and the borehole will be more or less eliptic. It will therefore be difficult to avoid horizontal and vertical movements of the bit. It is also very difficult to control important drilling parameters such as weight on bit due to friction between the drillstring and the borehole. If drilling deviates boreholes, the problems discussed above will be even more pronounced.

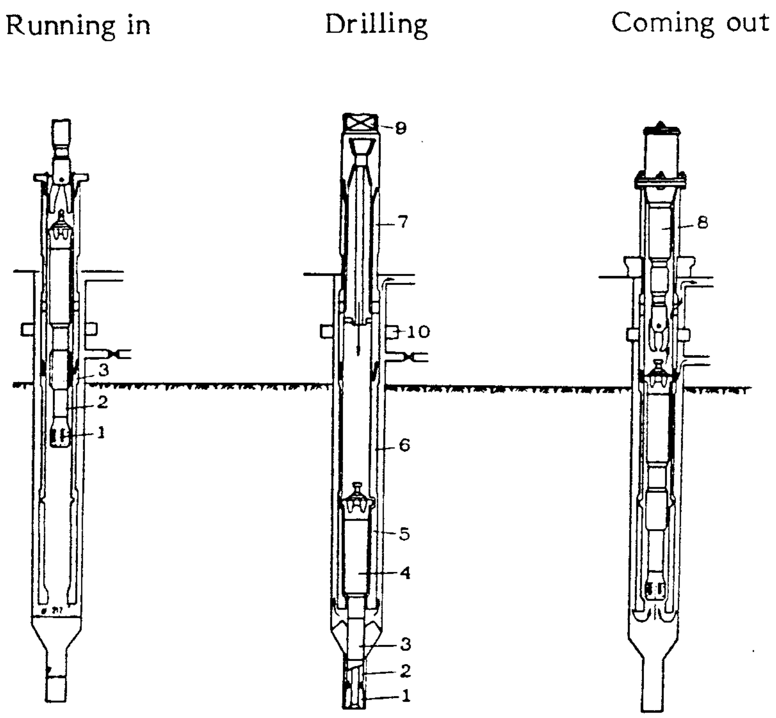
In comparison to Gravberg, very good coring results are being obtained at the scientific drilling at Cajon Pass some 3 km from the San Andreas fault in California. Instead of rotary drilling with rollercone bits, downhole motors and diamond bits were used. The present depth of the borehole is 3473 m, and 5% of the well has been cored with a recovery of 69%. Normally the coring has been performed with a positive displacement downhole motor with impregnated diamond bits. Typical RPM has been 300-350 and normal ROP has been 2 m/h. The great advantage of the downhole motor apart from the high RPM is also less movement of the bit and less space between the bit and the core catcher.

A first step to improve the system used at Cajon Pass would be to increase the RPM:s to about 600. A turbine motor is capable of this RPM but the torque capacity on turbines available is probably too low to make the bit rotate. Compared to Gravberg the rock stress situation seems to be more favourable at Cajon Pass. The stresses are probably lower due to the proximity of the fault.

The German KTB drilling constitutes a good reference for the possibility of performing continuous core drilling with mining equipment (wireline coring). After drilling to 4 000 m, the core recovery was nearly 100%. The borehole size is 6" (150 mm) which makes it possible to use all logging tools available on the market. A specially built $5\frac{1}{2}$" drillstring is used, giving a very small annulus between the borehole wall and the string. The advantage of this system is that there are only minor vibrations and the drillstring helps to stabilize the borehole wall. One part of the success is also a new type of silica-based mud that is extremely thixotropic. The future success with this coring system at greater depth is difficult to predict.

The Soviet Union is currently developing a coring system which is capable of retrieving cores and allows changing of the drillbit without pulling the drillstring (see Figure 4.3-2). The system is being field tested at the Krivoy Rog drillsite in the Ukraine. A tool is dropped inside the drillstring which consists basically of a coring bit, core barrel, reamer, downhole motor and a hook for an overshot. After the tool reaches bottom the bit and reamers are below the bottom of the string. By starting circulation the reamers expand and coring commences. The coring bit is a diamond-impregnated bit with a diameter of 152 mm. The reamers above it ream the hole to 217 mm which gives enough clearance for the drillstring. After a coring run is complete, reverse circulation is started and the reamers retract and the tool is pumped up the hole. So far 400 m of hole have been drilled using this technique with a core recovery rate of about 70%. Average drilling rates are 1-2 m/hr and daily rates have averaged around 25 m/day. A bit lasts for 10-40 m as do the reamers. A reduction gear assembly is used to increase the torque on the bit. On a couple of occasions the reamers have not retracted when reverse circulating, and it has been necessary to trip the drillstring.

A summary of the coring technique and coring records discussed is presented in Table 4.1.



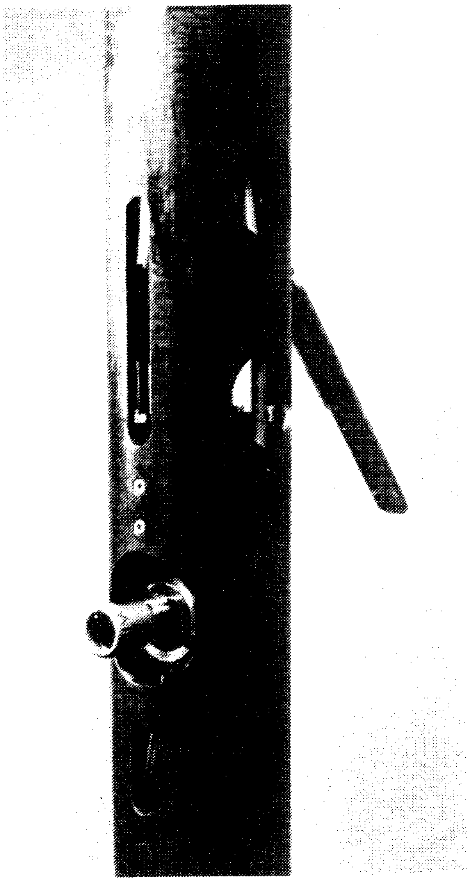
1. Bit
2. Core catcher
3. Retractable reamer
4. Motor unit
5. Motor housing
6. Drillstring
7. Perforated pipe
8. Hydraulic buffer
9. Kelly
10. BOP

Figure 4.3-2 New Russian coring system

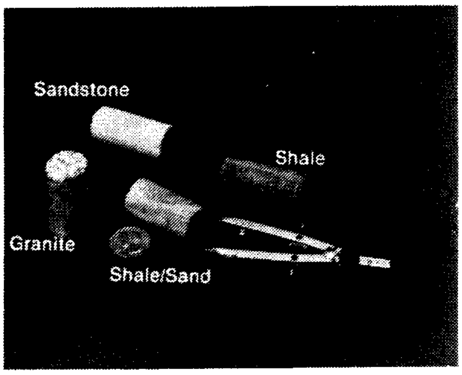
Table 4.1 Coring records from different boreholes

Name of the borehole	Depth m	Geology	Technique	Bit size	RPM	Coring m	Core recovery
Gravberg no 1	6600	Granite	Rotary drilling			104 m	8.6 m 8.3 %
			Roller cone core bits	8 $\frac{1}{2}$" core 3"	30-60		
Kola Super Deep	11 500	Mostly gneiss	Downhole motors	-		9325 m	3700 40.19
			Roller cone core bits				
Cajon Pass	3473	Gneiss	Downhole motors Diamond bits	8$\frac{1}{2}$" - 6" core 4"	300-350	174 m	120 m 69 %
KTB (Germany)	4000	Mostly gneiss	Rotary drilling Wireline coring Diamond 6:ts	6"	280	3600 m	99%
Krivoy Rog	3551	Metamorphic	Self-contained coring tool	152 mm	120-180	400 m	70%

In addition to full-size spot coring, a successful attempt with sidewall coring was carried out in Gravberg-1. Sidewall coring implies that small chunks of rock, 50x22 mm in size, are drilled out horizontally from the borehole wall, With the Gearhart tool used it is possible to take up to 32 cores on each run. One advantage of sidewall coring is that the choice of coring spots can be based on wireline logging results and therefore cover the most interesting parts of a borehole. The results from Gravberg also indicate that good core recovery is also possible in fractured parts of the rock column.



CORING TOOL



CORE SAMPLE

Core samples can be obtained from any consolidated formation such as sandstone, limestone, dolomite, and granite. Standard lab evaluations may then be performed on each sample.

TOOL SPECIFICATIONS

Diameter:	4-7/8"
Length:	17' (with gamma ray)
Positioning:	Gamma Ray
Capacity:	12 core samples
Borehole Diameter:	6-3/4" minimum 12-3/4" maximum*
Temperature:	300°F
Pressure:	20,000 psi
Cutting:	Horizontal

*can be equipped with arm extension for larger holes

GEARHART
THE GO COMPANY

Figure 4.3-3 Gearhart sidewall coring tool

Sidewall coring tools are fairly new on the market and the coring operation at Gravberg is still a world record depthwise in crystalline rock. Improvement of the tool is forseen in the future. In Germany, work is in progress on developing a tool that sidetracks from the original borehole and can thus take longer cores compared with the Gearhart tool, see Figure 4.3-4.

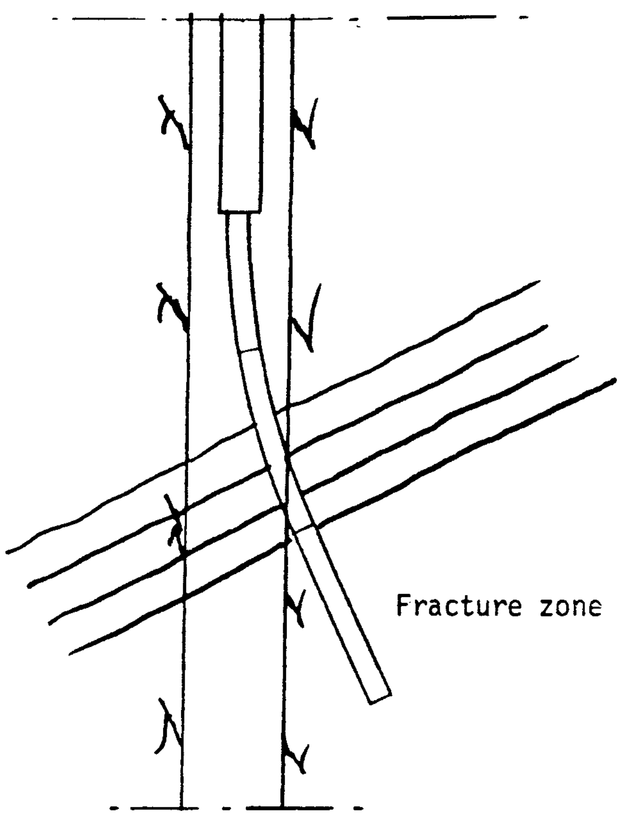


Figure 4.3-4 "Whipstock" coring tool

Conclusion

When drilling deep boreholes for storage of nuclear waste, the evaluation of the bedrock condition will mainly be based on drill cuttings and wireline logging information. Spot cores will be needed mainly for correlation of wireline logs, lithological studies and geochemical sampling. The best technique available on the market today seems to be a combination of downhole motors and diamond bits. One obvious disadvantage of traditional spot coring is that the cores have to be taken more or less blindly. Sidewall coring techniques would therefore seem to be advantageous. The coring spots can be based on other existing geological information and the result from coring in fracture zones is encouraging compared to full-size spot coring.

If a more detailed investigation of the rock mass is required, then an additional borehole with continuous coring could be drilled within the location for the deep borehole repository. The greater value of this borehole will probably be the possibility to correlate the response from different wireline logging data with the core. This will result in a better understanding and evaluation of logging data from the other deep boreholes within the repository. In addition, this pilot hole could also be used for detailed hydrogeological measurements and as a source for geochemical sampling and analysis.

Several ongoing scientific drilling projects in crystalline rock will contribute significant new knowledge about coring in crystalline rocks.

4.3.6 Hydraulic measurements

When discussing hydraulic measurements in deep boreholes one has to consider the drilling technique used. Drilling deep boreholes with oilfield equipment implies that a large volume of drilling fluids are circulated in the borehole. Owing to different conditions in the borehole the type of drilling fluid (mud) used will vary enormously from ordinary water to diesel oil. Various chemicals are added to the basic fluid for different purposes. All additives, for example bentonite, barite and cellulose polymers, will damage the formation and clog permeable fractures.

Since the pressure from the fluid column in the borehole is normally greater than the pore pressure in the rock formation fluid finds its way into permeable fissures and fractures.

Even when drilling with ordinary water the pressure in the borehole will be excessive due to an additional pump pressure and a possibly subhydrostatic pore pressure in the formation.

The main purposes of the drilling fluid are summarized below:

- To bring cuttings to the surface.

- To prevent fluids or gas from entering the borehole.

- To prevent the borehole from collapse and reduce the formation of breakouts.

- To reduce torque and drag while drilling.

- To cool the bit during drilling.

To meet these requirements, the mud system presented in Table 4.2 was used in Gravberg. Initially, the intention was to use only a fresh water drilling fluid. However, due to unforeseen drilling conditions several changes were necessary in order to be able to continue drilling.

Table 4.2 Drilling fluid system

Depth	System	Portion of hole
0-3932 m	Fresh water	Original hole
3932-4167 m	Fresh water & polymer	Original hole
4167-6081 m	Fresh water & bentonite	Original hole
4636-5799 m	Fresh water & bentonite	Sidetrack 1
5799-5938 m	Fresh water	Sidetrack 1
5938-6636 m	Fresh water & polymer	Sidetrack 1
5991-6673 m	Invert emulsion oil-base	Sidetrack 2
5850-6957 m	Invert emulsion oil-base	Sidetrack 3

As described in Section 2.4 fairly good hydraulic data were recovered to a depth at 3200 m when a freshwater drilling fluid was used. Below this depth only qualitative data are available due to formation damage by mud additives clogging permeable fractures.

The hydraulic conditions for a crystalline rock column are dependent on the fracture systems and their hydraulic properties. An evaluation of the hydraulic properties for a certain rock column should preferably be made in three steps.

A. Identification of fracture zones.

B. Determination of whether the fracture is permeable or not

C. Measurement of hydraulic properties.

The first step can be carried out with a fairly high degree of accuracy in deep boreholes as described in Section 2.3. Some results indicate that in-situ measurements are preferable compared to normal core mapping. There are results available in the literature about comparisons between core mapping and in-situ measurement when 90% of mapped fractures were drilling-induced. This will be pronounced at great depths when the core is released from very large stresses and with little control over different drilling parameters. The core handling at the surface also increases the amount of fracturing in the core.

Differentiation between open and closed fractures in deep boreholes is difficult. Hydraulic measurements are normally carried out over long intervals and measured hydraulic properties will always be average values for the tested interval. Analyses of results from different logging tools and especially sonic tube waves are sources of interest, and are probably a practical way of solving this problem in the future .

One important method for evaluating open and closed fractures is careful monitoring of fluid losses to the formation during drilling. This measurement is standard procedure during drilling for oil and gas. The measurements are carried out with floats in the mud tanks which monitor the fluid levels. If the mud system and mud logging operation during drilling is especially designed for these measurements, it will be possible to increase the accuracy compared with standard oil field drilling. For example, if the mud tanks are constructed higher with a smaller opening, the accuracy of the monitoring will increase.

In the German KTB drilling, the losses and gains of fluid are measured to the accuracy of 4 l/min and 200 l/day.

Hydraulic measurements in deep boreholes during drilling are normally performed as Leak-Off Tests (LOT) which give early information about the hydraulic properties of the rock and fracture gradients. LOT involve drilling fluid being injected into the borehole at fairly high pressure. During injection and a bleed off period, 5-15 minutes (longer if needed), the pressure is recorded at one-minute intervals. The bleed off period is used for an evaluation of the permeability of the tested interval. The tested interval is always the whole open section of the borehole below the casing shoe, see the figure overleaf. Continuous testing when drilling deeper will make it possible to evaluate different zones in the rock column. One disadvantage of this method is that the formation below the casing shoe will be exposed to a proportionately higher pressure compared to the deeper parts of the borehole. During drilling for oil and gas, LOT is used as a standard test to check the formation fracture gradient just below the casing shoe for safety reasons. Results from Gravberg show that LOT is an acceptable method for estimating permeability when drilling with fresh water. If the fluid system contains any additives these will be pressed into the formation, thus reducing the permeability measured.

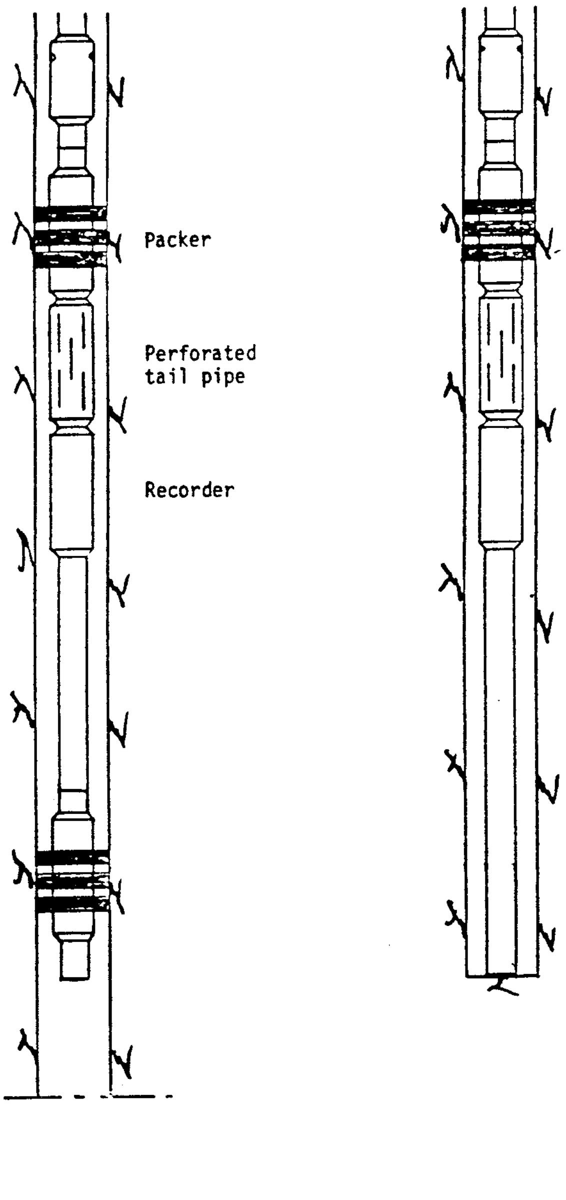
During Drillstem Tests (DST), formation fluid is extracted from the formation by underpressure in the drillstem between straddle packers or one packer and the bottom of the borehole. This test could be performed in both open hole or cased hole. Different methods of use are shown in Figure 4.3-5.

One major problem with DST in an open borehole is to find good packer seats. In a borehole such as Gravberg this is almost impossible. Any type of test needs to be preceded by caliper logging. In addition to the difficulty in finding packer seats, a DST is somewhat dangerous with respect to borehole collapse. In contrast to a LOT the pressure is reduced, thus increasing the risk of rock moving into the borehole and jamming the drillstring.

One method is to use the old DST technique with a rathole. This method will increase the probability of obtaining tight packers, see Figure 4.3-6. The cost will, however, be high because the intervals tested need to be drilled twice.

To achieve an appropriate differential pressure over the packer(s) when testing at deeper levels a water cushion with a tracer will be used inside the drillstring.

Normally the test string includes a fluid sampler which is activated at the end of the test. Depending on permeability, flow period and mud invasion it could be possible to recover a fairly non-contaminated fluid sample. It is also possible to go down inside the drillstring with a sampler on a wireline during the test.



A Straddle packer DST

B Single packer DST

Figure 4.3-5 Different types of drillstem tests, DST

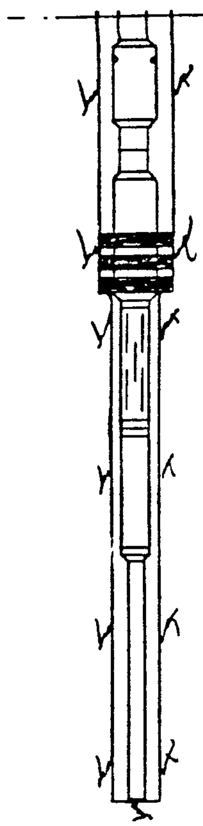


Figure 4.3-6 Single packer test with a rathole

When drilling an array of deep boreholes for storage of nuclear waste, a thorough investigation cannot be made of the hydraulic properties in all boreholes due to high costs and the long time needed to perform the test. One feasible way of overcoming these problems is to investigate certain boreholes very carefully with the methods described in this chapter and extrapolate measured data to other boreholes. The number of boreholes that need to be carefully investigated will depend on the site. One very important factor to consider is also the choice of mud system and additives. Fresh water should be used to as great a depth as possible.

Finally, it must be stated very clearly that measurements offering an accuracy similar to that displayed in shallow diamond coreholes cannot be achieved. From a hydrogeological viewpoint the rock mass needs to be treated more as an average problem in which the exact hydraulic property of each fracture zone is of less interest.

4.4 Crosshole investigations

4.4.1 Background

Crosshole investigation methods have considerable advantages over surface and single borehole methods when attempting to characterize the physical properties of a site under investigation. These advantages are, among others:

1) Distortions from near-surface effects are avoided.
2) Signal to noise ratio is generally greatly improved.
3) Resolution is increased.
4) Vertical features are more easily imaged.
5) Information is obtained covering a larger rock mass.

Points 2 and 3 are a direct result of avoiding the near-surface inhomogenities, such as the glacial till which is present throughout most of Sweden.

Crosshole techniques can employ both electromagnetic (EM) principles or seismic ones. In the case of EM studies the property of the rock which is of primary concern is its propensity to attenuate electromagnetic waves. For seismic studies, the properties which can in principle be measured are the rock's compressional and shear velocities and its effective attenuation of seismic waves. If data of sufficient quantity have been collected, then an image of some property of the rock between the boreholes can be produced using tomography methods. In the discussion that follows, only seismic methods will be considered since it is felt that these can provide most information about the rock in question and that it is crosshole seismics on which most research will be concentrated in the future.

4.4.2 Basic principles

Current field techniques employ seismic energy in the frequency range of 100 Hz to 10 kHz. At 1 kHz in crystalline rock the wavelength of the signal is about 6 m, giving a resolution in the order of 2 m. Methods for generating the signal range from explosive to piezo-electric to mechanical vibrations. The mechanical source is advantageous since it is capable of producing the important shear waves directly. Receiver systems are either of the hydrophone type, which measures the pressure of the fluid in the borehole, or of the geophone type, which are clamped to the borehole wall and measure the movement of the rock itself. The geophone receiver system is clearly the preferable one and allows the ambiguity of whether an arrival is a compressional wave or a shear wave to be resolved if a three-component geophone is used.

With the systems currently available, crosshole techniques have been used in boreholes separated by over 200 m with good results. In intact crystalline rock, crosshole imaging of up to 400 m should be possible with current technology. If an accurate tomographic image of the rock is desired, it is generally stated that travel time picks accurate to 0.1 msec are required, which is possible with the technology available. What is sometimes ignored is the accuracy required with regard to the spatial positions of the source and receiver. This implies that at boreholes which are 200 m apart, the positions of the sondes must be known to an accuracy of 1 m. In the case of deviated boreholes, this may not be possible. It is also desirable for interpretation purposes to have the boreholes in the same plane if they are deviated.

4.4.3 Results of crosshole surveys

Only two cases are known where crosshole techniques have been used in crystalline rock below 2000 m, these being the two hot dry rock projects at Fenton Hill (Los Alamos) and Rosemanowes Quarry (Camborne School of Mines). In both cases not enough data were collected to produce tomographic images. However, a number of interesting results were obtained concerning velocities and attenuation. At both sites the reservoir rock has been stimulated, thereby creating increased porosity and permeability. Comparisons of pre-stimulation surveys with post-stimulation ones show that velocities decreased over the stimulated zones. Furthermore, there was increased attenuation and decreases in the frequency content of the recorded signal. This is consistent with the effects of increased permeability and porosity in a rock mass. In addition, when the walls were pressurized after stimulation and surveys were conducted there were significant additional decreases in seismic velocities, frequency, content and amplitude of the signal in the reservoir rock, but only minor decreases in the rock which had not been stimulated. This observation implies that in the case of site characterization, conducting crosshole surveys in both the pressurized state and the unpressurized state may be a very effective method for locating permeable zones which extend away from the borehole.

4.4.4 Recommendations for crosshole surveys for site characterization

It is of primary importance to be able to generate shear waves and record them unambiguously in any crosshole survey. This is because the shear waves are more sensitive to changes in fracture density than compressional waves. In addition, information concerning the orientation of the fractures, and possibly the stress field, are more readily extracted from the shear waves than from the compressional waves. Repeatability of the source signal is also important when considering alternative studies of the rock mass. The receiver system should consist of a string of geophones (as many as possible) which can be clamped to the borehole wall and record data at a 50 kHz sampling rate. The receiver and source systems also have to be able to operate at depths of up to 6 km. With a system of the type described it should be possible to detect inhomogenities on the order of a few metres in intact granite between two boreholes spaced at 300-400 m intervals at 5 km depths.

5. ROCK MECHANICS CONSIDERATIONS

5.1 Introduction

In principle there are four processes that may contribute to the formation of a disturbed zone around a deep borehole for storage of nuclear waste:

- stress redistribution.
- damage by the drilling process.
- weathering.
- temperature loading.

Of these four processes, stress redistribution is by far the most important as it controls the size of the disturbed zone around the borehole. The redistribution of stresses in combination with orientation of the virgin stresses are also of the utmost importance to the borehole deviation. In the following section, the first part of the discussion is devoted to stress redistribution.

5.2 Stress redistribution

Since stress redistribution is regarded as being of the utmost importance for the borehole stability, and therefore the overall success of storage of nuclear waste in very deep boreholes, it will be discussed in more detail in this report under the following headings.

- borehole breakouts of intact rock.
- borehole stability of jointed rock.
- deviation of borehole.
- overall stability of inclined boreholes.
- borehole breakouts in the Gravberg well.

5.2.1 Borehole breakout of intact rock

When a borehole is drilled parallel to the direction of one of the principal stresses, the tangential stress in the rock around the wall is increased near the ends of a diameter parallel to the minimum stress in the rock mass. The rock in this region is subjected essentially to a biaxial compressive effective stress. If this stress is sufficiently great in relation to the strength of the rock, it fails by extensional splitting subparallel to the wall of the borehole. The splitting must occur quite close to the free surface of the borehole because the radial stress increases rapidly with radial distance into the rock mass, see Figure 5.2-1.

In a recent paper by Ewy et al. (1987) a demonstration is given of how progressive spalling of the borehole wall can lead to stable breakout shape. Based on existing knowledge of brittle failure of rock in uniaxial and triaxial compression, they define the following failure criteria for failure by splitting, shear and tensile failure:

Splitting: $\sigma_1 \geq + l\sigma_2$ for $\sigma_3 > T$ (5.1)

Shear: $\sigma_1 \geq k\,Q + n\sigma_3$ for $\sigma_3 > T$ (5.2)

Tension: $\sigma_3 \leq T$ (5.3)

where

σ_1 = major principal rock stress
σ_3 = minor principal rock stress
Q = compressive stress associated with onset of spalling at a free boundary
T = tensile strength (negative)
k,l,n = constants, $k \geq 1$

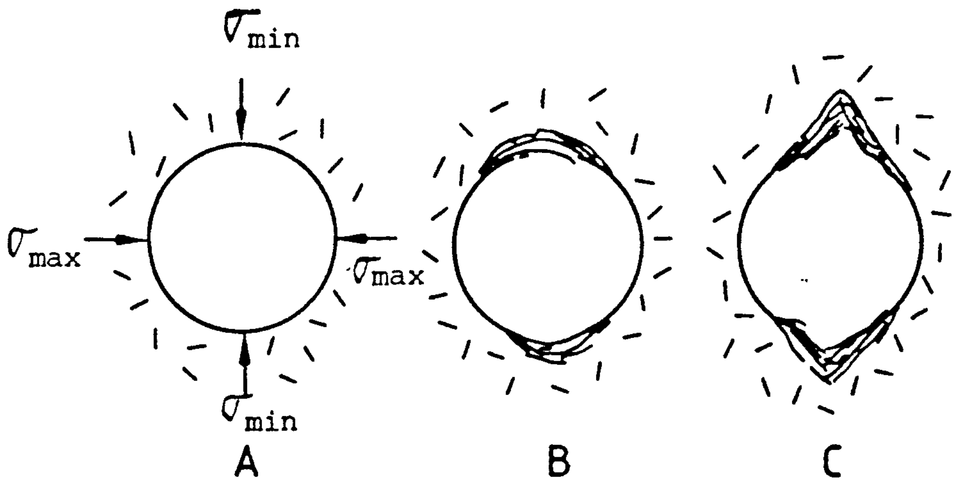
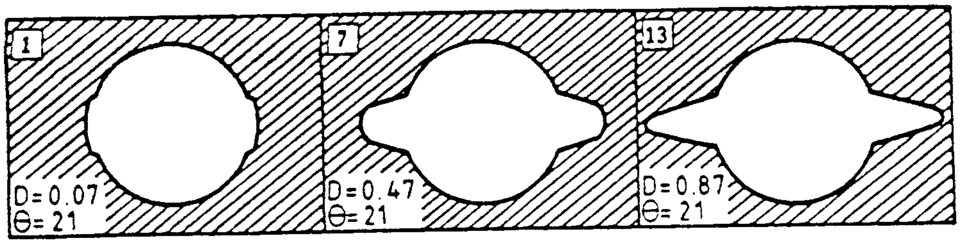


Figure 5.2-1 Hypothetical shape of borehole breakouts given the increasing state of stress A to C.

The splitting and shear failure criteria correspond to Mohr-Coulomb criteria expressed in terms of principal stresses. The constants l and n are related directly to the angles of internal friction.

Comparison of the stress around the borehole with the shear failure criterion eq. (5.2) reveals that only a thin skin of rock close to the boundary meets the failure criterion. Because these failure zones correspond exactly to those regions of the boundary subject to splitting failure, it is assumed that the more unstable splitting behaviour will be the mode of failure according to Ewy et al. (op.cit.). The tensile failure criterion given by eq. (5.3) is not met anywhere around the boundary at any deep borehole in the Precambrian rocks of Sweden and can be overlooked in this study.

To model successive spalling, boundary element analysis (BEM) was applied for systematically changing the cross-section of the borehole as a result of each episode. The final breakout shape of the borehole is allowed to evolve by a succession of such episodes until the tangential stress is everywhere less than the strength of the rock, see Figure 5.2-2.



D = Breakout Depth/Original Radius θ = Breakout Angle

[1] = Cycle Number

Figure 5.2-2 Cross-section through a borehole that illustrates the steps by which a stable breakout cross-section is obtained. After Ewy et al.(1987).

Using the methodology described above, stable wellbore breakout shapes have been determined for a range of ratios of rock stresses, relative to the strength of the rock, and the following general conclusions have been obtained:

(i) Breakout cross-section redistributes the stresses in the rock mass, so that the stresses become everywhere less than the splitting strength, shear strength and tensile strength and the final cross-section becomes stable.

(ii) The resulting cross-section depends upon the stress path; breakouts are smaller if they result from a gradual increase in stress than if an excavation is created instantly in a rock subjected to stresses.

(iii) Formation of tensile and/or shear fractures does not appear to have a significant effect on the spalling behaviour or the final shape.

(iv) Spalling may be a time-dependent phenomenon.

The stimulated stable breakout cross-sections for different values of field stress and biaxial strength have been applied to the breakouts in the Siljan deep borehole. The assumed field stresses are taken from eqs. (2.3, 2.4) and the compressive strength associated with the onset of spalling at free boundary, Q = 140 MPa, is taken from the study by Hakami (1986). The data are presented in Table 5.1 and shown graphically in Figure 5.2-3. Here we notice a very good correlation between the simulated zone of non-breakout and the onset of breakout in the Siljan borehole at about 1500 m depth, corresponding to a stress ratio of S_H/S_h = 1.53 and a normalized strength of $(S_H - S_h)/Q$ = 0.11.

Table 5.1 Field stress and strength of rocks at Siljan deep borehole

Depth	Maximum horizontal stress	Minimum horizontal stress	Ratio	Compressive strength	Ratio
	S_H	S_h	$\frac{SH}{Sh}$	Q	$\frac{SH-Sh}{Q}$
[m]	[MPa]	[MPa]		[MPa]	
1500	46.4	30.4	1.53	140	0.11
2000	59.7	39.1	1.52	140	0.15
3000	86.1	56.7	1.52	140	0.21
4000	112.6	74.2	1.52	140	0.27
5000	139.0	91.8	1.51	140	0.34
6000	165.5	109.3	1.51	140	0.40

A zone of no breakout is shown in the top left hand corner of the diagram in Figure 5.2-3, where tangential stresses and the application of spalling failure are not enough to give breakouts. The larger breakouts result from applying the stress first to the rock mass and then excavating the borehole. This represents what happens when a borehole is drilled.

As shown in Figure 5.2-3, a larger stress ratio for one and the same normalized strength will cause less breakout and more stable borehole walls. This might seem confusing but is due to the fact that the radial stress acting at the wall is much more active and gives a strong support and prevents spalling.

The result of the application of simulated stable breakout cross-sections for different values of field stress and the experimental compressive strength for Siljan granite give large confidence in the assumption of the field stress as presented in Chapter 2. Furthermore, this analysis provides us with the basic understanding of borehole breakouts in intact rocks.

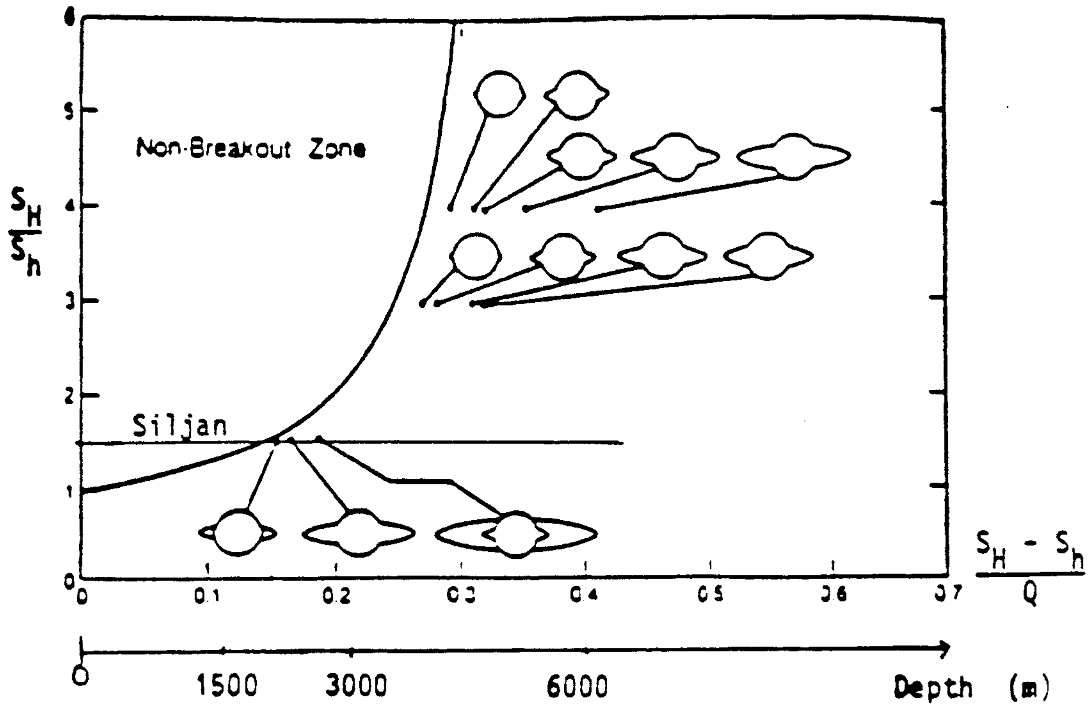


Figure 5.2-3 Simulated stable breakouts for different field stress and strength. Stable breakout cross-sections for Siljan deep borehole with the assumed stress field and compressive strength Q = 140 MPa. Notice the onset of breakouts at a depth of 1500 m and the severe breakouts for depths below 3000 m for zero mud pressure.

5.2.2 Borehole stability in jointed rock

The distinct element method (Lemos et al., 1985) was used in an investigation of the effects of joint patterns, borehole size and state of stress on the stability of deep boreholes and with special emphasize on the situation in Gravberg. The objective of this study was to provide insight into the processes leading to borehole breakouts and whether the breakouts are enhanced by the existence of joints and joint sets.

A special version of the MUDEC programme package for distinct element analysis has been used. The rock mass is treated as an assembly of elastic rock blocks, separated by deformable joints with non-linear deformation properties and Barton's strength characteristics, (ITASCA, 1986). The special version of MUDEC, applied to the stability analysis of the Gravberg well, allows for a study of the effects of a fluid pressure inside the borehole.

A rock mass model with dimensions of 2x2 m with two sets of joints and a borehole in the centre was analysed by means of MUDEC 1.02 N. The estimated state of horizontal stress was applied as boundary conditions and the stresses and displacements were analysed for three different mudweights at three depths along the borehole. Stresses were applied in accordance with the situation at Gravberg, and the direction of S_H is taken to be NW-SE. Borehole depth, borehole diameter and mud weights for the numerical modelling are shown in Figure 5.2-4.

The material properties used in the modelling are the following:

Joints

Joint distance	0,3	m
Joint set orientation	N-S, E-W	
Normal stiffness	100	GPa/m
Shear stiffness	100	GPa/m
Cohesion	10	MPa
Dilation coefficient	0,1	(tangent)
Tensile strength	0	MPa
Friction coefficient	0.8	
Joint roughness coefficient, JRC	8.0	
Joint compressive strength, JCS	60	MPa
Basic friction angle, f_r	26^o	

Intact rock

Density	2.7	t/m^3
Poisson's ratio	0.27	
Young's modulus	10	GPa

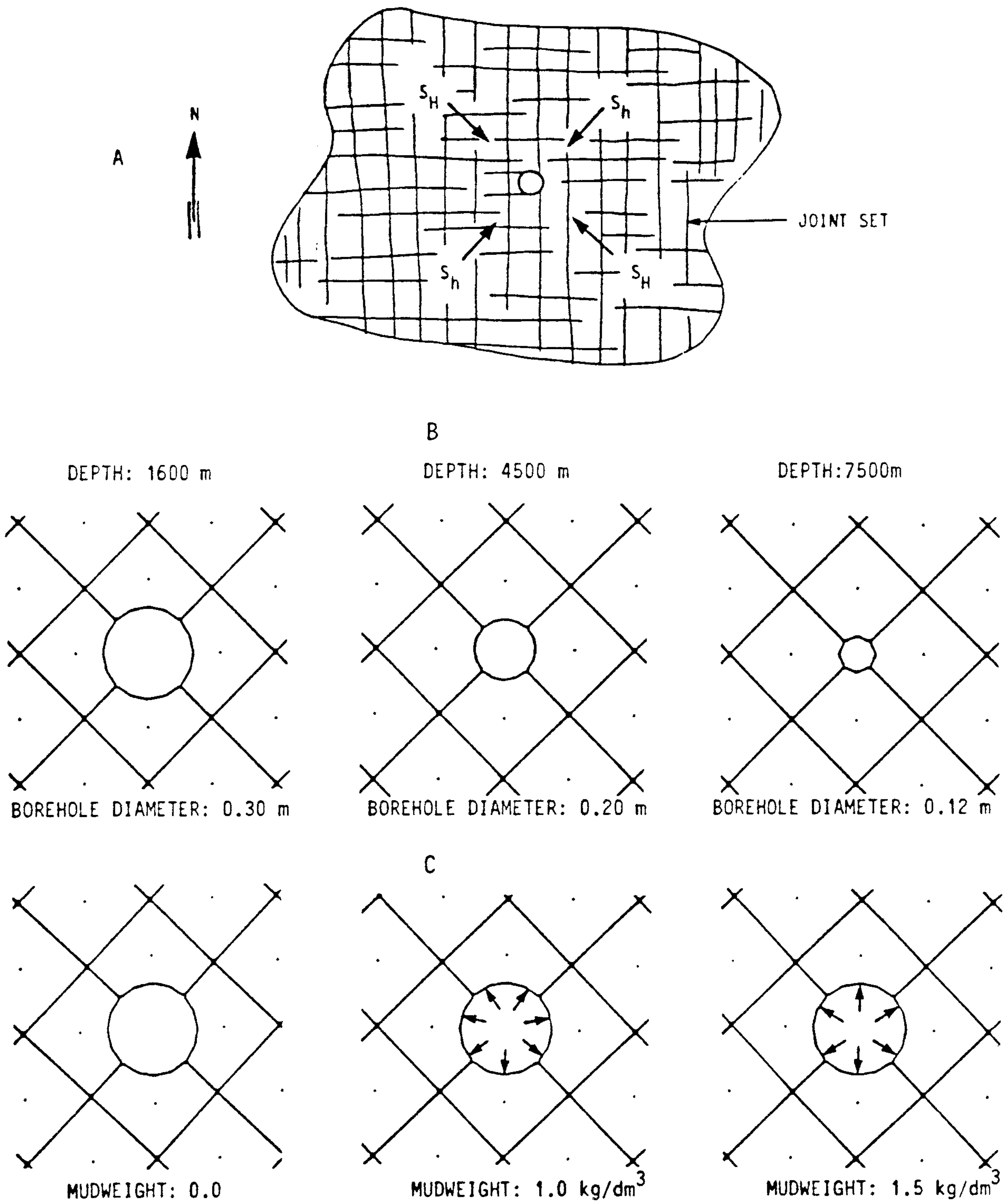


Figure 5.2-4 Numerical modelling of borehole stability in jointed rock.
A, orientation of sets of joints and horizontal stresses.
B, borehole depth and borehole diameter.
C, mud weights applied in the borehole.

The displacements and stresses around the borehole were monitored in 17 so-called history points during the ficticious time steps of the numerical analysis of the problem.

The modelling is conducted in four steps. Initially the stresses are applied to the 2x2 m block and the model is allowed to reach equilibrium. Then the borehole excavation is performed at zero mud weight and the model is again allowed to reach the new state of equilibrium. In the next step mud is introduced into the borehole and the joints and the mud weight times the depth will create a borehole pressure that counteracts the stresses and displacements by the virgin stresses. After a new equilibrium is obtained, the mud weight is increased again and the final result is obtained.

The results of the MUDEC modelling of borehole stability in jointed rock masses are summarized as follows:

- The borehole wall remains stable for any studied depth, borehole radius and mud weight.
- Noticeable displacements in the surrounding rock mass are found two borehole diameters (= 60 cm) away from the wall at a depth of 1600 m and three borehole radii (40-60 cm) away at 4500 m and 7500 m depth for borehole with mud pressure.
- Maximum tangential stress at the periphery of the borehole is 2/3 to 1/2 of the calculated stresses, assuming homogeneous, isotropic and elastic material.
- The mud weight is found to have only a minor influence on the stability and state of stress around the borehole in the case of large depth and small diameter hole.

The modelling results give a clear indication about the extension of the displacements in joints around a borehole in jointed rock masses. The results can be applied to the estimate of the mud invasion during drilling and the sealing above and below the deployment zone.

The smaller breakout amplitudes at great depth could also be due to a more isotropic state of stress. At greater depth S_h is closer to S_H compared to shallower depth.

5.2.3 Deviation of borehole

The general problem of borehole deviation is complicated. A large number of variables affect the final orientation of a borehole. The main factors influencing the borehole orientation have been summarized by Sinkala (1986) as being the following:

1. Rock-dependent; type of rock, rock anisotropy (foliation, bedding, fractures, faults), rock hardness, rock stresses, etc.

2. Operation and hole design-dependent; hole diameter, hole angle, thrust, angular velocity of bit, etc.

3. Machine-dependent; type of drill string, etc.

Referring to point 1, information on the rock mass properties at depth in the Siljan area is very sparse. Fractures and fracture systems have been mapped at the surface but an extrapolation of that information down to a depth of 5 km is a difficult task. However, the existence of planar structures, such as a dominant joint system of foliation, controlling the borehole deviation cannot be excluded. Field experience and theoretical analysis have shown (Sinkala, op.cit.) that the direction of deviation in this case depends on the angle between the planes of discontinuities and the borehole axis, Figure 5.2-5. A vertical borehole encountering planes of discontinuities dipping at a lower angle than approximately 45° will force the borehole down-dip.

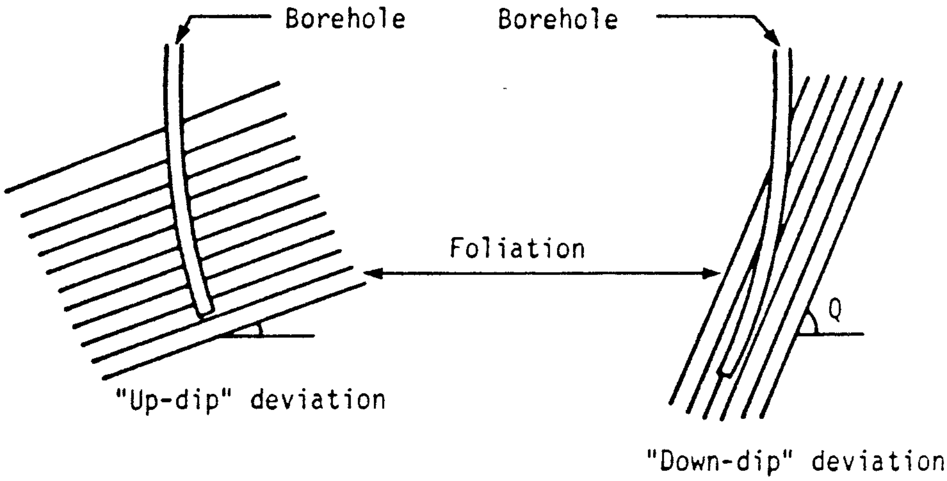


Figure 5.2-5 Observed borehole deviation phenomena in layered or foliated rocks. After Sinkala, 1986.

Points 2 and 3 above concern the actual drilling technique and will not be given further consideration here. This does not mean, however, that these factors are excluded as possible explanations for the deviation.

An analysis by Mastin (1988) gives strong support to the observed situation at Gravberg, that the borehole breakout is most likely to be oriented parallel to the direction of S_h. Furthermore, the borehole stability is demonstrated to increase as the borehole deviation increases, and for the most likely strike slip faulting condition prevailing at Gravberg the most stable borehole conditions are obtained for a horizontal borehole.

5.2.4 Overall stability of inclined boreholes

In order to evaluate the stability of the borehole, the stress distribution in the vicinity of the borehole must be compared with a failure criteria. Analyses are performed by comparing the elastic stress distribution around the borehole with the strength of the rock, e.g. according to Mohr-Coulomb failure criterion. The fact that a borehole deviates means that the virgin principal stresses have to be transformed to local stresses for the vicinity of the borehole wall. In addition, we have to take into consideration the mud density, at which the stress distribution at the borehole wall matches the failure criterion. If the mud weight is too low, compressional or shear failure will appear at the borehole wall. On the other hand, if the mud weight is too high, the borehole wall will fail in tension and a fracture will form and propagate perpendicular to the least virgin stress, see Figure 5.2-6.

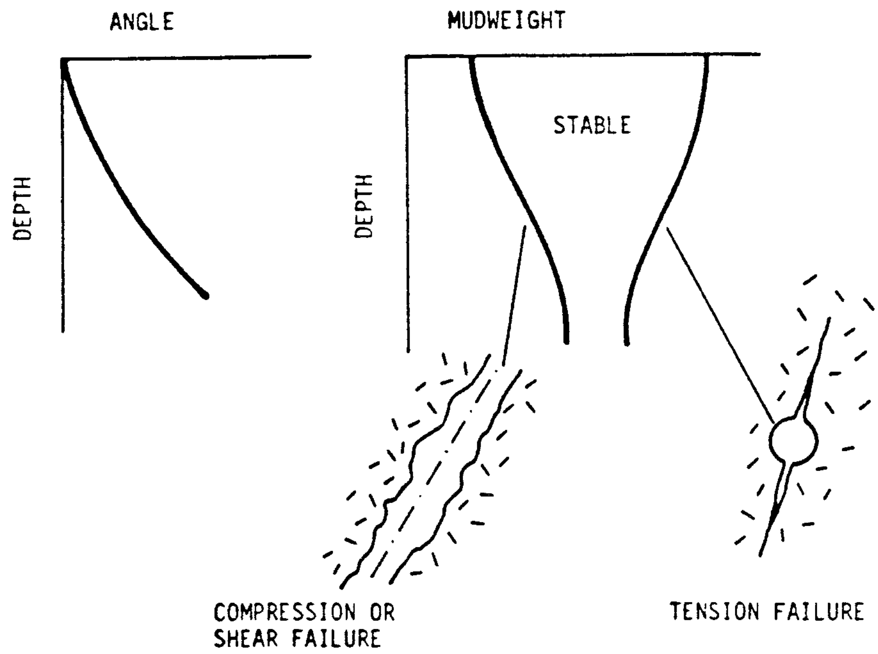
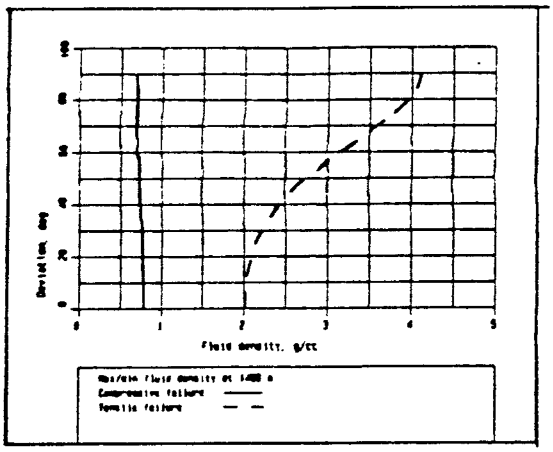


Figure 5.2-6 Failure modes as a function of mud weight for a deviated borehole. A low mud weight causes compressional or shear failure and a high mudweight will generate a hydrofracture.

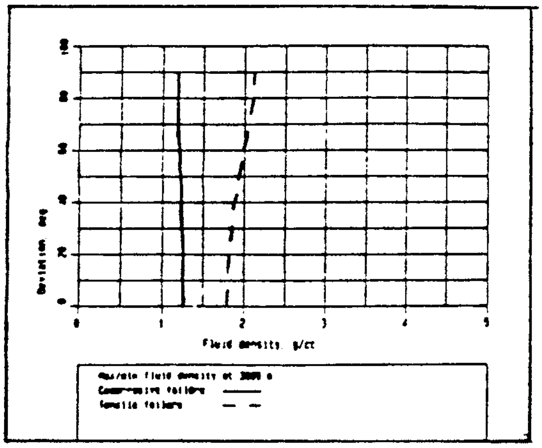
The stability of the Gravberg deep well has been evaluated for five different depths and for all hole inclinations from zero to 90 degrees and the results are presented in Figure 5.2-7. Stress magnitudes, orientations and rock strength data used in the analysis are those given by the situation at Gravberg. Rock strength data used in the analysis are those presented in Chapter 2.

The solid line to the left on the diagrams in Figure 5.2-7 shows the minimum allowable mud weight while drilling. Drilling with lower mud weight will cause the stresses in the borehole wall to exceed the compressive strength of the rock, resulting in breakouts. The dotted curve to the right shows the maximum calculated mud weight. Drilling with higher mud weights will cause the bottom hole pressure to exceed the minimum tangential stress in the borehole wall in a plane perpendicular to the borehole axis, resulting in a tensile fracture and loss of the drilling fluid out to the rock formation. The area between the solid line and the dotted line on each diagram represents the span for the mud weights to obtain a stable borehole.

The large difference in the shape of the curves and the width of the safe span between depths of 1489 m and 3689 m is mainly caused by differences in azimuth, resulting in a different orientation of the borehole axis with respect to the virgin stress field. The conditions for tensile failure change rapidly as a function of borehole inclination. This is due to the assumption that a tensile fracture (hydrofracture) will form parallel to the borehole axis, thus controlled by the minimum stress in a plane perpendicular to the borehole.



a)



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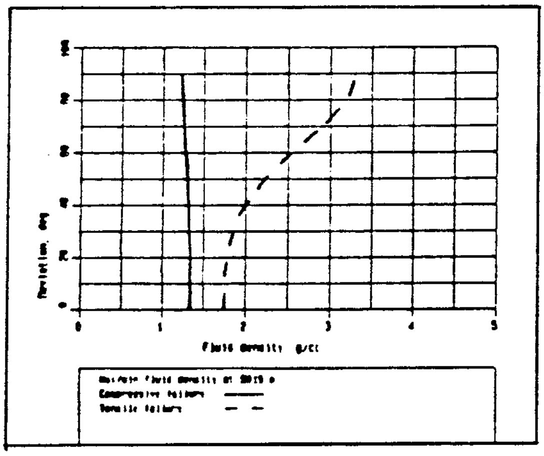

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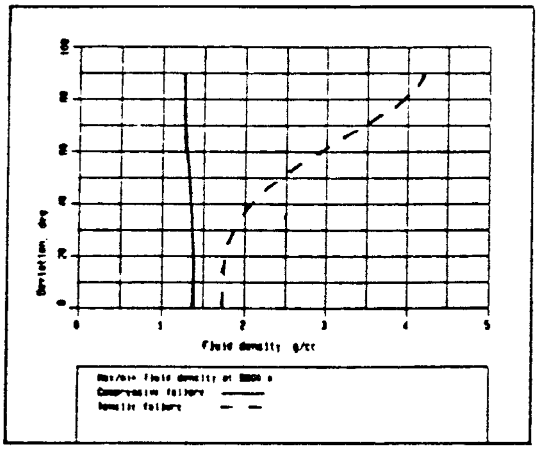
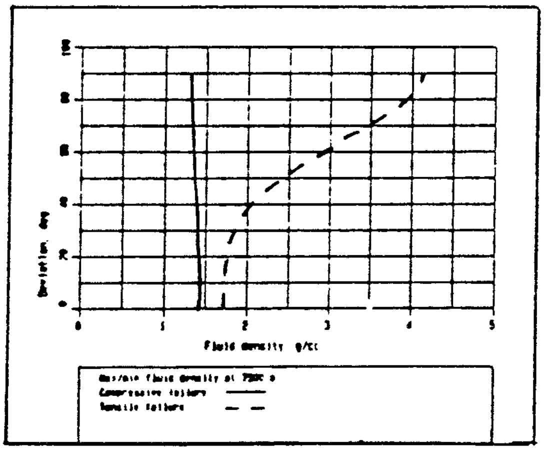


d)



e)

Figure 5.2-7 Results of stability analysis at five different depths in the Gravberg well. The fluid densities (mud weights) for compressive failure (solid lines) and for tensile failure (dotted lines) are plotted for all borehole inclinations from vertical at 0^{o} to horizontal at 90^{o}, a) 1489 m, b) 3689 m, c) 5019 m, d) 5900 m and e) 7500 m.

5.2.5 Borehole breakouts in the Gravberg well

The borehole at Gravberg has been measured by a four-armed dip meter caliper logging tool, which makes it possible to estimate the breakout, the deviation and the azimuth for the borehole as a continuous function of depth.

Breakouts started to appear at a depth of about 1500 m and remained more or less constant to a depth of 4800 m. Severe breakouts appear at a depth of about 5000 m. Cementing and re-entering of the borehole at a depth of 4500 m showed that breakouts appeared immediately when the drilling started again.

The data from the caliper log have been interpreted by the rock stress research group at the Department of Geophysics at the Technical University of Karlsruhe. The section from 1250 m to 3935 m was analyzed by Clauss (1987). Table 5.2 shows a summary of the breakout interpretation as calculated direction of maximum horizontal stress and the error for the calculated direction.

The average direction of maximum horizontal stress is found to be S_H = N109°E with a standard deviation of S_n = 16°.

Table 5.2 Calculated direction of maximum horizontal stress from breakouts in the Gravberg well. After Claus, 1987.

Depth (m)	Direction of maximum horizontal stress	Error
1250-1500	N115°E	< 15°
1500-1800	N146°E	± 10°
1800-2000	N123°E	± 20°
2000-2500	N105°E	± 20°
2500-2750	N95°E	± 20°
2750-3000	N105°E	< 20°
3000-3125	N100°E	< 15°
3125-3450	N87°E	< 15°
3450-3600	N95°E	< 15°
3600-3950	N120°E	<15°

There is a tendency for the shift in direction of horizontal stress to be related to the appearance of interpreted fracture zones. The fact that these zones show no breakouts is a strong indication that the magnitude of stress is less in the fracture zones than in the surrounding rock. Merging the information of no breakouts and change in direction of maximum horizontal stress indicates that the borehole is penetrating a rock mass composed of large blocks of several hundred metres in size. This can be interpreted to mean that major discontinuities or fracture zones are bounding the large blocks and the stress state is non-uniform, reoriented and probably less at the vicinity of the fracture zones. Breakout observations with consistent orientations inside each block, seem to indicate a fairly homogeneous state of stress. The breakout directions could also be affected by different structural discontinuities within different blocks.

There is a positive correlation between the mud weight and the breakout for the Gravberg deep well. Down to a depth of 3900 m water was used as a drilling fluid. From there on, drilling mud has been used and the mud weight versus depth is shown in Figure 5.2-8. A maximum mud weight of 1.48 kg/dm^3 was used for drilling at about 5800 m. The same diagram in Figure 5.2-8 shows an interpretation of the average breakout versus depth. Although the method is crude, it demonstrates the general trend of reduced breakout with increasing mud weight.

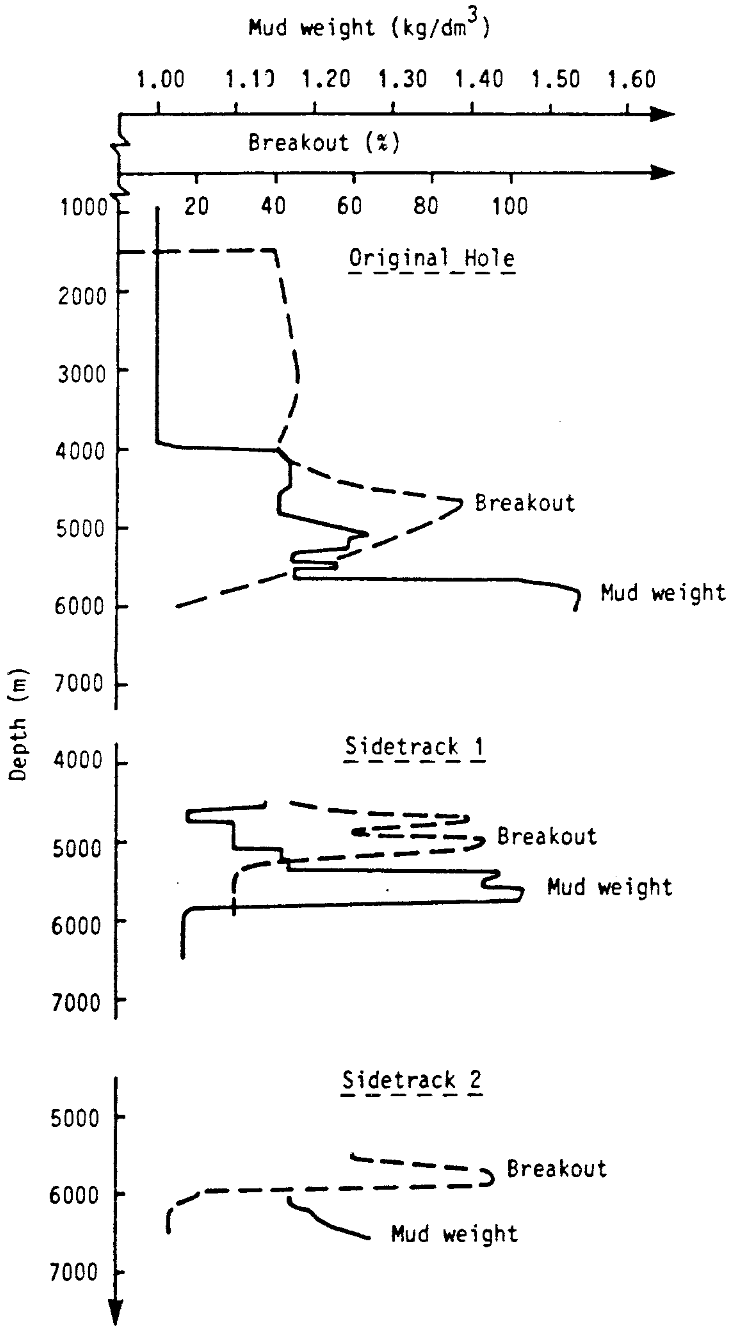


Figure 5.2-8 The relationship between average breakout and mud weight versus depth for the Gravberg-1 deep well.

Experience from rock stress measurements in granitic rocks in Sweden and elsewheres indicate an inhomogeneous state of stress versus depth. Major discontinuities like faults and shear zones cause stress jumps of the order of tens of megapascals down to the depths of 500 m in the Earth's crust, Bjarnason and Stephansson (1987). There is reason to believe that this type of discontinuous state of stress might exist at deeper levels, although there would be a tendency to eliminate any stress discontinuities as we reach the transition between the upper and lower crust, at a depth of about 15 km. Interpretation of borehole breakouts at Gravberg, analyzed by the group at Karlsruhe (Claus, 1987), seems to indicate a minor shift in the orientation at each side of the recorded fracture zones obtained from the geophysical logs. This is another indication of stress discontinuities at fracture zones.

If this model of the rock stress changes versus depth is valid, one would expect two different types of failure of the borehole wall: one for intact rock (A), and one for the fracture zones (B) as shown in Figure 5.2-9.

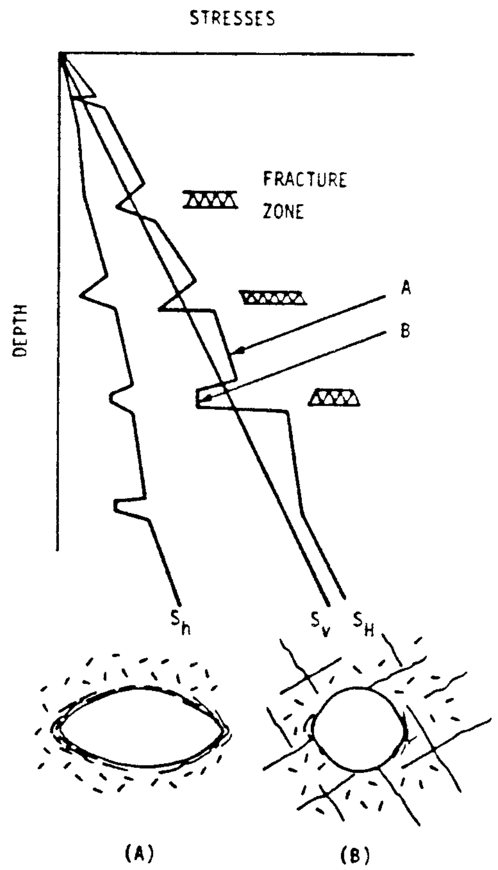


Figure 5.2-9 Suggested models for breakouts at the Gravberg-1 well. Stresses are assumed to be low across the fracture zones. Spalling is the most common mechanism of failure for intact rocks (A) and numerical modelling has demonstrated lower stresses and less displacements in the vicinity of a deep borehole in jointed rock mass.

5.3 Damage caused by the drilling process

The detoriation of the borehole wall due to drilling is caused by

- damage from the drill bit.
- damage from vibrations of the drillstring.
- damage from fluid flow.

Laboratory studies of borehole disturbance are being conducted at several laboratories around the world, Kelsall et al. (1982). Cylindrical samples of granitic rocks placed in a simulator at Terra Tek Inc., and subjected to confining pressure of 138 MPa and wellbore pressure of 34.5 MPa show that the thickness of the disturbed zone is less than three per cent of the borehole radius, and the disturbance effects are intergranular. Disturbed zone studies conducted at the University of Arizona show that the average damage zone is 2 mm and less for a 12.7 mm diameter hole drilled with percussion or diamond drill in granitic rocks. From existing laboratory tests it is concluded that the disturbance associated with drilling of the borehole is likely to be relatively minor.

Damage to the borehole by the drilling process can also be due to the drill string vibration and sideways push of the drill string. This means that the borehole is no longer circular in cross-section but strongly elongated, where the breakouts form two opposite and continuous slots. The drillstring will tend to lean into one of the slots. This will result in an eccentricity in rotation and a sideways push of the drill bit against the opposite wall at the bottom of the borehole.

5.4 Weathering

In the context of storage of radioactive waste in deep boreholes, weathering may be significant only in clay-bearing rocks. Due to unloading, swelling or sloughing might appear, but these processes are only likely to occur in clay-rich rocks like shales.

A special case concerns the swelling of clay fillings in joints or swelling of clay materials in hydrothermally altered veins and shear zones. If expandable clay fillings are present in the rock mass, this could lead to loosening of blocks from the borehole wall. Therefore, rock mass characterization prior to any deposition has to prove that swelling clay materials are not present in the rock mass or any other loose joint filling material that might be washed out or dissolved during the drilling operation.

In conclusion, weathering, swelling clays and groundwater interaction with the rock mass are relatively easy to detect and are considered to be of minor significance for this study.

5.5 Temperature loading

Calculations of temperature profiles around a deep borehole repository indicate a maximum temperature increase of about 100°C from the waste. Together with the virgin temperature of about 100°C at a depth of 6 km, the ambient temperature is estimated to be about -150°C.

In general, the brittle failure of rock is relatively insensitive to changes in this modest temperature in contrast to its behaviour in the ductile regime, where there are quite significant effects. However, as the confining pressure $\sigma_2 = \sigma_3$ increases, the temperature becomes more important. The true effect of the influence of temperature and confining pressure for charcoal granite is depicted in Figure 5.5-1.

Here we notice a moderate decrease in strength, $\sigma_1 - \sigma_3$, as the temperature increases for both low and intermediate confining pressures. Therefore, the effect on temperature is not considered to be a major problem for the borehole stability unless the borehole is affected by severe virgin stresses when a minor increase in temperature would trigger spalling.

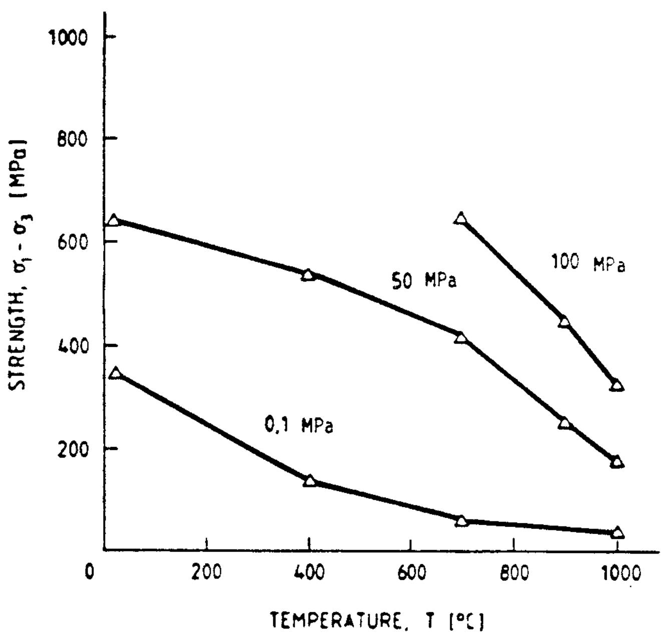
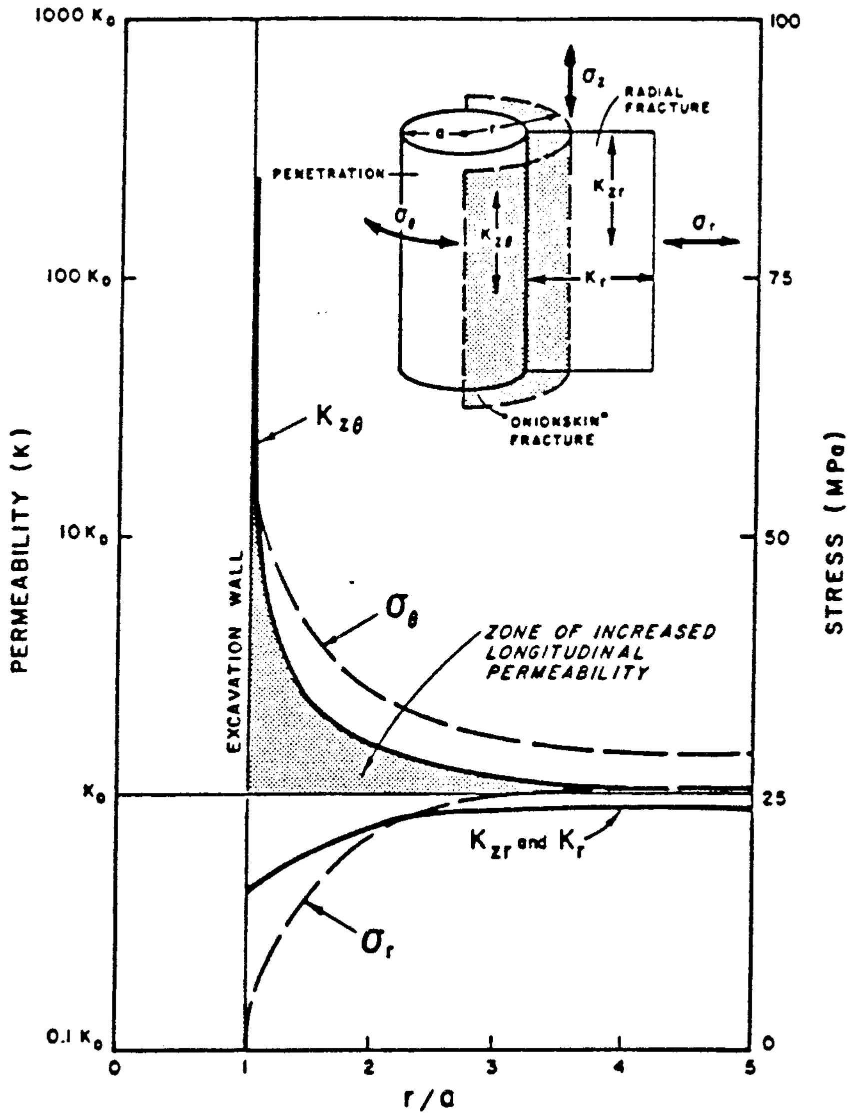


Figure 5.5-1 Strength versus temperature for charcoal granite at various confining pressures. Modified after Heuze, 1983.

The rate of heat load is known to have an effect on the stability of boreholes. However, if we assume an average rate of temperature increase of 70°C/10 years for a storage capacity of 80 W/m borehole, there is a minor risk of severe borehole breakouts and/or decrepitation. Any displacement in the vicinity of the boreholes is likely to appear along pre-existing joints. This was studied in a finite element model of a storage hole, where the swelling pressure of compacted bentonite and a temperature increase of 80°C was modelled, Stephansson et al. (1981). In the case of an isotropic state of virgin stress of 20 MPa, the normal displacement in the joints next to the opening was found to be 0.5 to 0.1 mm, and the maximum displacement of a triangular rock fragment next to the borehole was 2 mm. This gives an indication of the displacements one might expect from the thermal loading of a borehole in stanitic rocks.

5.5 Hydromechanical effects

Any breakout (circumferential or dog-eared) or decrepitation of the borehole wall will change the permeability in the vicinity of the borehole. The increase in permeability at the breakouts will enhance the convection along the axis of the borehole due to thermal convection. Kelsall et al. (1982) studied the stress distribution and the permeability change in the vicinity of a borehole in elastic rocks subjected to isotropic stresses in the plane perpendicular to the axis of the borehole. A cubic law permeability versus stress relation was applied. The stress distribution and the zone of increased permeability are shown in Figure 5.6-1.



K_{zr} = PERMEABILITY OF RADIAL FRACTURES IN THE VERTICAL DIRECTION
$K_{z\theta}$ = PERMEABILITY OF ONIONSKIN FRACTURES IN THE VERTICAL DIRECTION
K_z = TOTAL PERMEABILITY IN THE VERTICAL DIRECTION: $K_{zr} + K_{z\theta}$
K_r = PERMEABILITY IN THE RADIAL DIRECTION
K_0 = PERMEABILITY IN THE UNDISTURBED ROCK (ISOTROPIC)

Figure 5.6-1 Predicted disturbed zone permeability based on elastic stress analysis and cubic law permeability-stress relationship for fractured basalt - isotropic initial stress conditions. After Kelsall et al. (1982).

Due to the low radial stress, σ_r at the borehole wall "onion skin" fractures might form and the permeability in the vertical direction, K_z, will increase with respect to the initial permeability K_0 of the undisturbed rock. Any existing radial fracture will show a permeability of less than K_0 due to the closure of the fracture caused by the increase in tangential stress, σ_t. The total permeability in the vertical direction becomes $K_{zr} + K_{z0}$, cf. Figure 5.6-1. In the case of elasto-plastic rocks, both tangential and radial stresses are reduced in the plastic zone close to the excavation, which lead to increased permeability both in the radial and axial direction and the thickness of the disturbed zone will increase. There will always be a maximum thickness of the disturbed zone around a borehole if we assume that the loose material is left in the borehole. If the detoriated zone becomes 0.6 times the radius of the original borehole, Figure 5.6-2. Outside the new wall of the borehole, there might also be a zone of increased longitutional permeability, Figure 5.6-2. This stage of breakout and detoriation corresponds to borehole collapse. If the borehole is filled with a homogeneous clay plug in and above the deployment zone, and the swelling pressure is of the order of 1-5 MPa, the onion skin fractures will remain the major pathway for the groundwater due to convection.

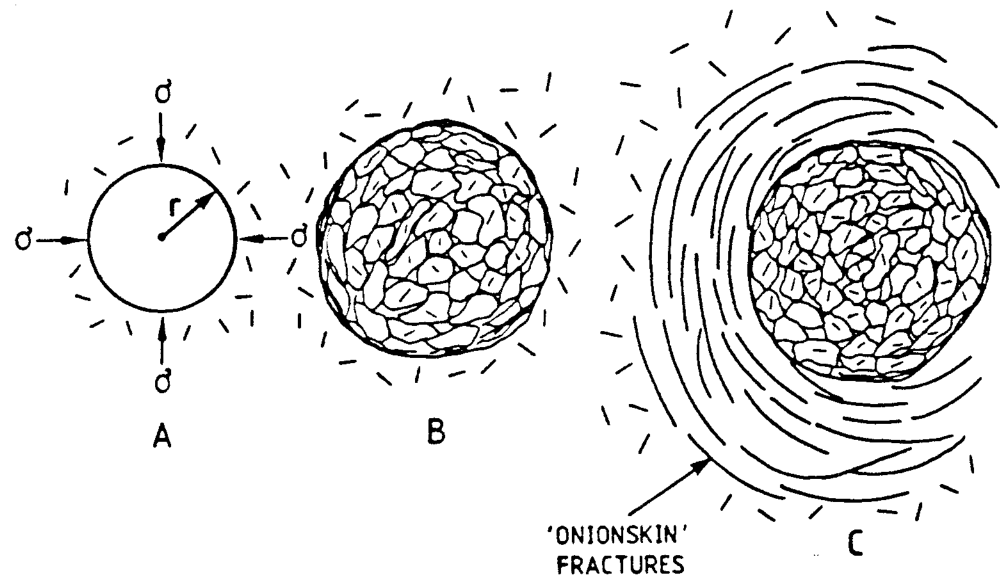


Figure 5.6-2 The development of collapsed borehole in an isotropic stress field. A, borehole in intact rock mass. B, filling of the borehole with detoriated material increases the ratios 0.6 r. C, collapse of the borehole and formation of onion skin fractures.

If the initial stress conditions are anisotropic, the more conventional type of breakouts will form. There seems to be a general trend that the more anisotropic the stress state, the more dog-eared the breakouts become. However, the shape of the breakouts is also a function of the strength properties of the rock mass. Figure 5.6-3 shows a schematic representation of different types of breakouts as a function of the applied stress ratio. For large stress ratio between the two horizontal stresses, tensile fractures are likely to appear in the direction of the maximum principal stress, cf. Figure 5.6-3 D. This fracture will have a limited extension of less than a borehole radius. The major pathway for convecting groundwater is likely to be the area at the breakout.

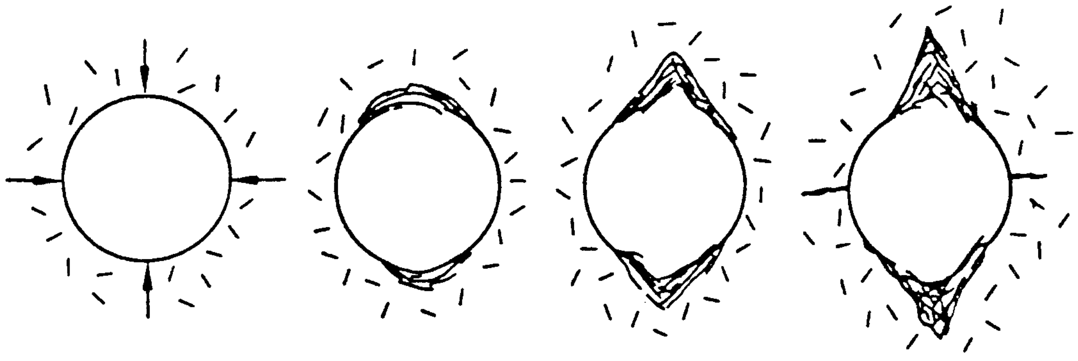


Figure 5.6-3 Schematic illustration of breakouts for different ratios of horizontal stress. A, intact borehole. B, moderate breakouts. C, strong breakouts with dog-ears and large stress ratio. D, severe breakout with formation of tensile fracture in the direction of maximum stress.

Classical studies of brittle failure and related problems of well-bore instability assume that the rock behaves as a linear elastic material. Failure is then predicted by comparing the stresses at the borehole wall, calculated using elastic theory, with the peak strength of rock measured in the laboratory. A recent study carried out by Santarelli and Brown (1987) has shown that, if the Young's modulus of the rock mass varies as a function of the least principal stress, then the maximum stress and rock failure may initiate at some distance from the borehole wall, and convection could take place in the rock mass outside the borehole wall. The theory of Santarelli and Brown was applied to the granitic rocks at Gravberg by Bjarnason et al. (1987), and it was found that the variations in Young's modulus, due to changes in confinements, are too small to give the stress concentrations and thereby the failure inside the rock mass.

In conclusion, we can overlook this mechanism of fracture generation in hard, granitic rocks and fracturing of intact rock is most likely to be initiated at the borehole wall and propagate into the rock mass. Any movement in the rock mass away from the borehole wall is most likely to take place along existing joints, as was demonstrated in Section 5.2.2.

5.7 Observations of drilling damage in the Gravberg-1 well

Drilling damage of varying types has been found in the Gravberg well. The most obvious have been the large breakouts, causing the shape of the hole to become ovalized. It is, however, apparent that the fractures generated in the drilling process penetrate deeper than can be observed by the hole enlargement. Fracturing also exists on the short axis of the hole, which is indicated by studies of the different spaced acoustic measurements. The drilling damage is apparently much more severe in the sections of more competent rock, compared to the sections where original fractures exist. Various logging observations, as well as the behaviour of the drilling, suggest that the well has been drilled in a vertical zone of weakness or fracture system. However, this system probably has no fractures that were open originally, but they were instead opened locally around the borehole while drilling. These fractures have been observed on the Formation Scanner Images as well as suggested by a very large separation between the different Laterolog resistivity measurements. The limited extent of these open vertical fractures is suggested by the high resistivity readings on the deep reading Laterolog.

5.8 Long-term stability of boreholes

Under this heading the present experience of long-term stability of boreholes in granitic rocks will be discussed. It is followed by an estimation of strain and displacement in the vicinity of an intact borehole assuming steady state creep of the granitic rock. Finally, the phenomenon of creep rupture and time-dependent spalling will be discussed.

Experience

Repeated borehole logging is the obvious technique to collect information about time-dependent instabilities of boreholes. For the scientific borehole at Cajon Pass (cf. Section 3.4.3) borehole breakouts were studied with different geophysical techniques. Borehole breakouts were detected by means of Caliper logs and Acoustic Televiewer and were first observed below 1830 m. About one year later, in March 1988, a repeated logging with the Televiewer was conducted with the intention of recording possible changes in the magnitude and direction of breakouts. The research group at the Departement of Geophysics, Stanford University, was responsible for this study and they were unable to detect any time-dependent changes of the breakouts, M. Zoback (personal communication).

As a part of the site investigations conducted by the Swedish Nuclear Fuel and Waste Management Company (SKB) for the final storage of high level radioactive waste, the Swedish Geological Co has core-drilled a large number of boreholes in the Precambrian rocks of Sweden The overall experience from the drilling is that the large majority of the boreholes are stable. If instability occurs it is bound to appear during the drilling phase. Once a borehole is drilled to a depth of 500 to 1000 m it remains stable for a long time, K. Ahlbom (personal communication). However, this fact does not exclude the appearance of breakouts in the deeper sections of the boreholes.

Rock creep

Rock is a viscoelastic material. Therefore stress or displacement can change with time when the loads of pressures on the rock change. The magnitude of displacement at the wall of the intact borehole can be calculated. Assuming the borehole geometry and the loading according to Fig. 5.8-1, and viscoelastic behavior according to a Burger material, the radial displacement, U_r, at the wall of the borehole as a function of time t, is:

$$U_r(t) = A + B\left(\frac{1}{2} - \frac{a^2}{4r^2}\right)\left(\frac{1}{G_2} + \frac{1}{G_1} - \frac{1}{G_1} e^{-(G_1 \cdot t/\eta_1)} + \frac{t}{2}\right) \tag{5.4}$$

where

$$\begin{aligned} A &= \frac{\sigma_H \pm \sigma_h}{4} \frac{a^2}{r} \\ B &= (\sigma_H - \sigma_h) \frac{a^2}{r} \cos 2\theta \end{aligned} \tag{5.5}$$

The maximum displacement at steady state creep or secondary creep at the wall of the borehole is:

$$U_r(t) = a \cdot \frac{\sigma_H}{2} \frac{t}{2} \tag{5.6}$$

Consider a 800 m diameter borehole at a depth of 4 km. The state of stress according to eqs. (2.3) and (2.5) will be σ_H = 113 MPa. The viscosity of the granite is estimated to be 10^{20} Pa/S for the temperature and pressure conditions at a depth of 4 km.

The calculation will yield a rate of displacement, U_r, of $14.2 \cdot 10^{-6}$ mm/y, which corresponds to 14 mm of displacement after one million years. An ellipticity of about 3 per cent, calculated with reference to the original borehole diameter, will have minor effects on the sealing material of plastic bentonite. Hence, any steady state creep of intact granitic rock is not likely to cause borehole instability.

Tertiary creep and spalling

As long as the stresses in the wall of the borehole are less than or equal to the creep stresses for secondary creep, deformation is maintained at a steady state. If stresses increase, and/or the time is very long, the deformation reaches the tertiary creep where strain is accelerating, cf. Fig. 5.8-1. This process will finally lead to rupture or failure of the rock material.

The only way stresses around the circular borehole can increase to a stage where tertiary creep and rupture will occur is when stable breakouts are formed. The new shape of the borehole will increase the stress level at the region of breakouts and the high stresses - although at local points - cause tertiary creep and later failure. Since the state of stress is complicated, the material properties non-linear and the failure mode is less known, there is no theory to apply to the problem of the time-dependent spalling of a borehole, cf. Section 5.2.1. New damage mechanics may enhance the knowledge in this field.

Borehole stability in jointed rock was treated in Section 5.2.2 of this report. The results show that the borehole wall remains stable for any studied depth down to 7.5 km, borehole radius and mud weight. It is important to remember that the analysis by means of the distinct element method does not consider time dependency. Water flow, weathering and hydration of the joint fillings are likely to cause a weakening of the joints in the rock mass. At present there are very few studies reported in the literature about the time-dependent properties of joints and joint fillings. This is an area that needs more research.

In conclusion, experience tells us that instabilities in a borehole are most likely to appear duting the drilling phase. The calculated rate of displacement from rock creep of the wall of 800 mm diameter borehole at a depth of 4 km is found to be $U_r = 14.2 \cdot 10^{-6}$ mm/y, i.e. an ellipticity of the borehole of about 3 per cent. When stable borehole breakouts are formed, the stress magnitude can increase to a stage where tertiary creep and rupture will appear. The same result will be obtained for slightly lower stress magnitude acting over a longer period of time. Water flow, weathering and hydration of joint fillings are likely to cause a weakening of the joints in the rock mass and thereby loosen the rock mass at the borehole wall.

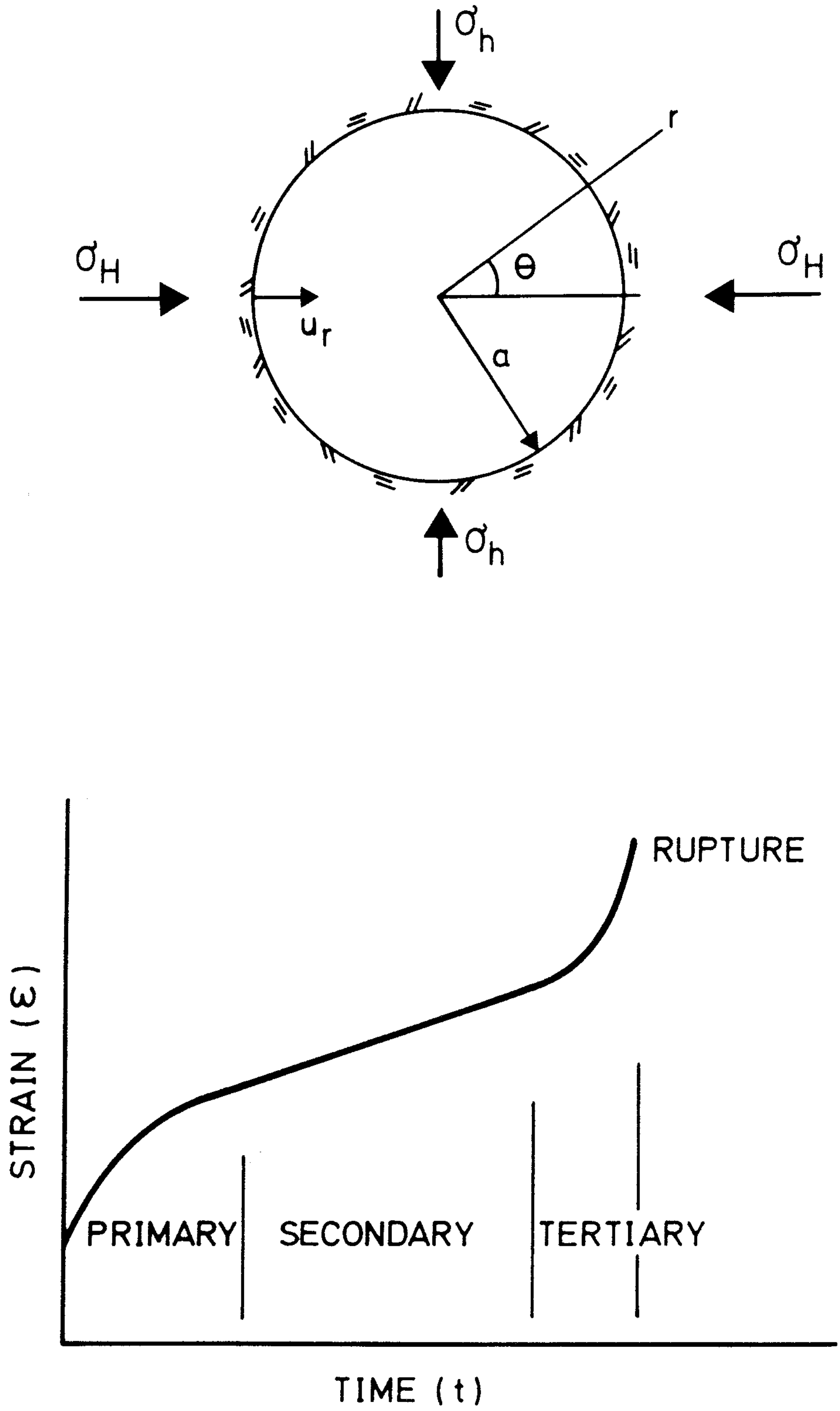


Fig. 5.8-1 Creep of intact borehole. a) Geometry and stress state. b) Creep curve of rock material.

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STORAGE OF NUCLEAR WASTE IN VERY DEEP BOREHOLES STAGE A: PRELIMINARY REVIEW

Feasibility study and assessment of economic potential

Summary and Conclusions

The objective of this report is to review current technology and make a first outline of a feasible system for disposal of radioactive high level waste in deep boreholes. The report presents Stage A of a programme with the following suggested content:

- Stage A: Preliminary review
- Stage B: Outline design and quality assurance review
- Stage C: Overall facility plan and cost analysis

The basic idea of the concept is to deploy the waste at such great depth that the time for migration of radionuclides to the biosphere becomes so long that either adequate decay has taken place or sufficient dilution of the waste has occurred to eliminate any safety hazard.

The basic concept is that the bedrock seems to be generally tighter below a depth of between 1000 to 2000 m. This statement is based on geophysical investigations and is now also verified by recent results from the Swedish deep gas drilling at Gravberg.

A review of previous studies is based primarily on the report "Very Deep Hole Systems Engineering Studies" written in 1983 by Woodward-Clyde, Consultant for the Office of Nuclear Waste Isolation, ONWI. The concept presented in that report is to a high degree based on significant technical development within the next 20 years. This may, however, not be the case, particularly when the oil industry today is in a state of deep decline and a limited number of deep wells are anticipated in the next 10 years. Our assessment is that the concept presented is based on non-existant technology to such an extent that the possibility of actually implementing the system should be considered as highly doubtful.

A VDH repository could, however, be designed and constructed with existing technology. Some development work will be needed and in this report we put forward some new ideas about casing programmes, waste deployment, plugging and sealing.

A summary is given below of the proposed concept:

- The boreholes should be 5-6 km deep with a deployment zone from 2-3 km depth.
- The borehole diameter should be in the range of 300-375 mm at the bottom.
- Due to safety aspects, the boreholes should be cased from top to bottom during deployment.

- o Above the deployment zone, a short section of casing should be removed to allow a final seal and plug to be installed.

- o Additional seals could be provided above the deployment zone at positions where "windows" have been milled in the casing to provide a multiple barrier system.

To accommodate the assessed quantity of Swedish high-level waste, about 31 boreholes will be needed. The boreholes should preferably be drilled deviated with a maximum of 7 holes from one site. A total number of 5 sites will then be needed for the complete storage facility. Possibilities may exist to reduce the number of boreholes required, and this should be considered in a future study.

The location of a VDH repository is to a large extent very likely to fulfil similar geological requirements as a mined repository. Initially, similar geological investigations to those carried out within the KBS project will be needed. After this, additional information from greater depths will be collected by geophysical methods and from deep exploratory boreholes.

A very brief cost estimate has been made which indicates a cost level of SEK 7000-8500 M excluding the encapsulation station building and surface waste handling. The total storage cost would appear to be well within the costs for a mined repository on the lines of the Swedish KBS-3-concept.

In conclusion, Stage A of the VDH-study has shown that disposal of radioactive waste in deep boreholes may be a feasible and economical concept using today´s technology. However, considerable engineering and development work will be needed in a number of fields. Some crucial parts put forward in this report are:

- o Development of a hydrogeological and rock properties model as a function of depth.

- o Surface handling.

- o Casing programme during drilling and deployment.

- o Waste deployment.

- o Plugging and sealing.

- o Safety aspects and concerns (Quality assurance).

STORAGE OF NUCLEAR WASTE IN VERY DEEP BOREHOLES:

PART II OVERALL FACILITY PLAN AND COST ANALYSIS

Vattenfall, December 1989

PREFACE

As part of SKB's (Swedish Nuclear Fuel and Waste Management Co.) research and development programme, alternative concepts for the permanent storage of high level nuclear waste are being studied.

The current study began in 1987 and two interim reports have been written:

Stage A: Preliminary Review

Stage B: Outline Design and Quality Assurance Review

This report represents Part II of the Final report which builds upon work done during Stage A and Stage B in addition to further studies carried out during 1989. Part I, which complements Part II, deals with the geological aspects of a very deep borehole repository. The two reports are:

Part I: Geological Considerations for the Very Deep Borehole Concept

Part II: Overall Facility Plan and Cost Analysis

This report has been prepared for SKB by Vattenfall, Dep. BEL as a joint venture between different experts in a number engineering and geological fields.

Stockholm, December 1989

Håkan Sandstedt

Storage of Nuclear Waste in Very Deep Boreholes: Part II
Overall Facility Plan and Cost Analysis

Contents

SUMMARY AND CONCLUSIONS

Introduction

This report constitutes Part II of a feasibility study for storage of high level radioactive waste in deep boreholes. The basic idea is to deploy the waste at such great depth that the time for migration of radionuclides to the biosphere becomes so long that adequate decay has occurred to eliminate any safety hazard. In the Interim report, Stage A, it was concluded that disposal of radioactive waste in deep boreholes may be feasible and economical using today's technology. It was also concluded that considerable engineering and development work will be needed in a number of fields.

Part II of this feasibility study concentrates on the engineering aspects of the very deep borehole concept (VDH).

Temperature field

The temperature around a borehole containing radioactive waste will be dependent on the amount of waste in each canister and the depth of the borehole. Maximum temperature increase from a canister with an average initial heat production rate of 58 W/m per year is 17°C at the canister/ borehole interface. This temperature is proportional to the amount of waste, and will decrease with time. The increased temperature will lower the density of the water in the borehole and generate a convection cell. This convection cell will be the main driving force for transportation of radionuclides to the surface.

Hydrogeological modelling

Initial estimates of water convection rates along an open borehole containing waste indicated the necessity of bentonite sealing. A model assuming radial symmetry and convection only in the surrounding rock with a permeability of 10^{-17} m^2 resulted in maximum flow rates of $0.061/m^2/yr$. In an effort to estimate flow rates in more complicated geological situations 2-D steady state modelling has been carried out in heterogeneous media. A problem with the 2-D modelling is that it is not possible to represent the true heat flux and temperature field around the borehole simultaneously. In order to reproduce the correct temperature field more heat must be input than will be the case if a true 3-D model is used. In addition, steady state solutions have been obtained using the high initial heat production rates. These two factors result in that the flow rates calculated are grossly overestimated, however, they are still low. Modelling of higher permeability fracture zones present in an otherwise homogeneous rock indicates that they are only a problem if they intersect the borehole.

Simple modelling of deploying the waste in saline formation water below fresh surface waters indicates that if the water is saline enough no water which has been in contact with the repository will reach the surface.

Engineering, cost and analysis

Three options for the VDH option are considered; option A, deployment of waste in an 800 mm borehole from 2 to 4 km; option B, deployment of waste in a 375 mm borehole from 2 to 5.5 km; and option C, deployment of waste in a 375 mm borehole from 2 to 4 km. At present, option A is considered to be the most attractive from an engineering and economic standpoint. Option A invovles the use of shaft drilling technology to reach the required depths for deployment. Based on previous experience it is considered that the proposed hole dimensions and depths for option A are possible to realize using todays technology.

Major innovation will be required for the casing which is necessary for the canisters to be deployed. The design is such that the casing will have a high void ratio and be composed of a nonreactive material to eliminate generation of gases. The high void ratio is necessary to allow adequate sealing by the bentonite plugs in the deployment zone. In addition, the casing is designed without the need for cementing it in place. This results in that the only materials present in the deployment zone will be the canisters, the non-reactive casing and the bentonite.

It is estimated that drilling time for option A is about 535 days including investigations in the borehole and coring in the deployment zone. An additional 365 days is necessary for canister deployment and plugging of the upper 2000 m. This implies that a borehole can be drilled, the waste deployed and the borehole sealed in less than 3 years. The cost for one borehole will be 388 MSEK including waste deployment.

Plugging and sealing

Isolation of highly radioactive waste in very deep boreholes implies that the deployment zone needs to be effectively separated from the biosphere. Bentonite clay is an excellent sealing material in deep boreholes. For option A it is envisioned that the borehole will be sealed in three stages, the deployment zone (2-4 km), the central part (0.5-2 km) and the upper part (0-0.5 km). In the deployment zone a special high density deployment mud will be emplaced prior to deployment of the canisters. In between the canisters, 1 meter long cylinders of highly compacted bentonite will be deployed. Swelling of this bentonite will result in maximum conductivities of 10^{-9} m/s in the deployment mud and 10^{-11} m/s in the bentonite plugs for a 10 % $CaCl_2$ salt water environment. If the salt consists of CaCl or if the water is fresher than the conductivities will be even lower. In the central part above the deployment zone the hole will be sealed using highly compacted bentonite plugs. Conductivities in the borehole in this zone will probably be lower than in the deployment zone since it is expected that the pore water will be fresher at these depths. It is suggested that the lower half of the upper part (250-500 m) should be sealed with asphalt because of its capability to remain viscous for long periods of time. The upper half of the upper part (0-250 m) should be sealed with concrete to protect against errosion and abrasion.

Strategy for site selection

The basic considerations for chosing the deployment site have been outlined in Part I and do not differ dramatically from those which would be used for siting KBS-3. The main difference in siting lies in the greater flexibility which the VDH concept allows. The VDH concept is not nearly as dependent upon specified near-surface geological conditions since the concept itself is based upon the rock quality improving dramatically with depth. This implies that greater weight can be given to logistical considerations, such as transport of spent fuel, when chosing the site than for the KBS-3 concept. If depths to highly saline waters prove to be fairly shallow along the Baltic then a suitable site may lie close to a port so that all transport to the site is by sea. It is also possible to use multiple sites if not enough area is available at one site or if logistics or safety aspects determine that multiple sites are an advantage.

Preferred borehole concept

From an engineering and economical standpoint the large diameter (0.8 m) borehole option to 4 km depth is the preferred choice for storage of the high level radioactive waste. Two possible canister configurations for deploying the waste in the large diameters boreholes with and without rod consolidation is presented. If no rod consolidation is done then 4 BWR or 2 BWR and 1 PWR can be deployed in each canister. In this configuration the waste emits an average of about 110 W/m resulting in a maximum temperature in the surrounding bentonite of 120°C at 4 km depth in the first year. With rod consolidation the canisters will emit approximately twice this figure resulting in a maximum temperature in the bottom of the hole of 150°C in the bentonite. Whether a temperature of 150°C in the bottom of the hole during the first few years of deployment is acceptable needs further study.

Total cost estimate for preferred concept

A cost analysis shows that deployment of the entire stock spent high level radioactive waste without rod consolidation will cost 19 390 MSEK. With rod consolidation this cost can be reduced to 10 526 MSEK. For a total comparison with the KBS-3 concept, however, it is essential to consider the cashflow and thus the present value discounted to year 1990. For the KBS-3 concept a fairly large part of the total cost is invested early and before deployment. For the VDH concept the drilling, deployment and sealing are integrated in one operation after each other during 3 years. The present value will, with an interest rate of 2.5%, be 6850 MSEK and 3719 MSEK respectively for the two VDH alternatives.

The above figures cover the actual drilling, deployment of the waste and borehole investigations. Other costs, such as pre-site investigations, emplacement of waste in canisters and transportation networks, are not included. Even with rod consolidation, the cost for the VDH concept is greater than that for KBS-3. However, cost savings may be achieved when considering some of the factors which are not included in the cost estimate such as safety levels, greater flexibility in site selection, and the greater flexibility in the VDH concept to adapt to technological innovations. Cost savings in the order of 3000 MSEK has been mentioned if a suitable location could be find close to CLAB.

Suggested future work

To determine if the VDH is a viable alternative to KBS-3 there exists a great deal of research which needs to be carried out. Much of the research is independent of the final concept which is chosen and is of such a nature that it will be valuable regardless of the chosen concept. It is suggested that reviewing of results from other deep wells be continued since a great deal of information is obtained at relatively minor cost. Further research needs to be done in the drilling field. Some of this is quite expensive, but if serious consideration is to be given to the VDH concept it is required. The same is true for plugging and sealing of the borehole, although here much of the research also has applicability to other concepts. Better modelling needs to be done on the effects of the deployment of the waste on water circulation patterns, preferably 3-D transient modelling where consideration of salinity differences are included directly. A pilot study should be carried out to determine the viability of electrical methods for determining depths to more saline water. Finally, the geological assumptions for the VDH concept need to be tested with the drilling a 3.0 km deep investigatory borehole. Aside from the VDH specific research concerning the drilling and hydrogeological modelling studies, all of the suggestions for future work will have important application to the general problem of storage of nuclear waste in crystalline rock.

1. INTRODUCTION

This report constitutes Part II of a feasibility study for storage of radioactive high level waste in deep boreholes (VDH). The basic idea is to deploy the waste at such a great depth so that the time for migration of radionuclides to the biosphere becomes so long that either adequate decay has taken place or sufficient dilution of the waste has occurred to eliminate any safety hazard. In the Interim report, Stage A and B, it was concluded that disposal of radioactive waste in deep boreholes may be feasible and economical using today's technology. It was also concluded that considerable engineering and development work will be needed in a number of fields.

Part II of this feasibility study focuses on the following issues:

- Temperature field around the repository and hydrogeological modelling
- Deep borehole engineering (drilling) and cost analysis
- Plugging and sealing
- Site selection
- Total cost estimates

Compared to the drilling technology discussed in the Interim report based on oilfield experience the present study has been concentrated towards drilling large diameter boreholes to 4 km depth with shaft drilling technology. The large diameter borehole, 0.8 m in the deployment zone, has some obvious advantages in cost and safety compared to slim boreholes. The cost for the VDH concept will be higher than the KBS-3 concept, but the VDH has many advantages with respect to safety and flexibility and the potential for cost savings for the system presented must be judged as great.

This report is written under the commission of SKB, Swedish Nuclear Fuel Waste Management Co. in very close cooperation with Anders Bergström, as a joint venture between the people below:

John Beswick	Deep hole engineering
Christopher Juhlin	Project management, Geoscientific issues
Gunnar Gustafson Urban Svensson Bengt Hemström	Hydrogeology
Roland Pusch	Plugging and sealing
Håkan Sandstedt	Project management, Technical issues

Part I of this feasibility study concentrates on geological aspects of the very deep borehole concept.

2. TEMPERATURE FIELD

2.1 Introduction

Once radioactive waste is deployed in the boreholes it will begin to generate heat which will raise the temperature in the surrounding cannister, bentonite and rock above that of the geothermal gradient. Knowledge of what the temperature field is due to this heat is essential for designing the bentonite plugs and for estimating the size and velocity of the convection cells generated by the deployed waste. In the section that follows the temperature profile around a single borehole is estimated for the slimhole concept. Although the larger diameter hole is the preferred concept the results scale linearly and have been used as input for calculations for the convection cells in Chapter 3 on hydrogeological modelling. Current thinking is that a temperature in the bentonite of 120°C should not be exceeded, however, it is possible that temperatures as high as 150°C in the deeper parts of the borehole are acceptable.

2.2 Temperature estimate

The temperature increase around a deep repository hole has been calculated using the finite element program ENERGY (Clay Technology AB). The nuclear waste is assumed to be deployed between depths of 3000 m and 5500 m in the boreholes.

Assumptions

The calculation is made with the following assumptions:

Inner diameter of canister:	0.18 m
Average heat generation:	58 W/m year 0
Maximum heat generation:	80 W/m
Rock boundary:	10.9 km

The element mesh is one-dimensional with rotational symmetry.

Material data:

Position	Material	λ (W/m, K)	c (Ws/kg, K)	ρ (kg/m^3)
canister	copper	380	390	8930
buffer	clay	1.0	2000	1400
rock	granite	3.0	740	2700

Heat production versus time (Linear interpolation between following values):

Time (Year)	Slim hole Heat generation on average (W/m)	Large hole option 1 Heat generation on average (W/m)	Large hole option 2 Heat generation on average (W/m)
0	58.0	110.0	286.0
20	43.9	83.3	216.6
60	28.2	53.5	139.1
260	12.5	23.7	61.6
960	4.76	9.0	23.4
3000	1.57	3.0	7.8
10000	0.941	1.8	4.7
30000	0.392	0.7	1.8
100000	0.086	0.2	0.3

Results

The temperature increase is calculated from start to 1 milion years. The results are summarized in Figure 2.2-1 where the temperature increase is plotted as a function of the distance from the centre of the hole at certain chosen times from 9 days (9D) to 434 years (434Y). The figure shows that the maximum temperature is reached after 6 years with $T = 17.0^{o}$ at the canister/clay interface and $T = 13.3^{o}$ at the clay/rock interface.

In this calculation the average heat generation along the centre line of the hole has been used with the initial value 58 W. The effect of a change in power can easily be calculated from Figure 2.2-1 since the temperature is proportional to the power. The influence of a temperature increase caused by surrounding holes can also be easily calculated from Figure 2.2-1 since the temperatures can be superimposed.

The temperature, especially in the clay, can locally exceed the values shown in Figure 2.2-1 since the initial power is 80 W/m canister in the 4.3 m long canisters. The maximum temperature in the clay can thus be calculated by multiplying the values in Figure 2.2-1 by 80/58.

Figure 2.2-1 shows that the influence at the distance 500 m is not seen until after several hundred years. After 500 years $T = 0.5^{o}$, but at that time the temperature in the clay is so low ($T < 5^{o}$) that the superimposed temperature increase from all surrounding holes will not make the total temperature increase of the clay exceed 17^{o}.

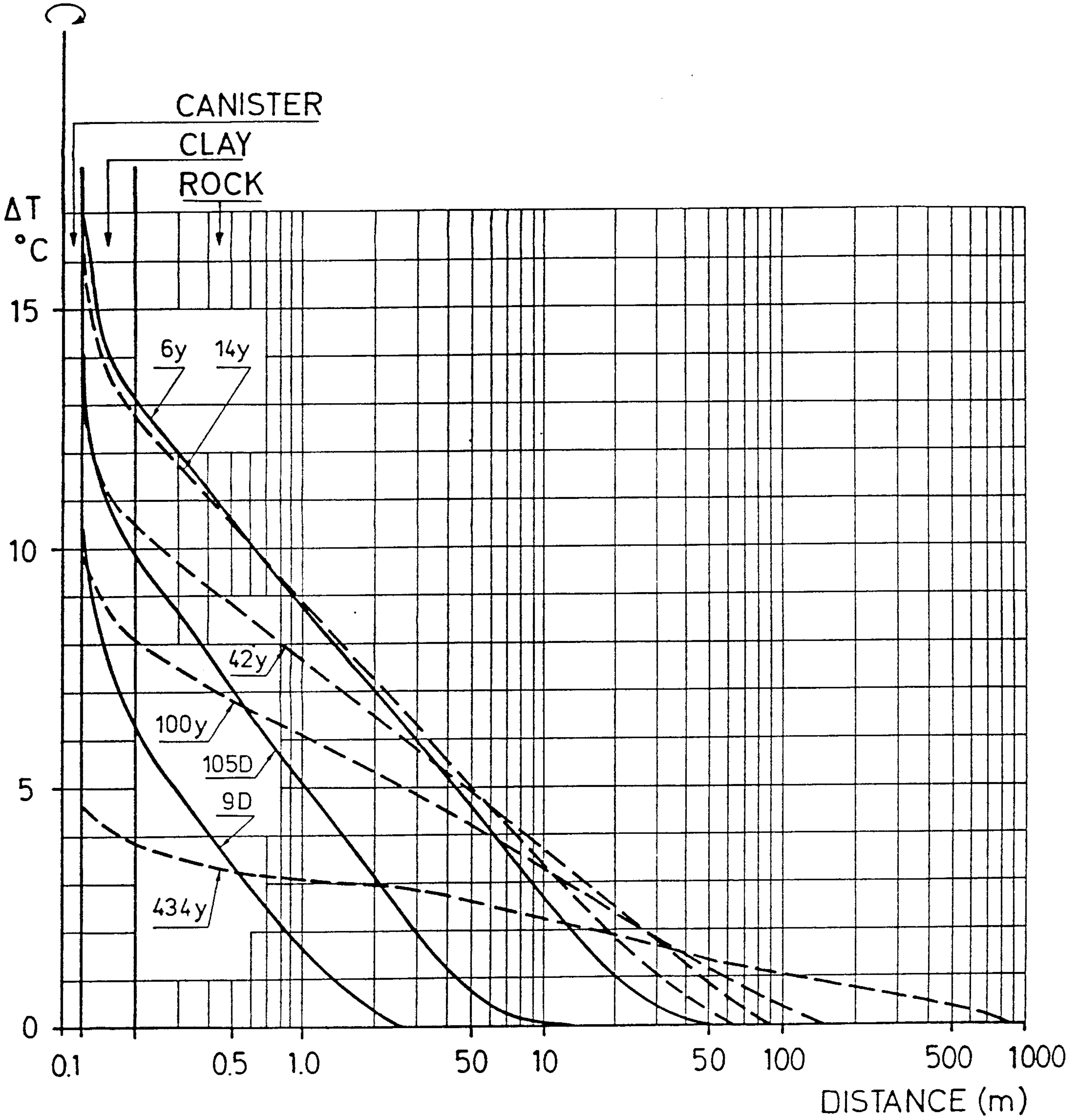


Figure 2.2-1 Temperature increase from one borehole as a function of the radial distance from the centre of the canister for the slimhole option.

2.3 Estimation of the temperature in the canister-clay interface versus canister sizes and depth

The temperature calculation presented in Section 2.2 is valid for a fairly slim canister designed for a 5.0 km deep borehole. In a shallower borehole the temperature impact from the souronding rock will be less. This makes it possible to deploy more uranium for each metre and still not exceed the temperature limit.

An estimate is given below of the maximum heat production of the canisters in order not to exceed 150°C, 125°C or 100°C respectively in the canister-clay interface. The results are presented in Figure 2.3-1.

Assumptions

- The temperature gradient in the rock is 16.1°C per km.

- The surface temperature in the rock is 4°C.

- The design of the canisters is in accordance with data presented in Chapter 7.

- $T_{max} = 4°C + D \cdot 16.1°C + 0.293 \cdot$ heat production (W/m)

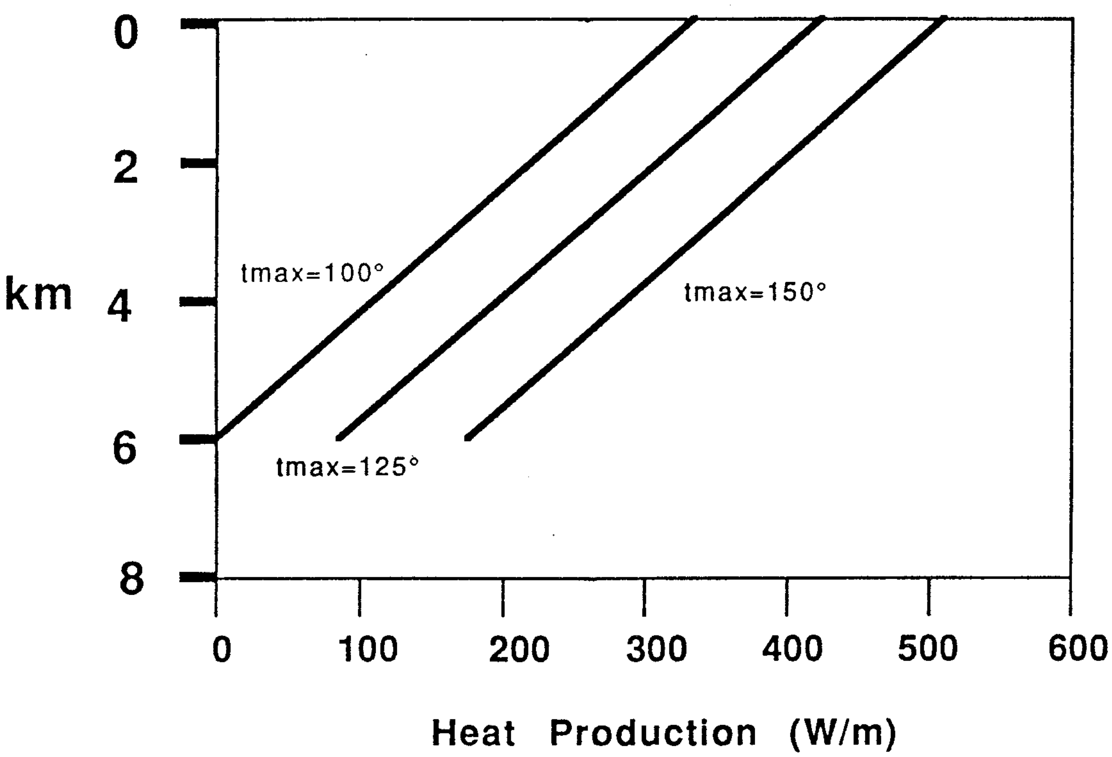


Figure 2.3-1 Waste deployment per meter versus depth for maximum permissible temperatures of 150°C, 125°C and 100°C.

3. HYDROGEOLOGICAL MODELLING

3.1 Introduction

Convection cells generated by the deployment of radioactive waste have been calculated for a variety of assumptions. All of these assumptions imply simpler geological conditions than exist in reality. No consideration has been given to flow along individual fractures, instead, the rock has been modelled using equations valid for flow in porous media. Reasons for this are twofold. First, at this stage, order of magnitude calculations are of greatest interest and porous media modelling is deemed adequate. Secondly, flow in fractured media is poorly understood and using any of the variety of models that exist may lead to uncertain results. The most complicated geometry which has been employed up to now is 2-D heterogeneous models. Further insight into the effects of deployment of waste in deep boreholes can probably be gained by going to 3-D heterogeneous models rather than by using more complicated 2-D models.

Initial calculations in 1988 of flow for a radially symmetric geometry around a single borehole similar to the slim hole concept considered three different cases (see Appendix 1).

1. No bentonite sealing (open borehole).
2. Bentonite (impermeable) plugs 50 m long every 450 m.
3. Continuous bentonite (impermeable) sealing along the borehole.

The first case resulted in a flow of 65 litres/day along the borehole which was considered unacceptable. Case 2 resulted in 4.1 litres/day along open hole sections which was also considered unacceptable. The third case resulted in a flux of 0.06 litres/m^2/year through the surrounding rock which is considered acceptable and indicates the necessity for plugging and sealing effectively.

In this chapter results are summarized from additional work carried out in 1989. Most of the work has focused on calculating convection cells in the surrounding rock assuming effective continuous bentonite sealing where the influence of fracture zones has been taken into account (Hemström 1989). The models are two dimensional and assume a water of similar composition throughout the depth of the borehole.

Consideration has also been given to the case where the water composition changes with depth, i.e., the salinity increases (Appendix 2). This is a situation which is probably likely to be encountered in some areas and may provide very advantageous conditions for deployment using the very deep borehole concept.

3.2 Geological Models

A series of simple geological models have been set up and calculations have been made for flow fields based upon these models. In the modelling it is assumed that the waste is deployed from 2 to 4 km and that 286 W/m is emitted from it along the deployment zone. This value is quite high and results in a maximum temperature of 150° in the bentonite seal. The figure 286 W/m has been used since at one point it was thought that the bentonite could withstand these temperatures. In reality, an initial average heat flow of about 110 W/m is more likely and this results in that the velocities presented in this report are too high. Another factor contributing to high velocities is the steady state assumption used to solve the differential equations for flow in porous media. The heat flow from the repository will decrease with time, however, it has been assumed that the heat flow is constant with time at a rate of 286 W/m which is only true for the first year of deployment. The amount of heat flow into the rock is also over-estimated by the 2-D geometry used in the modelling. In order to obtain an equivalent temperature profile as a function of distance from the borehole as that presented in Chapter 2 it is necessary to allow twice as much heat per unit area to flow into the rock in the 2-D case as compared to the radially symmetric case. The above factors result in that the amount heat of that flows into the surrounding rock is grossly overestimated and that the resulting specific discharges are also grossly overestimated for the given geological models. An additional factor to consider is that the cannisters will not leak immediately. If they remain intact for 100 years then the temperature is almost half of the initial further reducing the possibility of radionuclide migration.

Eleven geological models have been run consisting of the following:

1. Base case model with a constant permeability of 10^{-17} m^2 (Fig. 3.1-1a).

2. Upper 1500 m of rock has a permeability of $6x10^{-16}$ m^2 while the rock below 1500 m has a permeability of 10^{-17} m^2 (Fig. 3.1-1b).

3. Permeability decreases with depth according to the function $k = e^{0.0018z} \times 10^{-15}$.

4. A 30 m wide vertical fracture zone is present 200 m away from the repository (Fig. 3.1-1c).

5. A 30 m wide fracture zone intersects the repository at a 45^o angle (Fig. 3.1-1d).

6. The rock has a mean permeability of 10^{-17} m^2 with a standard deviation of 1.0.

7. As 6, but with a standard deviation of 2.0.

8. As 6, but with a standard deviation of 4.0.

9. The rock has a mean permeability that decreases with depth as in model 3 and with a standard deviation of 1.0.

10. As 9, but with a standard deviation of 2.0.

11. As 9, but with a standard deviation of 4.0.

In the modelling the following were assumed

Porosity of rock; ϕ	0.3%
Thermal conductivity of rock,	3.0 W/m/K
Specific heat of rock, Cp	740.0 J/kg/K
Density of rock, p	2700.0 kg/m^3

The assumed permeabilities are based on measurements which have been carried out in crystalline rock. An average permeability of 10^{-17} m^2 for crystalline rock below 1500 m may, in fact, be quite high. However, lacking data on permeability at these depths this relatively high value has been assumed.

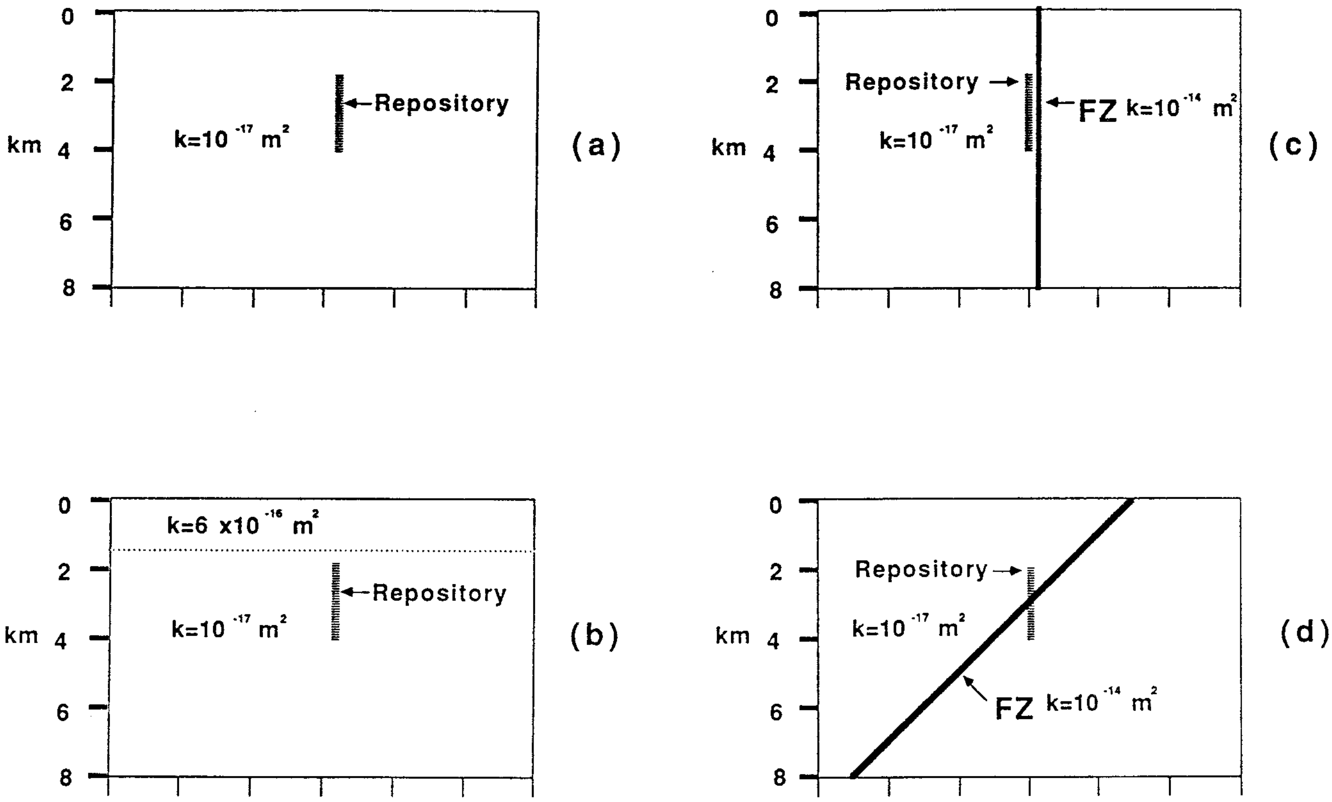


Figure 3.2-1 Examples of some of the geological models where fluid convection due to the deployment of waste has been calculated. Vertical and horizontal scales are the same.

3.3 Results from the modelling

The maximum vertical specific discharges along the borehole are listed in Table 3.3-1.

Table 3.3-1. Maximum specific discharge along the borehole for the various models.

Model	Max. spec. dis. ($l/m^2/yr$)	Spec. dis. at 500 m
1	0.88	0.019
2	1.01	0.160
3	0.76	0.140
4	0.47	
5	17.0	2.4 at 870 m in FZ
6	3	
7	20	
8	200	
9	3	
10	< 20	
11	< 200	

In all cases the flow in the rock along the borehole is greater than that found for the radially symmetric case (Appendix 1) of 0.06 $l/m^2/yr$. Note, however, that in those cases where the specific discharge has been calculated at 500 m the vertical flow is considerably less than in proximity to the borehole. It should also be noted that the flow rates have probably been overestimated by at least an order of magnitude for the reasons discussed in Section 3.1. Even though the flow rates are overestimated a great deal of insight may be gained from studying the relative values and the flow patterns.

There is a good correlation between the permeability of the rock and the specific discharge rate through the rock. This is as expected since the water will tend to flow towards and in the more permeable zones. The very high maximum velocities in models 7, 8, 10 and 11 are due to flow in zones which have permeabilities considerably greater than 10^{-17} m^2. In these models there are also zones with very low specific discharge rates (see Hemström 1989). The average vertical discharge rates do not differ that much from the corresponding deterministic models (models 1 and 3). As long as these zones of high permeability do not form a network which connects the repository to the surface the actual transport time of radionuclides will not differ dramatically.

Models 4 and 5 (Figs. 3.2-1c and 3.2-1d) can be considered cases where the fractures form a continuous system of higher permeability which provide a short circuit for fluid transport to the surface. Model 4 illustrates where a vertical fracture zone is present close to the borehole, but does not intersect it. This type of fracture zone is the most difficult to detect with surface and borehole geophysics. Results from the modelling of the flow field are shown schematically in Figure 3.3-1. The vertical specific discharge rates are very high in the fracture zone itself with a maximum of 30 $l/m^2/yr$. However, the majority of this water comes from depths below the repository which will be uncontaminated. This water then has a tendency to flow in the horizontal direction from the fracture zone towards the repository. The end result is that the fracture zone has minor influence on the amount of radionuclides which can be transported in water to the surface. In the case where a fracture zone intersects the borehole, model 5, the situations is quite different. A schematic of the results from the modelling is shown in Figure 3.3-2 where it can be seen that water in direct contact with the repository is transported rapidly away from it in the fracture zone. As the water flows to the surface some of it is rerouted back to greater depths as specific discharge rates decrease within the fracture zone, but a considerable portion could still reach the surface. This case illustrates the importance of sealing off any major fracture zones which intersect the well which may have permeable paths leading to the surface.

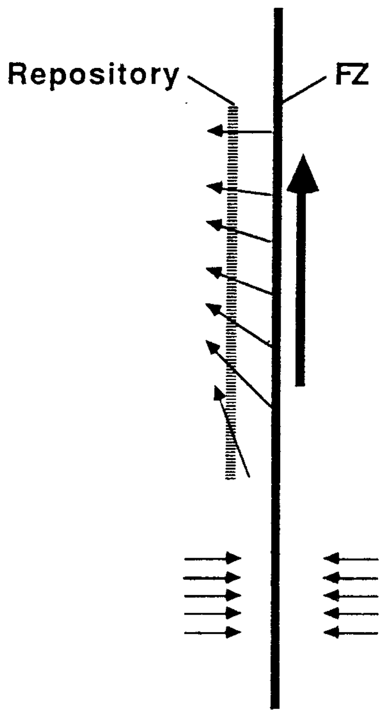


Figure 3.3-1 Main flow paths when a vertical fracture zone is located 200 m from the deployment zone.

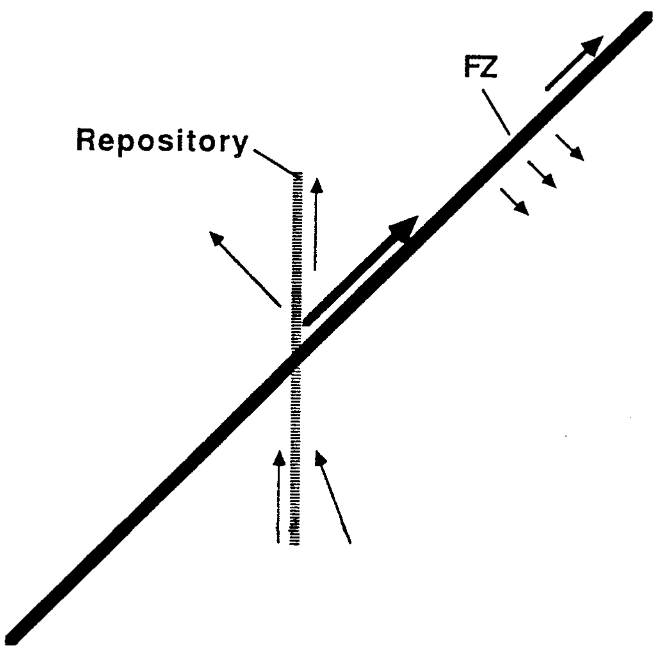


Figure 3.3-2 Main flow paths when an inclined fracture zone intersects the deployment zone.

3.4 The influence of an increase in salinity with depth

As noted in Part I of this report, it is normally found that the salinity of the pore water increases dramatically with depth in most areas and in some cases approaches saturation. Ideally, this increase should be included in the modelling which was discussed in Section 3.2, however, this has not been possible due to the complexity of the problem. Instead, a simpler approach has been taken where it is assumed that the pore water can be divided into two zones, an upper one containing fresh surface waters and a lower one containing saline formation waters. It is assumed that the repository is placed in the saline zone and that this water is then heated resulting in its density decreasing. Calculations are then made to see how far up the boundary between the salt and fresh water will migrate in order to maintain a constant pore pressure below the bottom of the repository (see Appendix 2 for details). Rock properties used in the calculations are those used for the previous modelling. However, the heat input in the present case is considerably more realistic since the calculation assumes radial symmetry and that the source has an initial average heat production of 110 W/m. It has not been possible to take into account directly the changes of heat production rates with time, however, calculations have been made using different rates (Table 3.4-1).

Results from the calculations show that if the saline water has a salinity, close to that of sea water that the boundary will migrate upwards 2454 meters for a constant heat production rate of 110 W/m. If the saline water is close to that found in deeper parts of the Gravberg-1 borehole (15%) then the boundary will migrate upwards 614 meters. As heat production rates decrease the migration of the boundary will be less. These preliminary results indicate that if full scale modelling of non-steady state flow were to be done the actual migration of the boundary would be considerably less than that of constant heat flow of 110 W/m. Clearly, a repository in a saline environment with fresh water above is highly desirable. If the water is highly saline, it appears that no radionuclides at all will be transported to the surface by convection.

Table 3.4-1 Calculated bounds for the rise of a halocline above a single deployment borehole as a function of distance from the borehole and constant heat generation.

Time after deployment (years)	Radius of influence (meters)	Heat generation (W/m)	Density contrast (g/cc)	Rise of halocline from from centr of borehole 0.4 m	4 m	40 m
		Seawater/freshwater				
0	1	110	0.025	2 454	1 766	1 069
10	32	50	0.025	2 143	1 592	939
100	100	18	0.025	781	567	349
40 000	2 000	0.3	0.025	15	11	8
		Brine/freshwater				
0	1	110	0.100	614	442	267
10	32	50	0.100	537	385	235
100	100	18	0.100	195	142	87
40 000	2 000	0.3	0.100	9	3	2

3.5 Relevance to real geology

An important question is how relevant the results from Sections 3.2 and 3.3 are to real geological conditions. The results from Section 3.2 do show that vertical specific discharges will be very low provided that:

1. Flow is adequately described by equations valid for porous media.
2. Permeabilities of the rock mass are less than 10^{-17} m^2 on average.
3. No zones of higher permeability intersect the borehole or if they do they have been adequately sealed.
4. The material used to seal the waste in the borehole itself has a permeability comparable to that of the rock mass.
5. No high conductivity zones parallel to the borehole are created during the drilling operation.

Point 1 above is probably the most uncertain and further work needs to be done on the subject. All measurements of permeability in crystalline rock indicate point 2 to be true. Zones of higher permeability can probably be adequately sealed and waste will not be deployed along them in any case. As discussed in Chapter 5, point 4 appears to be true. Point 5 is also uncertain to some degree and field tests probably need to be carried out to determine the extent of any such phenomenon.

The above assumptions all become much less crucial if the waste can be deployed in a highly saline environment with fresh water above it. In such an environment, the saline waters will probably not convect to the surface for the quantity of waste currently being proposed to be deployed in each borehole.

The modelling done up to now shows that the very deep borehole concept is viable from a geological perspective regardless of the pore water composition assuming the above 5 points are valid. If the repository is placed in a highly saline environment with fresh surface waters above it then the safety assurance can be increased several fold.

4. ENGINEERING AND COST ANALYSIS

4.1 Introduction

Following a Preliminary Review (Interim report Stage A) and an Outline Design and Quality Assurance Review (Interim report Stage B) for the concept of the storage of nuclear waste in very deep boreholes, this study addresses in more detail options for the overall facility plan and the likely cost of implementation.

This section deals with the drilling and associated aspects for a deep hole disposal concept, but of necessity the content overlaps with other topics being reviewed by others. Through various meetings between the various co-workers, a consensus view has been possible and the results presented here take into account the principal considerations of the other key issues.

The chapter includes proposals for the deep hole design and construction of the holes to the final depth together with the design and construction of liners through the deployment zone, the deployment of the canisters and the subsequent sealing operations. The chapter also includes preliminary estimates of time and costs for the implementation programme.

4.2 Geology, temperature and stress regime

For the purpose of this study, it is assumed that the emplacement of a disposal facility would be in a typical Swedish basement granite complex, albeit that candidate sites will be selected with due regard to the local geological setting avoiding undesirable structures and areas of special complexity and variation.

The rock types assumed are therefore granitic with mechanical properties, fabric and mineralogy typical of those for Swedish granites. Uniaxial compressive strengths will probably lie in the range 100-200 MPa with quartz contents in the range around 30%. Hence the main construction of the facility will be through moderate strength, abrasive, brittle rocks with fine to medium sized mineral assemblages. Note that the granites in the Gravberg-1 borehole were relatively weak with uniaxial compressive strengths of the order of 110-140 MPa and quartz contents 30-35%. Sites with significant sedimentary or drift cover have not specifically been addressed, but this possibility would not introduce significant changes to the overall proposals.

It is also assumed that sites will be selected where the hydrogeological regime is suitable for the facility.

Temperature gradients thoughout the Baltic shield are estimated to be about 16°C/km giving in situ rock temperatures of about 70°C and 100°C at 4000 m and 6000 m respectively. This order of rock temperature poses no special problems for drilling.

Data on rock stress profiles at depth are limited, but the recent experiences during the drilling and testing of the Gravberg-1 well in Dalarna has given some data on the likely stress situation at depth and the effects of stress on hole stability in that area. As would be expected in a strong rock, the horizontal stresses are anisotropic with the minimum horizontal stress approaching a value of 0.6 times the vertical stress at depth in the Gravberg-1 well (Stephansson). The vertical stress is assumed to be equal to the weight of the rock and is given by the equation:

$S_V = 0.027\ z$ (MPa) where z is the depth in metres.

It would appear that the maximum horizontal stress within the depths to be considered is higher than the vertical stress, but the difference is small.

The stress difference, i.e. the ratio of maximum to minimum stress, in the Gravberg-1 well is of the order of 1.7 which is not as high as some other experiences in deep granites, but associated with the relatively modest strengths, this gave rise to stress related breakout which started to be evident in the borehole at about 1500 m depth and became severe below 5000 m. The extent of breakout in the Gravberg-1 borehole is summarized in Figure 4.2-1. It can be seen that elongation was typically up to 50% of the drilled diameter. The breakout direction was perpendicular to the maximum horizontal stress direction which ranged in azimuth from 087° to 146° (average 109°) based on an analysis by Clauss from the Technical University of Karlsruhe (reported by Stephansson). Variations in direction of the breakouts apparently observed over different intervals of the Gravberg-1 borehole may be due to the influence of structural control rather than any changes in stress direction. Similar changes in breakout direction were noted in the recent extension of the Cajon Pass borehole in California which penetrated down to 3500 m.

If this stress model is relevant to Sweden as a whole, it has important implications in the debate about the depth of any deep disposal borehole. At this stage, the effects of stress anisotropy with depth and hence the depth of onset of breakout and the degree of hole instability with depth do not warrant a rigorous analytical approach as there are too many variables and inadequate field data. However, taking a pragmatic view, it would appear that boreholes up to about 4000 m are less likely to be troublesome to drill than holes to greater depth. This is also the experience from the Gravberg-1 borehole.

In addition to the potential drilling problems associated with breakouts, the relaxation process also gives rise to displacements in the near wellbore vicinity, the geometry of which will be influenced by the stress environment. This is of concern in the discussion on adequacy of sealing the system after emplacement.

The extent of breakout that may occur in any particular rock depends, on several factors including rock strength, mineralogy, grain size, discontinuity spacing, loading and unloading history and pore pressure changes in the fracture network during drilling. The experience at Gravberg may not be typical for Swedish granite as a whole as the effect of the impact may have resulted in some modification to the rock properties and fracture frequency.

In the Kola super-deep borehole in the USSR, breakout was also reported. A summary is illustrated in figure 4.2-2. This indicates roughly that a 25-35% elongation occurred in the first 4000 m.

In Cornwall, England, in young granite about 350 million years old, no breakouts of any significance were observed in the 2800 m borehole length (2650 m true vertical depth) penetrated.

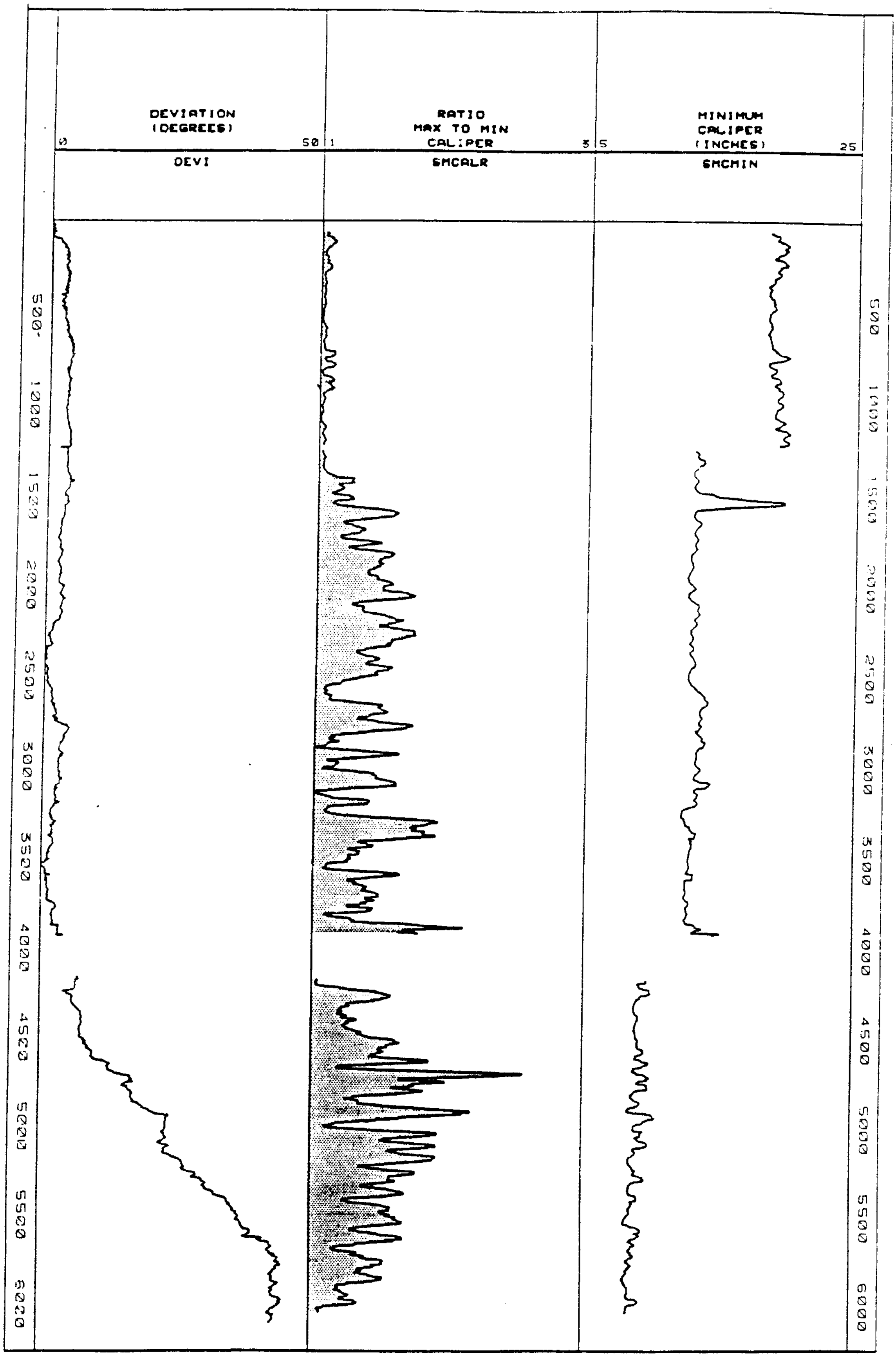
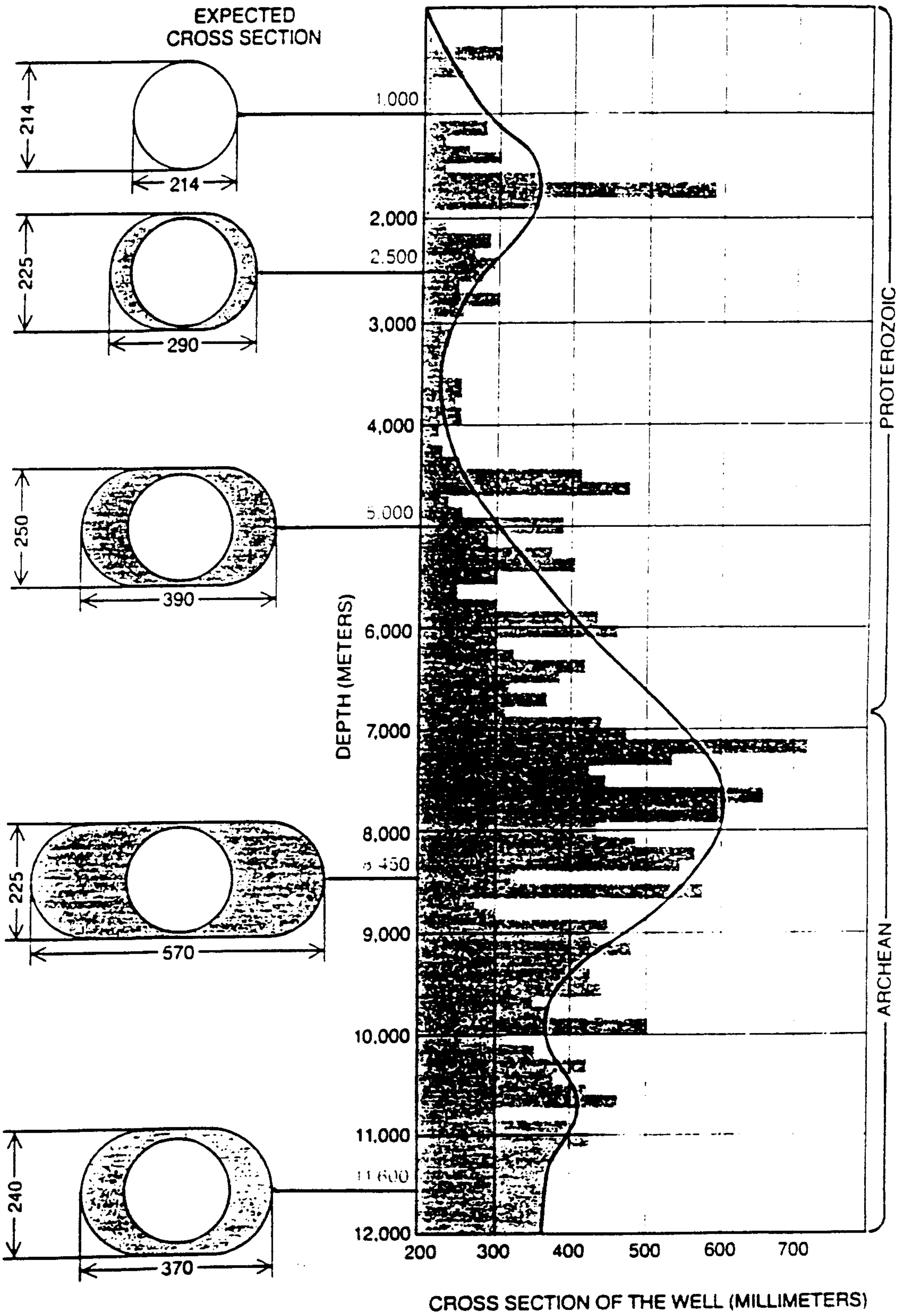


Figure 4.2-1 Summary of the extent of breakout in Gravberg-1 well



Ref: Kozlovsky, Scientific American, Vol 251, No 6, Dec 1984

Figure 4.2-2 Summary of breakout in the Kola super-deep borehole in the USSR

In the 4000 m deep pilot hole for the KTB super-deep scientific borehole in West Germany, elongations were reported in the intervals from 500-1200 m and 1300-2500 m. Some of this may have been due to mechanical 'wear' of the borehole by drilling tools as the borehole inclination varied between 5° and 12°. Below 2500 m, less breakouts were reported, but there is some suggestion that this was due to favourable stress and rock strength conditions.

Antoher factor is the degree to which stress induced breakout during drilling can be mitigated by improvements in the fluid loss properties of the drilling muds. This fact may have contributed to the reduction in breakouts observed in the KTB pilot hole below 2500 m. About this time the drilling team became more aware of the advantages of low fluid loss characteristics of the mud system with the result that corrective action was taken to try to reduce breakouts and hence improve verticality control.

Any deep hole construction project should be preceeded by a comprehensive investigation phase which for the prototype facility should include one or more deep boreholes which are comprehensively logged and probably cored throughout. Heavy duty wireline systems now available are suitable for coring to 4000 m.

4.3 Options considered

Three borehole configuration options have been considered for this study:

Option A: Large diameter borehole to 4000 m
800 mm diameter borehole at TD
Deployment zone 4000 m to 2000 m

Option B: Small diameter borehole to 5500 m
375 mm diameter borehole at TD
Deployment zone 5500 m to 3000 m or 2000 m

Option C: Small diameter borehole to 4000 m
375 mm diameter borehole at TD
Deployment zone 2000 m to 4000 m

These sizes are tentative and should not necessarily be considered fixed, although the order of size should be similar.

The proposed general arrangement of Options A and B for the disposal boreholes are illustrated in Figure 4.3-1. Option C is similar to Option B, but shallower.

The approach in the study has ben to treat Option A (large diameter) as a shaft construction and Opotions B and C (small diameter) as large diameter conventional oil and gas wells utilizing the extensive experience from Gravberg-1 to develop an outline design and costing.

For Option A, assistance has been given by Cobbs Engineering, Tulsa, USA, specialists in big hole design and construction. Cobbs have been involved in many shaft engineering projects including the now aborted Basalt Waste Isolation Project in Washington State, USA (Hanford Project) for the US Department of Energy. Extracts from a report and supplementary studies by Cobbs are included in this report. Copies of the original material provided by Cobbs are available as separate documents.

It should be recognized that all three options present unusual drilling challenges and have not been attempted previously in granites in the depth and size combinations proposed.

Notwithstanding the later discussion on the detail of the various options, the study has highlighted that there is significant preference for Option A as it provides a disposal diameter more suitable for the waste material than the small diameter options. Furthermore, although the cost per hole is larger, the number of holes is reduced. The small diameter options are very restrictive in available diameter probably necessitating rod consolidation of the waste material and much less scope for effective sealing. Hence, the report concentrates on the large diameter option.

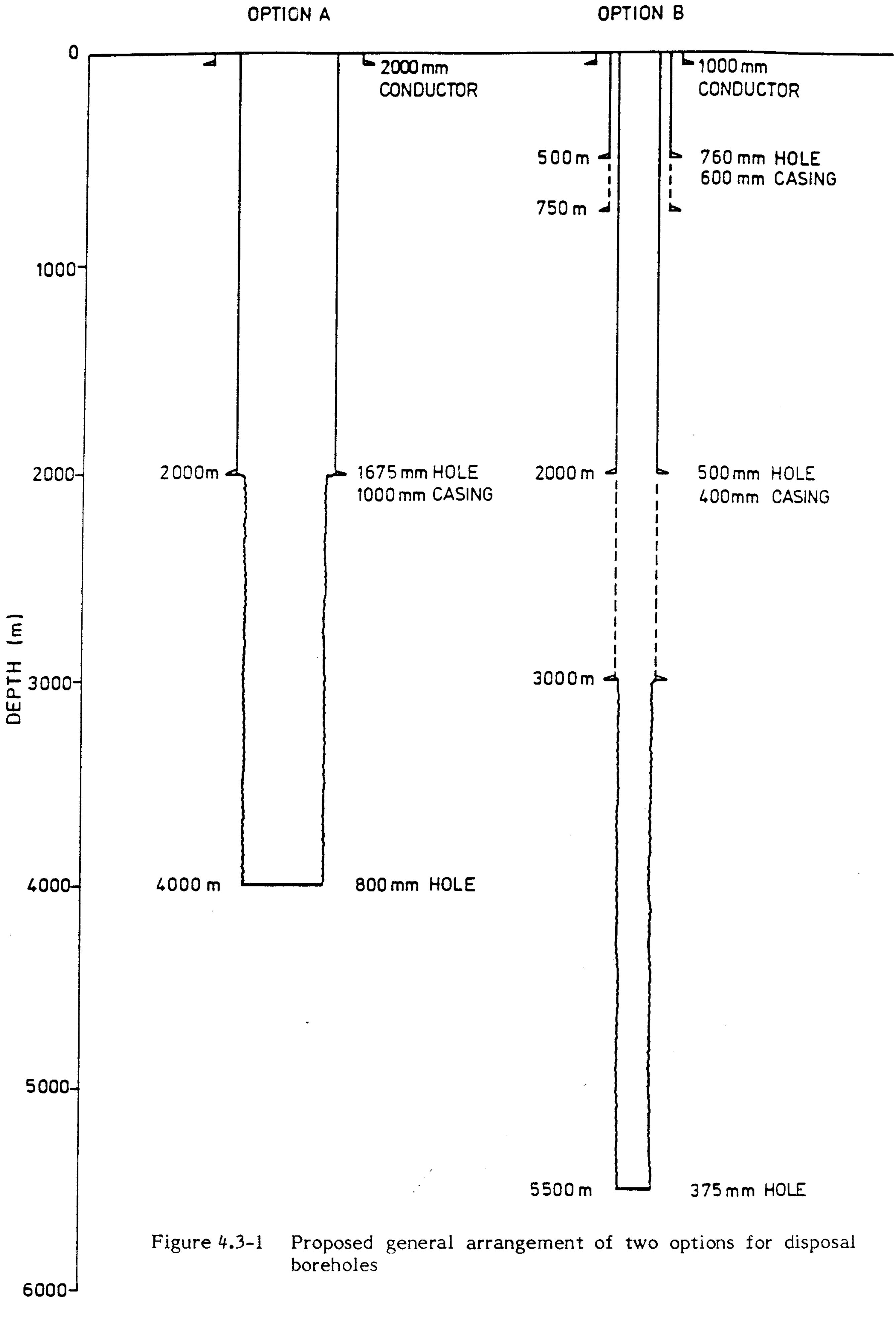


Figure 4.3-1 Proposed general arrangement of two options for disposal boreholes

4.4 Large diameter borehole construction

4.4.1 Hole sizes

For any deep drilling application, some form of initial entry conductor arrangement is necessary. The conductor would normally be installed prior to the mobilisation of the deep drilling equipment in this class of work. The size of conductor should be consistent with the diameter of the top section of drilled hole and will act as a guide for the drilling assembly during the first drilled section. For the large diameter option, the proposal is for a 1375 mm (54 in) diameter hole for the first drilled interval. The conductor tube needs to be a minimum of about 1800 mm internal diameter. For the purpose of this intial design, a nominal 2000 mm diameter conductor has been proposed.

For the intermediate hole from the base of the conductor to 2000 m, the hole diameter proposed ideally would be 1675 mm, but in order to reduce the annulus area to effect a high quality seal, the hole size should be reduced to 1300-1400 mm (51-55 in), say 1375 mm (54 in). For the interval through the deployment zone, the proposed diameter is 800 mm (32 in). Amendment to sizes may be necessary after the detailed design, but the outline proposals presented here are indicative of the likely suite of sizes.

4.4.2 Casing sizes

Other than the conductor pipe, only one primary 'casing' is proposed as part of the actual drilling operation. This casing will be installed in the 1375 mm hole to 2000 m and it is proposed that this casing should have an internal diameter of 1000 mm. This scale of operation has been achieved before when 1860 m of 1375 mm casing was run in July 1969 in the Amchitka Islands, Alaska. The casing weight was 1820 tons and the wall thickness was 64 mm (2.5 in).

On completion of the drilling, it is proposed to install a liner through the deployment zone suspended from the lower part of the primary casing just above 2000 m. For the purpose of this study, it has been assumed that the internal diameter of the liner which will be necessary through the deployment zone will be such as to give an internal diameter of 600 mm (24 in) allowing the deployment of a nominal 500 mm (20 in) canister string.

4.4.3 Drilling bits

For the large diameter case, the proposal is to use a conventional flat bottom bit body design and hard formation tungsten insert cutters and drill the hole intervals in a single pass. Advances in drilling bit and associated assembly design was stimulated in the 1960's by the US Atomic Energy Commission programme of deep large diameter holes. The development of the plate bit in the 1960's came about in an attempt to solve problems associated with hole verticality control. The flat bottom or plate bit coupled with the use of heavy collar assemblies have proven to be effective in maintaining verticality. Bits up to about 4500 mm diameter have been used and designs for bits up to 6000 mm have been prepared.

The use of flat bottom bits for drilling deep, large diameter holes in granite has not been demonstrated to any great degree. The success will partly depend on the achievable weight-on-bit which will be dictated by the drilling rig capacity. Wear rates of the cutter bearings and teeth are also important and to some degree unknown for a deep granite application. In small diameter hole drilling in granites, cutter wear is less of a problem than bearing wear, but developments in bearing design in recent years have provided bits with consistent performance.

Development of bit bodies and cutters is continuing for other large diameter hole applications and if this proposal was be pursued to a field implementation stage, lead time must be allowed for specific bit and cutter design and manufacturing. The implementation of deep shaft projects in the period up to the year 2000 will promote further bit development and hopefully give the necessary confidence that a large diameter hole option is feasible at economic cost.

4.4.4 Drilling assemblies

The proposals for drilling the large diameter hole would be to use a conventional 'big hole' assembly comprising a flat bottom bit, roller reamer stabilizer, drill collar mandrel with cast iron, split interlocking, doughnut weights and a top roller stabilizer. A typical assembly for a large diameter hole is shown in Figure 4.4-1.

It is common practice to drill with about 60% of the available drill collar weight, but this may be increased to optimize the available weight for the larger hole interval.

This type of assembly provides a very good pendulum effect and in other large diameter hole applications, verticality problems have virtually been eliminated with this arrangement.

4.4.5 Drillstring

For the large diameter option, it is proposed to utilise a string of 13-3/8 in (340 mm) diameter drill pipe with 16 in (405 mm) connections in 9.1 m lengths. This size and construction of drill pipe has become a standard for large diameter holes and a string was recently manufactured for the Basalt Waste Isolation Project in the USA.

4.4.6 Drilling fluid system and solids control

Several drilling fluid circulating systems have been considered. These include:

Air

Only relevant for very shallow applications or dry holes. Reverse air circulation requires a rotating pack off device between the surface casing and the drill pipe. This system has been used effectively in small diameter holes. The problem with air drilling in large diameter holes is that air is not a good transport medium as the density and viscosity is so low. If water is encountered, compressor costs become prohibitive.

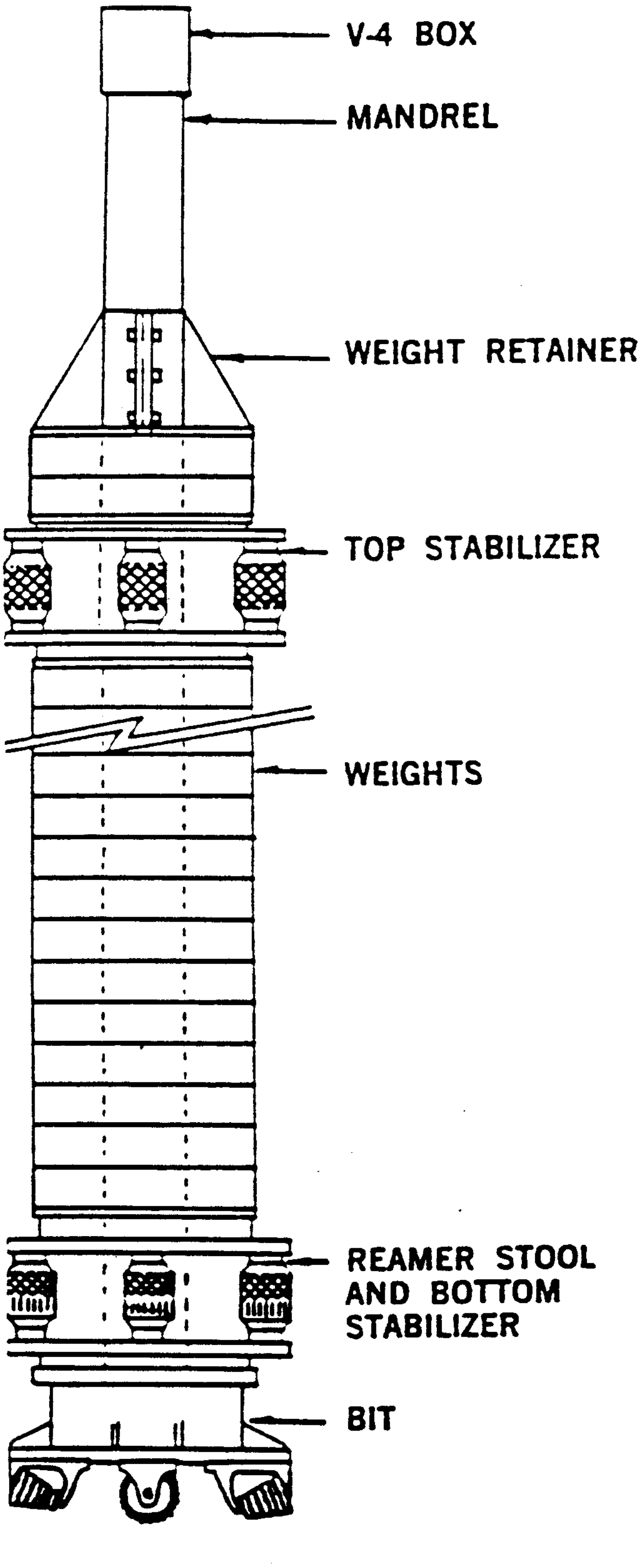


Figure 4.4-1 Typical large diameter hole drilling assembly

Direct mud circulation

This is the normal system used for oil and gas drilling and involves pumping the drilling fluid down the drillstring, across the bit face and back up the annulus to remove the cuttings. This is an effective system in small diameters, but because of the large cross-sectional area in a large diameter hole, the ascending velocity is extremely low and in order to transport cuttings effectively, the particles must be very fine and the gel strength of the drilling fluid must be relatively high.

Reverse air assist circulation

This is the most common circulating system in use for large diameter hole drilling. In this system, the drilling fluid circulates down the annulus between the drill pipe and the hole wall (or casing), across the bit face and back up the drill pipe. Circulation is achieved by suspending an air string to some depth within the drill pipe to aerate the mud column from the point of injection back to the surface, creating a pressure imbalance and hence inducing a flow across the bit face and up the drill pipe.

Jet circulation

This system is in two forms. Both require concentric tubes for drill pipe. The concentric tubes can either be formed by running a string of casing independently inside the drill pipe or by the use of purpose made dual concentric drill pipe lengths. In either case, drilling fluid is injected into the annulus within the drill pipe to flow to the bottom of the drilling assembly and is discharged at the bit face through nozzles. The system uses air assist to maintain circulation. There are several variations of this system.

The most efficient and economical circulating system for large diameter drilling is considered to be reverse air assist. As the depth of the hole increases, a joint of air line is added until approximately 100 m of air line is suspended within the drill pipe. No additional pipe is necessary. The aeration of the upper 100 m of drilling fluid inside the drill pipe serves to create the required differential pressure for flow to occur. The circulating circuit is then from the mud pits or tanks by gravity flow into the borehole annulus, down the annulus, across the bit face, back up the drill pipe and then to the mud pits or tanks through whatever solids control system is used.

The reverse air assit circulating system for a single pass borehole construction is illustrated in Figure 4.4-2.

As a guide, the introduction of an air flow of 85 m^3/min at a pressure of 1.4 MPa is sufficient to induce a flow of drilling fluid at the rate of 15 m^3/min (66 gal/min). Flow rates will vary depending on the drilling fluid properties and the exact depth of submergence of the air line.

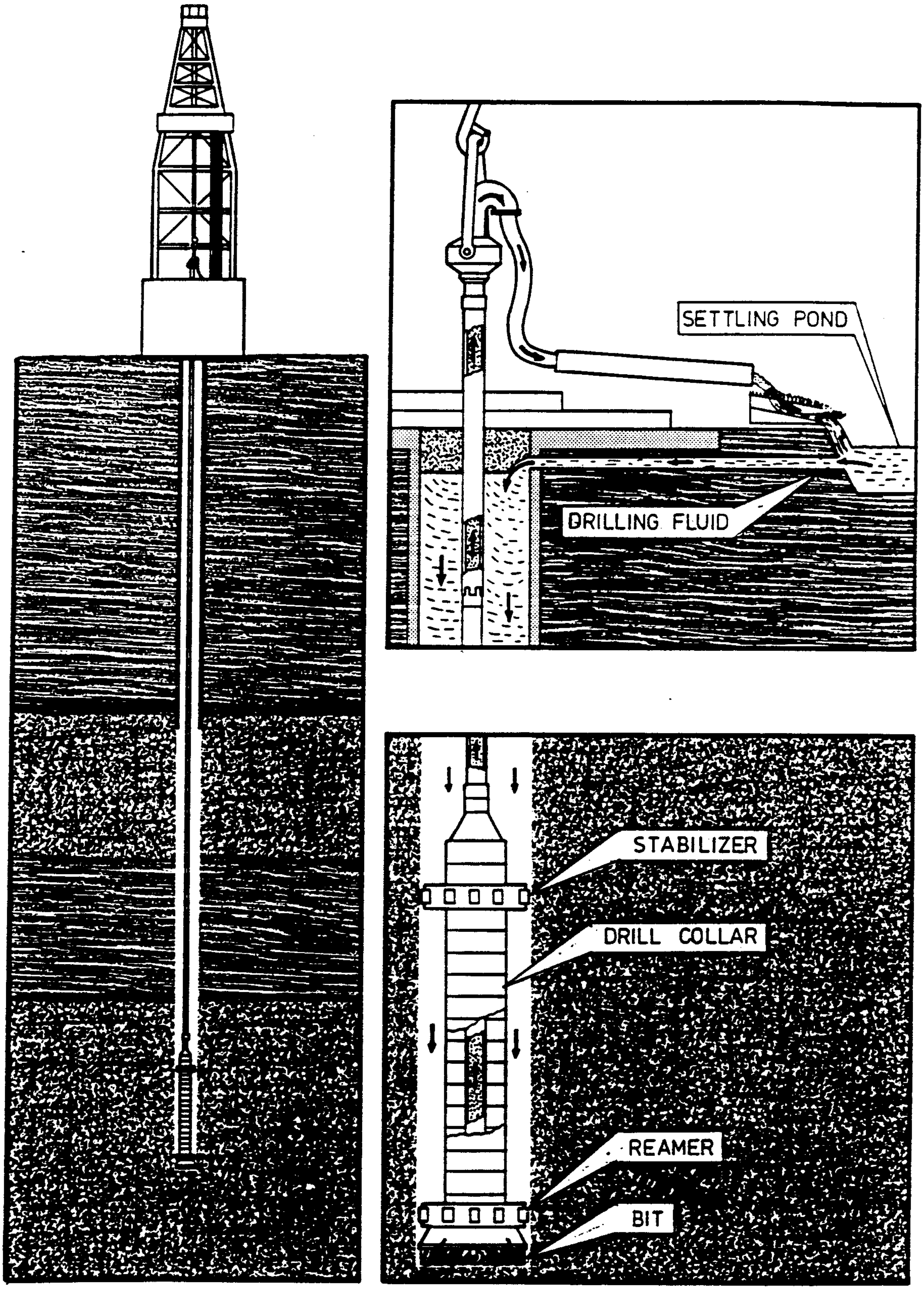


Figure 4.4-2 Diagram showing single pass air assist reverse circulation method

There are a number of advantages with the reverse circulation system including the velocity that can be achieved in the returning drilling fluid within the drill pipe which is sufficient to have good carrying capacity for cuttings. Another is that the velocity in the annulus is so low that there is virtually no tendency to hole erosion and the frictional pressure loss within the annulus is insignificant. The combination of virtually total hydrostatic head in the annulus and extremely low velocity give good support for the borehole wall which is important if fractured rock is encountered or there is tendency for rock to spall from the borehole as stress induced relaxation increases with depth and breakout. Support can be increased by increasing the fluid density in the normal way and the drilling fluid can be tailored to assist in the control of borehole wall instability.

In addition, this system requires no rig pumps for normal drilling.

In general, for this application to the depths envisaged, an inexpensive, light gel fluid using bentonite will probably be adequate and will be very cost effective.

Two options can be considered for the surface circulating system either mud pits formed by excavation or steel tanks.

Site conditions may be the governing factor. Mud pits offer simple separation of drilled solids by settling. A steel tank system will be required if a solids control system is used. Such a system would include shakers, hydrocyclone desanders and desilters and possible centrifuges. A steel tank system with solids control equipment would require less fluid and be easier to operate in winter conditions. Combination systems of steel tanks with solids control equipment and mud pits for further separation and storage can also be considered.

4.4.7 Primary casing design

The study has addressed many aspects of deep hole design in granite and questioned the conventional design for boreholes of this scale. It is assumed that the geology and pressure regime at the selected site will be well known from prior investigation and that a pressure control problem does not exist such that there will be no requirement for blow out preventers. This assumption leads to the proposal that the primary casing should be perforated or formed from a high void ratio material to facilitate the proposed sealing process using pre-compacted bentonite (see Chapter 5).

In order to appreciate the scale and design procedure for a casing of this kind before addressing the novel proposal using different materials and a high void ratic wall, the proposal for a standard steel design is presented. If this was a conventional borehole for say mining applications, the 1000 mm internal diameter casing to be set at a depth of 2000 m is of small enough diameter to make the choice between a heavy wall unstiffened and ring stiffened casing difficult. Cobbs Engineering have used their proprietary computer program to investigate the optimum casing design for a steel solid casing for this application. The casing design can be optimized on the basis of weight or cost. To achieve minimum weight, the casing needs to be ring stiffened thin shell casing, but this minimum weight is achieved at the cost of more expensive fabrication in the installation of stiffening rings.

For the preliminary design and cost estimate, Cobbs selected a combination of stiffened casing which yields the minimum cost based on current fabrication experience in the USA. A tabulation of the various casing sections from the first unstiffened section to a depth of 2000 m is presented in Table 4.4-1.

In this preliminary design, it was assumed that the casing would be rolled to a 1% out-of-roundness tolerance which is the requirement for both the American Petroleum Institute (API) and the American Society of Mechanical Engineers for pipe and pressure vessels. The material of construction has been assumed to be a steel equivalent to ASTM 588 or ASTM 441, both of which have a minimum yield strength of 50 000 lb/in^2 (7140 kPa). The steels are esstentially the same, however, the ASTM 588 steel is produced in greater wall thickness than the ASTM 441 steel. The steels have been commonly used in the fabrication of large diameter shaft casing. The casing was designed with a factor of safety of 1.5 against a normal pressure grdient of water with the strength of the casing being reduced by the assumed out-of-roundness. This is common practice in the design of steel casings for drilled shafts.

It is assumed that it will be impossible to deliver full 30 m long casing lengths to the site. To minimize rig time, it would be the intention to make up 30 m long joints from sections at the site.

The most common casing installation practice is to use semi-automatic welding procedures such as metal inert gas using either bare welding wire or flux cord welding wire. The practice is to use as many welders as can be conveniently grouped around the casing to minimize total elapsed time. For the 1000 mm casing, it should be possible for four welders to work simultaneously and the usually preferred procedure is to use bare wire welding to eliminate weld flux clean up.

Following various discussions on materials and concepts, it was decided to investigate a novel design based on the use of a high void wall casing in a copper alloy to complement the canister design and to provide for effective sealing throughout the borehole. Firstly, the assumption that no pore pressure problems were likely to be encountered and the fact that the granite will probably support itself for the most part and with minimal support elsewhere, means that a solid casing can be dispensed with. The design study then considered possible materials and the construction and installation of a solid rib and high void ratio panel concept as well as a rolled pre-perforated plate construction to provide the necessary strength for installation and support with acceptable ovality. The objective is to give a high potential for movement of bentonite through the walls in the sealing process. A titanium casing will be considered if the deployment canisters also will be of a titanium alloy.

The investigation into this novel approach was tentative and addressed two materials and a liner size suitable for the deployment zone. The details are described below in the section dealing with the liner. However, the conclusion of this simple investigation is that a suitable ring and stringer construction or rolled perforated plate walls for casing is a feasible proposition. The design and construction would follow the same lines as indicated for the standard solid steel casing noted above.

TYPICAL STEEL CASING DESIGN

FOR RING STIFFENED CASING

Table 4.4-1

DEPTH TOP METERS	DEPTH BOTTOM METERS	WALL THICKNESS MM	STIFFENER MMXMM	BAY LENGTH MM	UNIT WEIGHT Kg/M	JOINT WEIGHT TONNES	CUMMULATIVE WEIGHT TONNES	UNIT COST $/m	JOINT COST $x1000	CUMMULATIVE COST $x1000
1920	2000	65			1601	128	128	3523	282	264
1680	1920	60			1478	355	483	3252	780	1044
1440	1680	55			1355	325	808	2981	715	1760
1200	1440	50			1232	296	1104	2710	650	2410
960	1200	45			1108	266	1370	2439	585	2996
810	960	40			985	148	1517	2168	325	3321
750	810	30	75x75	450	1111	67	1584	2110	127	3447
720	750	30	75x75	600	1023	31	1615	2001	60	3508
690	720	35			862	26	1641	1897	57	3564
660	690	30	75x75	900	936	28	1669	1872	57	3621
630	660	30	75x75	1050	911	27	1696	1861	56	3677
600	630	30	75x75	1350	878	26	1722	1820	55	3732
570	600	25	75x75	600	891	27	1749	1714	51	3783
540	570	25	75x75	900	804	24	1773	1605	48	3831
510	540	25	75x75	1050	780	23	1797	1574	47	3878
480	510	25	75x75	1350	747	22	1819	1533	46	3924
450	480	25	75x75	1650	726	22	1841	1507	45	3970
420	450	20	50x75	450	731	22	1863	1448	43	4013
390	420	20	50x75	750	640	19	1882	1311	39	4052
360	390	20	50x75	900	617	19	1900	1277	38	4091
330	360	20	50x75	1350	579	17	1918	1220	37	4127
300	330	20	50x75	1650	565	17	1935	1199	36	4163
270	300	20	50x75	2100	552	17	1951	1179	35	4199
240	270	15	25x75	600	460	14	1965	1016	30	4229
210	240	15	25x75	900	432	13	1978	952	29	4258
180	210	15	25x75	1200	418	13	1991	920	28	4285
150	180	15	25x75	1800	403	12	2003	889	27	4312
90	150	15	25x75	2100	399	24	2027	800	53	4365
30	90	15	25x50	2100	391	23	2050	873	52	4417
0	30	15	25x25	2100	383	11	2062	867	26	4443

4.4.8 Cementation of primary casing

In a conventional borehole with steel primary casing, the cementation would comprise a neat Portland API Class H cement with a slurry density of 15.6 lb/gal. This is approximately equivalent to ASTM Type 1 or British Standard ordinary and rapid hardening Portland cements to BS12. This is an expensive approach even for conventional shafts.

The actual cement or grout design usually incorporates a number of features which may be desirable or more economical than this neat cement proposal. Use of extenders, such as bentonite, to increase the volume yield or silica flour to prevent strength regression resulting from elevated temperatures can be considered. The use of bentonite as an extender will of course reduce the strength of the cement, but this is sometimes acceptable in conventional shafts.

The primary purpose of cementing casing in place is to support the casing in the shaft and obtain both mechanical and hydraulic bonding between the casing and the borehole wall.

The cement is usually placed through grout lines run inside grout line guides attached to the exterior of the casing. A typical arrangement is illustrated in Figure 4.4-3. This placement method is analogous to the use of tremie placement or concrete or grout beneath water.

The cement is generally placed in multiple stages with the first stage serving to anchor the casing and prevent it being floated from the hole by the cement. After each stage when the grout has developed it's initial set, the following stage is palced. For the 2000 m of 1000 mm diameter casing, the cement volume is estimated as 17 000 ton or 12 500 m^3 if a conventional steel casing cemented to surface were to be used.

Quality assurance of cementation of deep borehole casings is of concern and this is another reason for questioning whether the conventional approach to casing could be improved for this special application. The revised proposal, using high void ratio casing, will not necessitate cementation, other than the possibility of a small seating cement. The casing will be given some lateral support by centralisers or standoffs bearing on the drilled borehole wall.

In view of relative weakness of this proposed primary perforated casing and the proposed construction and material type, it may be necessary to install a temporary protection casing during the subsequent drilling of the lowermost section of the borehole through the deployment zone. This temporary casing would be in steel and have an internal diameter of about 850-900 mm (33.4 - 35.5 in). Flush or tapered connections will be necessary to the outside to facilitate removal after drilling without damage to the permanent casing. It should be possible to reuse at least part of this string for subsequent boreholes after inspection.

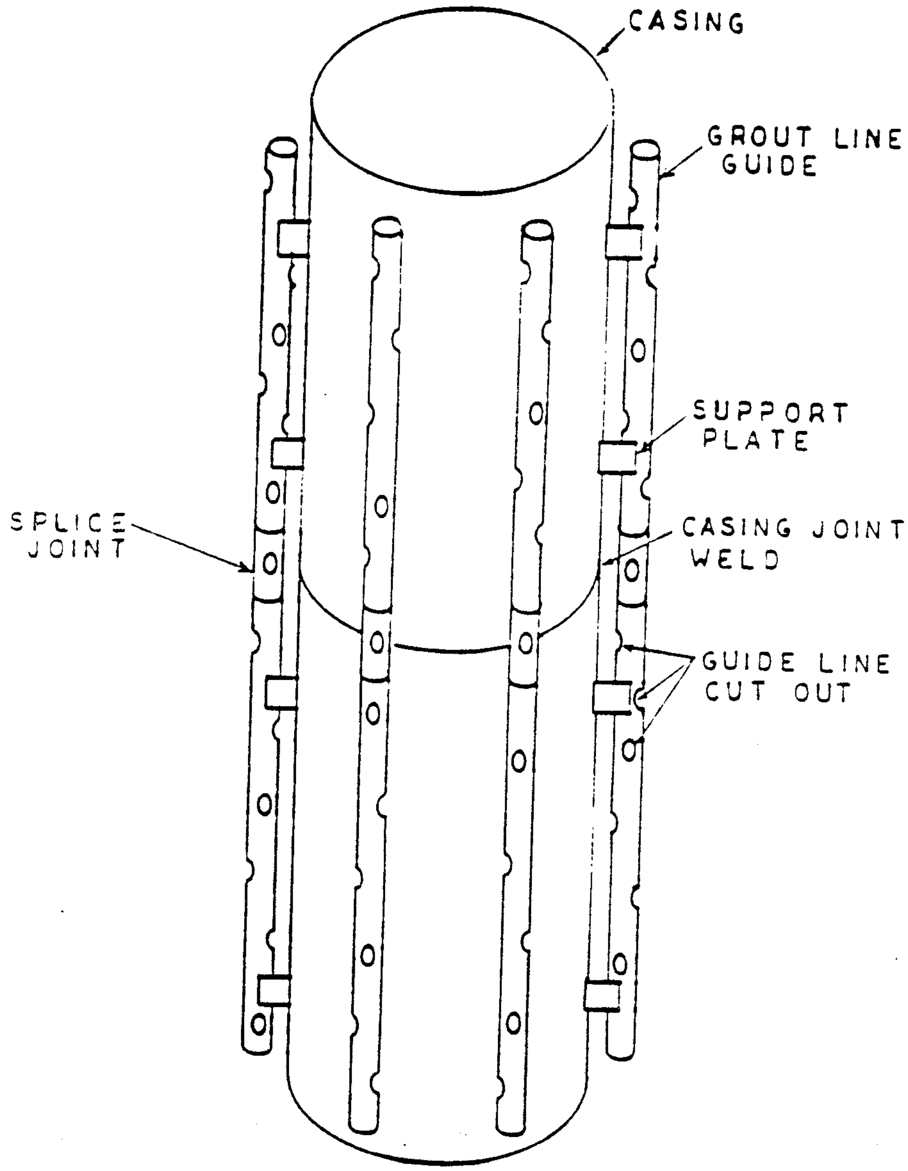


Figure 4.4-3 Typical grout line arrangement for casing cementation

4.5 Small diameter borehole construction

4.5.1 Hole sizes

The small diameter option is similar to the Gravberg-1 deep gas exploratory borehole drilled in Dalarna except that hole diameters are larger. The hole and casing sizes for the proposed general arrangement are shown on Figure 4.3-1. The proposed intervals for the deeper of the two options, Option B, are listed below together with the actual Gravberg-1 hole and casing sizes for comparison.

	Proposed borehole	Gravberg actual
Conductor hole	Mined to 30-50 m 1000 mm conductor	Mined to 10 m 508 mm conductor
Surface hole	Depth 500-750 m Diameter 760 m Casing 600 mm	Depth 1250 m Diameter 445 mm Casing 340 mm
Intermediate hole	Depth 2000-3000 m Diameter 500 mm Casing 400 mm	Depth 4167 m Diameter 311 mm Casing 245 mm
Lowest section*	Depth 5500 m Diameter 375 mm	Depth 5799 m Diameter 216 mm

For Option C, the shallower small diameter hole option, the intermediate hole would be limited to say 2000 m and the lowest section through the deployment zone would be similar in depths to those proposed for the large diameter option, Option A.

The Gravberg-1 well was further cased at 5799 m with a 197 mm liner installed to 5453 m and then drilled on to a final depth of 6957 m in 165 mm diameter. To the depths of interest in this study, the final hole diameter at Gravberg was some 58% of the proposed borehole for waste deployment. This has implications on the time required for drilling costs for drilling equipment and casing, and the potential problems below about 4000 m.

4.5.2 Casing sizes

The casing sizes listed above are nominal and would be sized to try and use standard sizes wherever possible, particularly for bits. Some oilfield casing sizes related to the nominal hole dimensions proposed are presented below:

Outside diameter		Drift diameter		Weight		Hole size	
in	mm	in	mm	lb/ft	kg/m	in	mm
30	762	27.813	706	310	462	26	660
		28.313	719	235	350	26	660
24	610	21.313	542	304	453	20	508
		22.937	583	113	169	20	508
		23.039	585	101	150	20	508
20	508	18.188	462	169	252	17.5	446
		18.563	472	133	198	17.5	446
		19.003	483	90	134	17.5	446
16	406	14.000	356	146	218	12.25	311
		14.250	362	128	191	12.25	311
		14.500	368	109	163	12.25	311*
		15.011	381	70	104	14.75	375
		15.209	386	53	79	14.75	375

* Probably can be obtained with 375 mm drift.

Other special sizes and grades are manufactured by the oilfield tubular goods suppliers such as Mannesmann ('Big Omega' series).

This means that if standard steel casing was used, the suite may be as follows:

Surface string: 610 mm (24 in) with drift diameter 508 mm (20 in)

Intermediate string: 406 mm (16 in) with drift diameter 375 mm

This would permit a 375 mm hole to be drilled below the intermediate casing. The liner through the deployment zone would have to be either a special clearance 340 mm (13.375 in) or 298 mm (11.75 in) with 311 mm (12.25 in) and 273 mm (10.75 in) internal diameters respectively.

The concept of high void ratio casing could also be considered for the intermediate string and the liner as outlined above for the large diameter option. However, the limited available diameter most likely not more than about 275 mm severely restricts the scope for waste disposal. This is one of the reasons why the small diameter options are not favoured.

4.5.3 Drilling bits

Standard bit sizes should be adopted wherever possible, although hard formation bits in the sizes required are unusual and will require special order. Lead times of nine to twelve months should be expected. The suite of bits sizes proposed for the small diameter option are:

Bit size		Bit type
in	mm	
30	762	Plate bit would probably be more suitable
20	508	Roller cone
14.75	375	Roller cone

The large diameter bits, 762 mm (30 in) will probably only be required to drill a surface hole as far as a competent granite and hence can use steel tooth cutters unless extensive boulder beds, weathered granite or overlying sedimentary rocks are expected. The site selection should endeavour to identify sites where the granite is close to surface to avoid a long length of large diameter hole.

4.5.4 Drilling assemblies

Drilling assemblies will be similar to those used for the Gravberg-1 well with a near bit roller reamer and two string roller stabilizers being the normal assembly to maintain hole alignment and straightness. Pendulum assemblies will be necessary to help maintain verticality. Large diameter drill collars will be desirable to optimise the effect of a pendulum assembly.

4.5.5 Drillstring

In view of the large diameter drilling required for this option, by oilfield standards, it would be advisable to utilise a string of 6-5/8 in (168.30 mm) 25.20 lb/ft (37.58 kg/m) API drillpipe for the majority of the drilling and at least to the top of the deployment zone at 2000-3000 m. Thereafter an option would be to use standard 5 in (127 mm) 19.5 lb/ft (29.08 kg/m) API drillpipe as was used in the Gravberg well until the 7.75 in (197 mm) liner was run at 5799 m.

4.5.6 Drilling fluids and solids control

Much has been said and written recently about drilling fluids for deep drilling in granite, especially in relation to the Gravberg-1 deep gas well where various systems and mud densities were used. The debate on fluids is related to the control of wellbore stability in a strong, anisotropic stress environment. There is a view that very deep drilling in granite in such conditions should only be contemplated with oil based muds. For this application, however, where the desire is to use bentonite based materials for containment and sealing, it would be desirable to use bentonite based muds if at all possible as this permits the construction of the sealing system with bentonite based materials throughout the drilling and deployment phase.

However, although the drilling conditions may be better in other areas of Sweden compared with the area of the Siljan impact structure, the problems encountered at Gravberg are a warning that very deep drilling below say 4000-5000 m may be particularly difficult and necessitate oil based muds for lubrication and fluid loss control reasons. This is another reason why a deep solution is less attractive than the shallower options.

Fluid circulation would be achieved by normal circulation using rig pumps.

A conventional mechanical solids control system comprising shakers, desanders and possibly centrifuges with the ability to allow further separation in reserve mud pits would be appropriate for these options.

4.5.7 Casing design

In a normal drilling programme, the design of the various casings and liners would follow well known procedures. As there is no pressure control considerations in this case, the detailed design should be straightforward. However, for the large diameter option, a high void ratio casing wall is proposed to allow a sealing procedure based on precompacted bentonite to be used. This type of casing construction using materials such as copper alloys should be feasible for the intermediate casing and the liner through the deployment zone with the small diameter options. The surface casing could be solid or perforated. Some lateral support for these casings would be provided by centralisers and standoffs.

4.5.8 Cementation of casing

Unless a solid casing is adopted with the necessity to adequately seal the annulus, cementation would be unnecessary except perhaps over a bedding interval at the based of the string. Any solid casing could be cemented back to surface if desired using normal oilfield displacement methods. Cementation quality is always questionable due to channelling and contamination by mud stringers. For this application, therefore, a solution without cementing with perforated casing seems the preferred method.

4.6 Deviation control

The comments made in the Interim report Stage B about verticality control remain valid. Good verticality control is essential if the boreholes, small or large diameter, are to reach their target depths and be usable for the proposed purpose. The drilling system must incorporate a regular and effective surveying programme.

For the large diameter option, verticality control will probably be easier than for the small diameter options. This is because the drilling assemblies comprise very heavy short collars such that a good pendulum effect can be achieved. Large diameter drilled holes are usually constructed for mine shafts and good verticality is necessary for mine hoists. Hence there is good experience in this important area.

For the small diameter option, verticality control is not likely to be a problem to 4000 m providing care is taken during drilling and surveying is carried out regularly to monitor any tendencies to deviate. It is essential that borehole trajectories are maintained as near vertical as possible. This may require less than efficient drilling from time to time to re-establish verticality resulting in less than optimum penetration rates and bit life. At depth, below 3000-4000 m, there may be an increasing tendency to deviate which will be exacerbated by rock stress breakout effects. The experience at Gravberg should not unduly influence the view about verticality control in small diameter boreholes at depth in the granite, but it does illustrate the need for special caution at depth and the merits of limiting the depth of these proposed disposal boreholes to about 4000 m.

For the smaller diameter option, as large a diameter of drill collar as is possible to accommodate, while still allowing effective fishing potential, should be used to give a short, stiff assembly. For example, in the lowest section where the hole size is 375 mm (14.75 in), a drill collar diameter of 280 mm (11 in) could be considered if verticality control becomes a problem.

In both large diameter and small diameter options, surveying should be carried using at least single shot surveying equipment to give both inclination and azimuth of the borehole track. This is important if tendencies due to rock stress breakout effects, geological bias or other reasons are to be monitored effectively and the appropriate corrective action taken. More sophisticated surveying or measurement while drilling instrumentation should be considered and may be readily available for this kind of application in the next few years.

In general, good borehole verticality control is considered to be achievable using both oil and shaft drilling methods to 4000 m depth. Where stress breakout becomes more influential with depth, the maintenance of verticality may be more troublesome and hence a shallow option is preferred to a deeper solution. In the event that deviation becomes a problem the matter can be correlated, but at added cost to the drilling.

4.7 Coring, logging and testing during drilling

An important aspect of this project will be to develop a strategy for obtaining scientific information through the ground penetrated, or at least through the section for deployment of the waste material, in all holes drilled for disposal purposes. This is in addition to a comprehensive site investigation borehole or boreholes to the full depth of the facility, or even deeper, that will be required prior to commencement of the main construction of the facility. This aspect is discussed elsewhere. The purpose of the investigation during construction will be to verify the conditions on which the containment premises are based, identify any special features and problem areas which may necessitate a revision in the zone of deployment, and to provide a database of scientific information for the repository complex.

Such investigations during construction, however, is difficult to design in practice. For the purpose of this discussion, such activities as may be carried out are considered initially for the large diameter hole to 4000 m. Firstly, there is a question as to whether or not the degree of investigation need to be the same above the deployment zone as within it. On the basis that a substantial seal plug will be formed on the top of the canister string in the section from 2000 m to 4000 m, it could be argued that the primary target of such investigations should be within the deployment interval.

The methods for collection of scientific data are principally:

- Collection of returning chip samples and mud from the mud stream
- Coring
- Wireline logging
- Hydraulic testing
- Borehole seismics

With reverse air assist drilling, the collection of samples from the returning drilling fluid should be quite straightforward, but the potential value of the information is limited to identification of lithology changes and the assessment of changes in chemistry, or gases, providing the data are not overwhelmed by the properties of the mud itself.

Coring in granite can only be considered satisfactory where the process is similar to that used in the mining industry with small diameter, low area ratio bits. Such conditions cannot be achieved in a full size hole of this scale. Hence, it is proposed to carry out pilot coring ahead of the advancing shaft or borehole at intervals to provide core for analysis and a suitable hole for carrying out selective wireline logging (see below).

Such an operation would necessitate the installation of a temporary small diameter liner sealed at the base to allow a conventional heavy duty wireline coring system to be deployed in the borehole for the investigation process. Such equipment would typically drill a 159 mm or 123 mm diameter hole with 100 mm or 80 mm core diameter respectively. The wireline system would permit fast tripping of the corebarrel. The length of extension ahead of the advancing borehole 'face' is debatable, but extensive small diameter holes ahead of the main construction should be avoided and must at least have a trajectory within the main borehole envelope. The idea of drilling a slim cored hole from surface to final depth to be followed by the main borehole, or even in long sections, is not considered practical or desirable. The likely trajectory of the slim hole will be variable and of a different sensitivity of track change to that possible with a large borehole.

A suggested extent of advance drilling of 200 m is proposed at this stage limiting the section to be investigated to the deployment zone between 2000 m and 4000 m. This means that ten 200 m long intervals would be necessary per borehole adding about ten days for each 200 m section making a total time addition of approximately 100 days.

Details of a typical heavy duty wireline drilling system which could be adopted for this exploratory work as well as for the initial site investigation programme is included as Appendix 2.

Logging suites through the advanced cored holes should be designed to calibrate the rock quality and discontinuities against comprehensive classification information obtained during the initial site investigation of the proposed repository site. It should be possible to limit log runs to only those of direct relevance to the key issues of permeability and containment potential. It may even be possible to use water for coring and logging programmes carried out through the temporary technical casing to permit the use of such devices as the borehole televiewer which requires relatively thin, clean muds to give the necessary resolution. The advanced core hole size should be suitable for high resolution data acquisition using standard logging tools.

If such advanced investigation can be achieved with water through the technical casing, hydraulic testing with water to assess the transmissivity of the rock mass at low heads using rising, falling head or constant head tests would be quite feasible.

4.8 Waste deployment zone liner

4.8.1 Completion fluid

On the completion of drilling with a bentonite mud system, it is proposed that the bentonite mud should be displaced with clean Na bentonite drilling mud with a specific gravity of 1.15. It would be the intention to install a liner across the deployment zone through this clean mud. The bentonite would form part of the sealing system after deployment of the canisters.

4.8.2 Liner design

As mentioned above, some form of permanent liner through the deployment zone will be necessary to assure repeated re-entry into the hole during the preparation and deployment process. In addition to providing protection to the deployment interval, a high void ratio material is desirable to permit movement of bentonite based sealing products into the annulus to construct an effective barrier. For the purpose of this discussion, the remarks are principally directed towards the large diameter hole concept which is the preferred option, but the same principle could be used for the slim hole options.

In the preliminary study of this concept, Cobbs has investigated the possibility of constructing a nominal 600 mm liner of a brass alloy for deployment in a shaft 4000 m deep. The preliminary, stress analyses were performed using a brass with a yield strength of 3.28 gr/cc, 560 MPa and a density of, for both a ring and stringer liner and for a cylinder rolled from perforated brass plate.

Two alloy types have been considered to date. They are as follows:

Alloy	UNS	Density gr/cc	Cu	Ni	Be	Si	Cr	Al	Fe
Beryllium Copper	C17200	3.17	97	0.2	2	0.2		0.2	0.4
Ampcoloy	C18000	3.40	96.5	2.5		0.6	0.4		

Of the two alloys considered at this stage, the beryllium copper alloy is less dense and has a higher strength, but the Amcoloy 940 may prove a better choice due to lower cost and easier weldability. Titanium may be another possibility.

The high void ratio ring and stringer concept has assumed 600 mm internal diameter rings, 25 mm thick, giving an external diameter of 660 mm. The stringers were 25 mm x 25 mm bars, 1000 mm long attached to the rings on 30^{o} centres for 12 bars per ring. Inside the liner, a brass wire mesh was used with a 25 mm mesh size with 3 mm diameter wire. This concept is illustrated on Figure 4.8-1.

An alternate solution was analysed using a tapered, perforated rolled brass plate, 19 mm thick reducing to 6 mm thick, with 25 mm diameter holes spaced at 75 mm centres on 60^{o} staggered centres giving a void ratio of 8.6%.

These tentative analyses demonstrate that the construction of a suitable permeable liner is feasible and that the same method could be adopted for the upper primary casing to ensure that the upper sealing system is equally effective. However, this novel aspect of the construction will need a very detailed study of the optional designs of suitable liners, materials and installation methods. For example, there are several possibilities for the 'permeable' wall of the casing or liner including woven meshes, expanded metal, punched wrapped sheet and composite construction of rods and helical wraps such as used in standard water well screens.

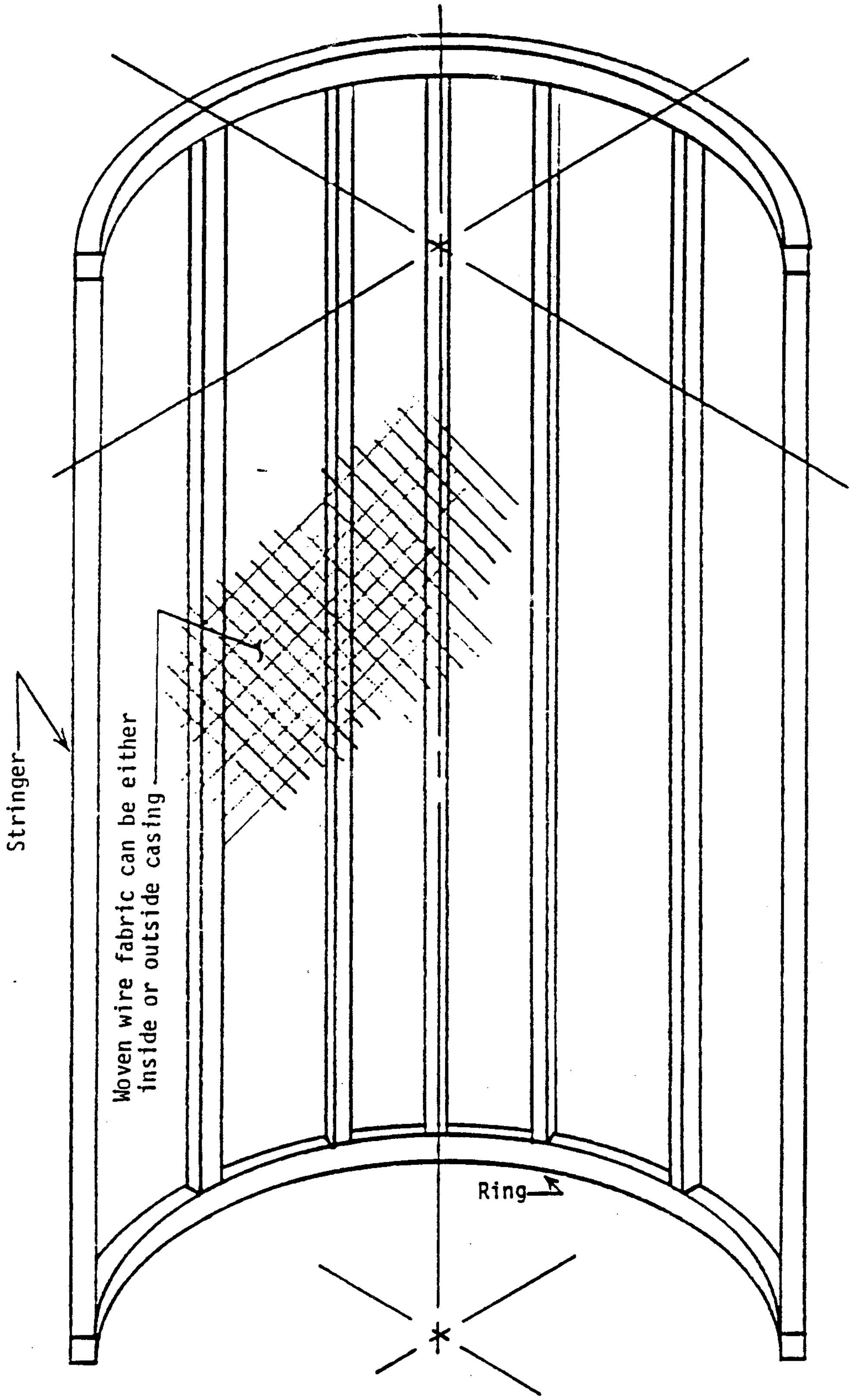


Figure 4.8-1 High void ratio ring and mesh liner concept

Obviously, the design of such an unusual lining component will require rigorous analysis and the construction of some model and even prototype lengths to fully evaluate the feasibility of the concept, but the initial indications are that a workable solution can be engineered for the liner requirement for this application.

4.8.3 Liner installation

Whether a large or small diameter option is adopted, it is envisaged that the liner through the deployment zone will be installed by lowering in sections. For the large diameter option, the connections would be welded on the drill floor. Screwed connection should be possible for the small diameter options. It may be practical to prefabricate long lengths of the liner at site to reduce the amount of rig time during installation, which is especially important if a significant welding exercise is involved, but this can only be assessed once the final design is available.

In the case of the preferred large diameter option, the proposal to install such a heavy liner of novel construction may necessitate the use of some innovative methods to reduce the stress on the liner and rig loads during installation. Weight reduction could be achievable by the use of a buoyancy system or the use of a running string. The latter would permit circulation if problems of installation were encountered, but would complicate the running procedure at the rig floor due to the need to run concentric strings. However, even if this was necessary, a running procedure could be developed and the liner lengths made to suite the running string lengths.

4.9 Canister deployment

4.9.1 Deployment mud

The various discussions during the study has helped focus on the integration of the drilling and completion aspects with those of the canister deployment process and final sealing system. The concept is to build the sealing system throughout the construction, firstly by the use of a bentonite based drilling mud, followed by displacement with a clean bentonite mud in readiness for the liner installation and deployment of the canisters.

The latest work has been directed to moving away from the necessity to run canisters through a long section of a 'heavy deployment mud'. Instead, it is considered entirely feasible to place a very dense bentonite based fluid with low permeability properties in the deployment zone over short intervals immediately prior to the installation of one or two canister sets. The plan would be to construct a special vessel to contain the thick mud and run this assembly into the borehole on drillpipe. The mud would then be extruded by pressure into the bottom of the available hole to form a dense medium which would be compatible with the precompacted bentonite to be deployed with the canisters and between canisters. The canisters would be deployed, appropriately centralised, and intermediate precompacted bentonite material placed as described by Pusch in the section on sealing. Following detailed discussions, Pusch has also addressed the options for this 'deployment mud' and demonstrated that acceptable low permeability characteristics can be achieved.

The process would be repeated in cycles of one or two canister lengths until the deployment was complete. Pusch suggests either two 4.4 m long canisters separated by a 1.0 m long bentonite pack or a string of four 4.4 m long canisters with a 1.0 m long bentonite pack. The shorter string is probably the better to maintain good control and allow additional bentonite to be introduced into the system.

4.9.2 Canister deployment

Canisters would be installed on drillpipe to the required depth. The assembly would incorporate a release mechanism and a heavy collar section to assist in the penetration through the thick deployment mud. A vibrator mechanism would also be included in this assembly to assist in the partial consolidation of the sealing system. Vibrator design needs addressing in detail, but a system energised by pumping from the surface should be practical. This possibility is made easier by the adoption of the thick deployment mud concept placed only over a short section of the bottom of the borehole.

With a borehole protected by the deployment zone liner, repeated re-entry should be assured and the process should be relatively straightforward.

Depth control will be important and both accurate string measurement methods and a computerized instrumentation system to monitor depths by aggregations of the movement of travelling equipment on the rig, even with redundant systems, would be necessary to be certain of the spacing measurements during this critical activity.

Pusch has suggested that deployment should be about 200 m per month. This means that it would take 10 months to fill a 2000 m large diameter hole.

4.10 Sealing

Sealing proposals are discussed by Pusch elsewhere. These proposals are developments from the earlier ideas and incorporate the amendments to the concept relating to perforated liners and deployment mud mentioned above. However, it is appropriate to include some comments on items not specifically covered by Pusch.

These comments are principally directed at the large diameter option, but apply equally to the small diameter options if all other aspects are acceptable.

4.10.1 Deployment zone

At the top of the deployment zone above the canister strings, it is proposed to install a length of precompacted bentonite up to the top of the section. This poses no special problems with respect to the drilling and completion aspects. This section will be protected with the high void ratio liner that extends to the total depth of the borehole which will freely allow movement of the sealing medium from the main hole to the annulus and rock.

4.10.2 Main seal

For the 'main seal', which in the large diameter hole case extends from some 2000 m to 500 m, the proposal is to construct the seal in two parts:

A: A short section at the bottom of the primary casing where the casing is removed by milling and precompacted bentonite is placed across the interval.

B: The remainder which is constructed in a similar way to the deployment zone with precompacted bentonite packs placed in thick deployment mud. Over this section, the casing would be similar to the deployment zone liner with a high voids ratio allowing free movement of the sealing medium from the hole to the annulus and rock.

For the lower part of the seal, provision can easily be made in the primary casing string for a section which is ultimately to be removed by milling. The remainder of the plan rests on the ability to design, construct, install and protect the special 'permeable' copper alloy casing during the remainder of the drilling.

4.10.3 Abutment

The proposal includes a top seal of asphalt (500 m to 250 m) and concrete (250 m to surface). It is suggested that the best method to adopt for this important, although largely cosmetic seal, is to treat the exercise as a mining operation. On completion of the main lower seal, a cement cap could be formed and the shaft dewatered for access by miners. The casing could then be anchored into the rock with bolts and the lowermost part removed to allow the base of the asphalt to abut the rock. Similarly, at the asphalt/concrete interface, the casing could again be removed and the hole undermined to give a positive mechanical cap to the system.

Through the sections where the casing is not removed, the casing could be backgrouted using standard tunnelling pressure grouting methods. Alternatively, further sections could be removed as the plug is constructed to give direct contact with the rock.

The concrete through the upper part of the top seal should be high quality structural concrete placed with civil engineering equipment under high quality control.

The treatment of the upper seal as a mining operation would also assist in overall programming in that the drilling rig used for the drilling and deployment phases up to the top of the lower seal could be moved to the next location and the final seal carried out concurrently with the initial drilling rig on the next borehole.

4.11 Rig sizing and availability

A tentative specification for the drilling rig and ancillary equipment required for a large diameter option is presented in Appendix 3. For the small diameter options, the rig requirements would be similar to those for the large diameter option for the 6000 m case, but somewhat smaller for the small diameter 4000 m deep option.

Drilling rigs of this size are available. However, if this concept were to be adopted, the commencement of the field programme will be many years hence and it is likely that further deep drilling and large shafts will be constructed in the intervening years and the range of equipment available may be different and much improved. The rig size used for the Gravberg-1 deep gas well, DDP 20, is somewhat light for the large diameter option, although it could be modified.

4.12 Time schedules and costs estimates

4.12.1 Estimated time schedules

An estimated time schedule for the actual drilling operation for Option A, the large diameter 4000 m deep borehole, is shown on Figure 4.12-1. This indicates that the drilling will take about 435 days. Figure 4.12-2 shows the effect of investigation coring, logging and testing at 200 m intervals through the deployment zone from 2000 m to 4000 m. This adds an estimated 100 days to the programme. The deployment and sealing operation has been assumed as requiring 365 days to complete which should allow sufficient time to place the necessary seals. This may be too short if long periods are required to allow the bentonite sealing medium to partially consolidate.

For the purpose of this study, this approximate estimate serves to illustrate the likely order of time and hence permits the calculation of a tentative cost estimate.

For Option B, the small diameter option to 5500 m, the time will be somewhat reduced to about 320 days drilling time plus the 165 days deployment time allowance and a further 100 days provision for the investigation activities. A typical drilling time curve for this option is presented on Figure 4.12-3 plotted together with the actual time data from the Gravberg well to the same depth. The longer time indicated for the waste disposal case is because of the larger diameters involved and the need to allow sufficient provision for verticality control.

If Option C was considered, this would reduce the drilling time to about 200 days. The rest would be similar to Option B.

In summary, the times suggested at this stage are:

Option A: Large diameter to 4000 m

Drilling	435
Investigation	100
Deployment	365
Total	900 days

Option B: Small diameter to 5500 m

Drilling	320
Investigation	100
Deployment	365
Total	785 days

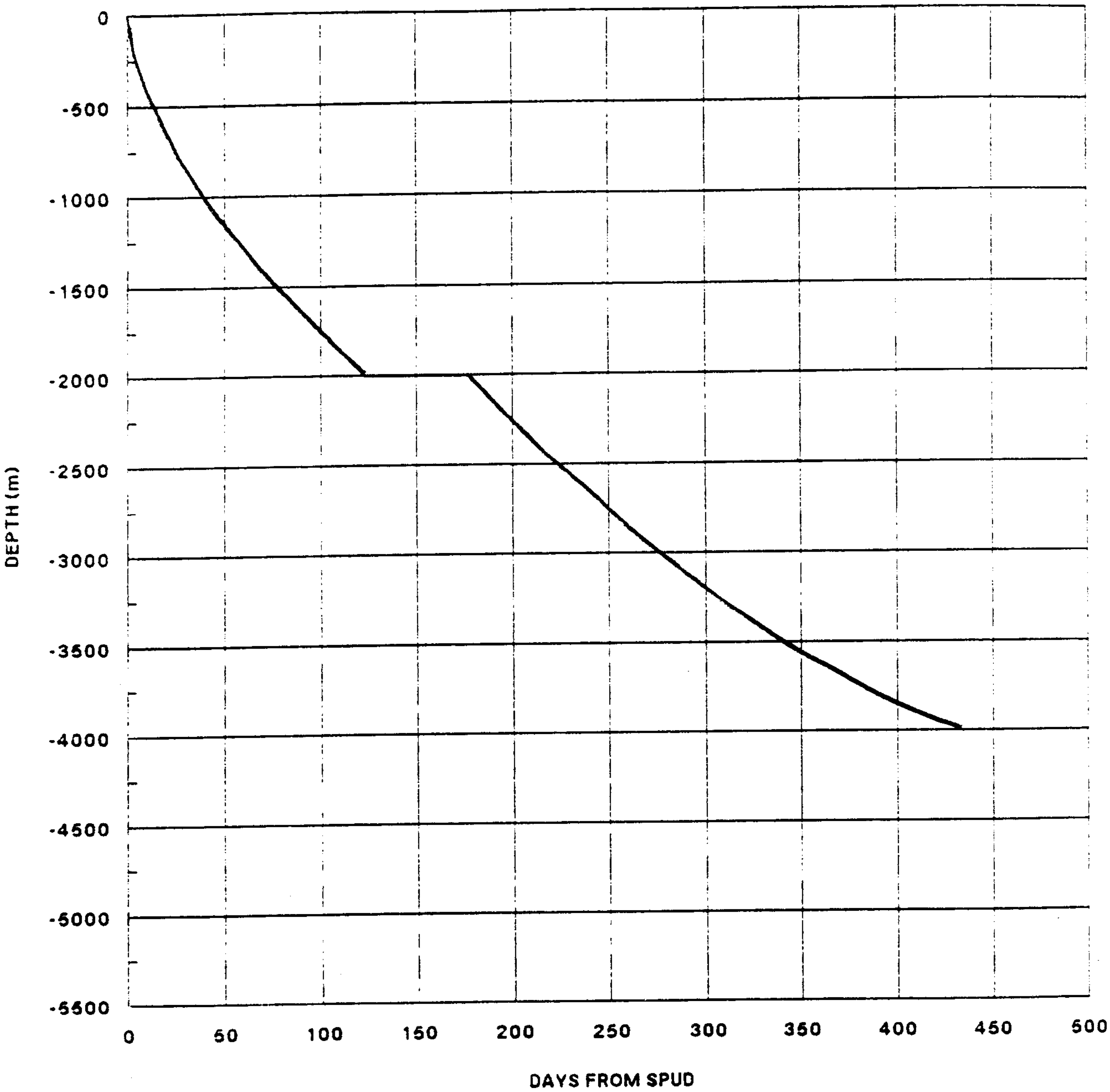


Figure 4.12-1 Time V depth for Option A, large diameter borehole

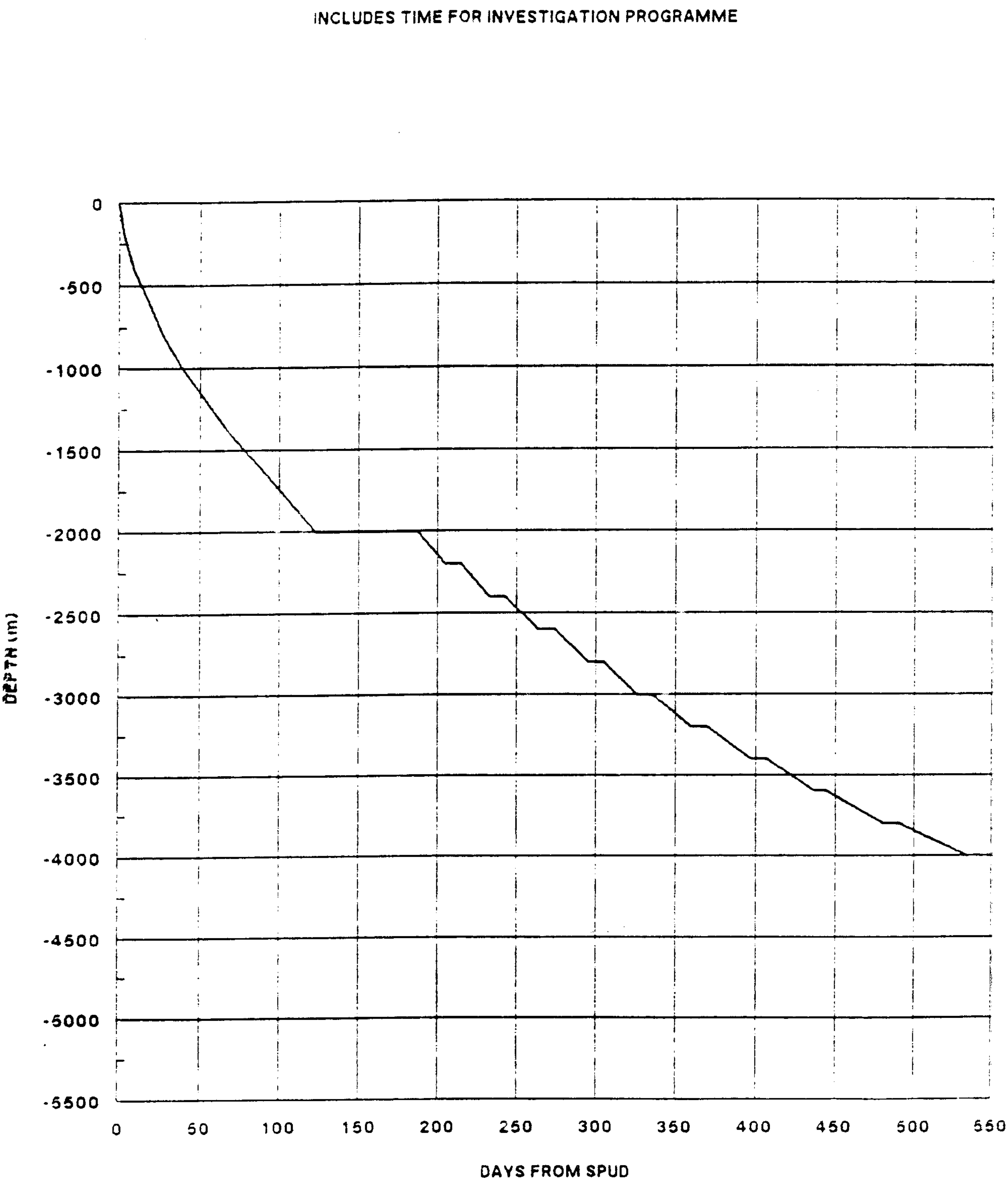


Figure 4.12-2 Time V depth graph for Option A, large diameter borehole

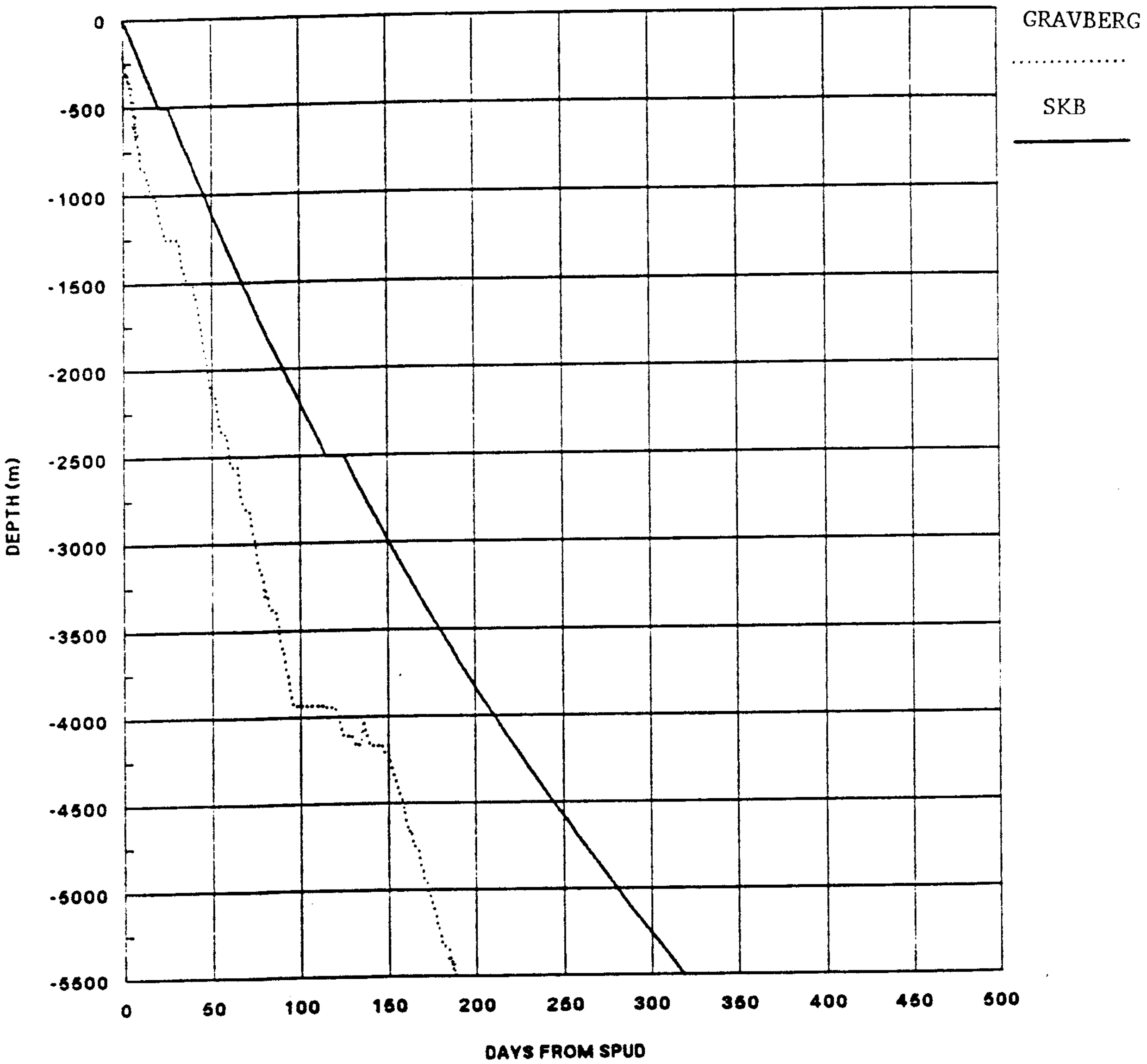


Figure 4.12-3 Time V depth for Option B, small diameter borehole

Option C: Small diameter to 4000 m

Drilling	200
Investigation	100
Deployment	365
Total	665 days

These time estimates are tentative, but for a long programme with repeat operations, the learning curve may significantly reduce the time required. Conversely, if problems arise, they may take time to resolve and so the above guide times are probably a good starting point for this debate.

4.12.2 Cost estimate for large diameter hole

A tentative cost estimate for the large diameter option is set out in Appendix 4. This indicates that the cost for each borehole will be about 388 m SEK (excluding preliminary study). Guide unit costs per unit volume of available disposal volume and unit time are also included in this table.

This estimate indicates that the cost per cubic metre of disposal hole volume based on the internal diameter of the liner will be about 686 000 SEK.

Other than the general uncertainties associated with this scale of operation for a novel application, the principal unknown if the cost of the special copper alloy or titanium high void ratio casing and liners. This could be very high, although the cost will be mitigated by the reduced weight of material in view of the permeable construction

4.12.3 Cost estimate for small diameter holes

A tentative cost estimate for the 5500 m deep small diameter hole is set out in Appendix 4. This indicates that the cost per hole will be about 210 m SEK. Guide unit costs are also included in the table. Cost for the shallower version of the small diameter hole have not been calculated, but the saving would be about 15-20 m SEK.

The estimate indicates that the cost per cubic metre of disposal hole volume based on the internal diameter of the liner will be about 1 712 250 SEK, i.e. about 2.4 times that for the large diameter hole concept.

4.13 Risk analysis

This brief review of the risks associated with the proposals relate principally to the drilling and associated in hole activities. The comments address the status of the main elements of the proposal in terms of previous experience and industry standards or achievements. A more rigorous analysis will be required after the design of the facility is better defined.

4.13.1 Risks associated with deep drilling

There are many inherent risks with deep drilling which must be accepted. However, these can be minimized by good engineering and supervision. In the petroleum industry, wells are drilled on a fairly routine basis to depths in excess of 5000 m, although these are considered unusual and are generally in deep sedimentary basins such as the Anardarko Basin in Oklahoma. The deepest exploration borehole in the western world is the Bertha Rodgers-1 well in Oklahoma drilled in 1973 to a depth of 9583 m. Similar very deep wells have been successfully drilled in Austria to depths up to 8553 m in complex alpine geology. In the USSR, super-deep scientific drilling in relatively slim holes (8-1/2 in diameter) have reached 12 000 m (Kola).

There is very little experience of deep drilling in granites in the western world below 5000 m. Hard rock drilling is commonplace in the Rocky Mountains area of North America. In the geothermal research programmes, purposely deviated wells have been drilled to 4660 m in granites. Risks obviously increase with depth and as a general rule, drilling below about 5000 m becomes increasing difficult and costly. Gravberg-1 is an example.

Rock stress induced instability or breakout is the most troublesome problem in deep drilling in granite given that the rock is otherwise competent. The depth of onset of breakout will depend on the particular geological and rock stress environment. This phenomenon is not restricted to hard brittle rocks. There is now a wider acceptance in the petroleum industry of the effect of rock stress in shales for example and this has resulted in a wider application of rock mechanics principles in designing trajectory directions and mud programmes.

Large diameter drilling to depth is less common. Much of the pioneering work was done in the 1960's as part of the US Atomic Energy Commission programme in the Amchitka Islands in Alaska. This programme included a 2.286 m diameter shaft in crystalline rock to 1875 m at the 'C' site using Parco Rig 3. This borehole holds the record for the heaviest casing string. A 1820 ton 54 in (1372 mm) was run in 1969 which had a wall thickness of 2-1/2 in (64 mm).

The deepest 3.048 m (120 in) diameter hole is the UC-4 hole in Hot Creek Valley, central Nevada using Shaft Drillers Rig 1 in 1968. This shaft was 1676 m deep. A 4.267 m (14 ft) diameter shaft has also been drilled to 632 m in Australia in 1982.

The heavy casing strings run in shafts of this scale are generally run using a hydraulic jack system. In the Amchitka Islands, a jack system with a capacity of 8 000 000 lb was designed.

The proposal to drill 4000 m deep large diameter holes with a final diameter of 800 mm (32 in) is unusual, but it is considered that this is within the envelope of experience and the risks are acceptable at this stage, especially as the design provides a high level of assurance once the casing and liners are in place. The principal risk is hole instability and given that the host rock is expected to be relatively strong, the effect of rock stress spallation or breakout and the degree to which this can be mitigated remains the unknown.

4.13.2 Extraordinary drilling risks

In this category is the prospect of penetrating unforseen geological conditions which prevent the planned approach to the construction of the borehole. This may be a fault structure or significant change in lithology. Much of this risk can be reduced by the preliminary geological studies and exploration drilling and geophysics. Remedial treatment of the borehole to stabilize local zones of instability can be considered using standard cementing methods or special deployment of high strength grout, possibly containing reinforcing fibres, by a bottom dump vessel similar to that envisaged for the deployment mud placement.

Another special risk is the possibility of losing a drilling assembly due to drill pipe failure or hole instability such that remains of the assembly and drill string cannot be fished from the borehole. This is one of the attendant risks of any deep drilling and can happen at any depth. The inclusion of an intermediate casing or support string at 2000 m in the proposals is to limit the exposure to this risk as much as anything else. Only experience in the host rock selected will permit a more informed estimate of these risks which cannot be quantified at this stage. A rigorous programme of preventive inspection of all drilling components will minimize the drilling equipment failure risk. During the actual drilling process, indications of hole problems will become evident and these warnings can be acted upon to reduce the possibility of a more serious situation arising. Again this is a construction risk which is only a problem while the borehole is unlined and not during deployment of waste.

4.13.3 Casing installation risks

This category includes both the primary casing and liner through the deployment zone. The necessity for the casing string or liner to be as large a diameter as possible to restrict the annular space presents special risks. This is necessary to optimize the proposed sealing system. In competent rock with stable walls, this should not be a problem, but there must be recognition of the risk of debris being dislodged from the hole during the installation of the special copper alloy or titanium construction proposed. It is probable that the installation will be by a special hydraulic jacking system with sufficient capacity to lift the casing or liner. Any minor problems can probably be overcome by working the liner slowly up and down to dislodge any protruding rock fragment into the hole. The bentonite based mud system will offer some lubrication. The use of casing jacks will permit the rig to be used to run drillpipe for local flushing or jetting to assist if any hang ups occur.

On the downside, serious hole instability would create a major problem. However, the plan must be that the hole cleanliness and stability must be relatively assured before the casing or liner installation process commences. The leading length of casing or liner could be strengthened and include a bullnose type shoe to help guide the string. The casing or liner strings should be smooth on the outside except for vertical rubbing strips, rubber or polyurethane standoffs and flexible centralisers.

The novel approach to the lining with the high void ratio copper alloy or other suitable material construction in the sizes and depths envisaged must be accepted as unusual at this stage. There is no previous history of such construction.

4.13.4 Risks during canister deployment

Once the borehole has been lined with the proposed liner through the deployment zone and the mud changed for clean bentonite, the repeated re-entry into the borehole for canister deployment should be assured. The problems that can arise will be principally related to premature unlatching of the canister during deployment or conversely problems of unlatching the canister at the desired depth.

The first problem requires some study to assess the velocity a free canister will sink in the mud system envisaged and the braking effect of the hydrodynamics through the completion mud and the thick deployment mud. Premature release may not be a serious problem and the situation may easily be rectified by nudging or vibrating the canister into position later.

The possibility of not being able to release the canister needs some thought on methods of detection that the canister has not released. If this does occur, there is always the standard oil industry practice of severing the string just above the canister with an explosive charge. The effect of this needs study to check the level at which no damage will occur to the canister. This should not be a special problem as it is used with fairly sophisticated tools in deep hole drilling without major damage to adjacent components.

There is always the possibility that the structure movements on the ground or extreme rock pressure at depth may distort the casing. It is recommended that routine calliper runs are made to monitor any changes in hole shape throughout the deployment process to minimize this risk.

4.13.5 Risks associated with stress relaxation

The manifestation of the anisotropic stress regime and the relationship to rock strength is a spallation of the borehole wall initiating as two 'ears' on either side of the wall in the direction of the minimum principal stress. As this breakout process proceeds, the well becomes irregular in profile with the 'ear' like failures combining to give an overall general elliptical shape to the hole. The final shape and orientation is also controlled to some degree by any structural bias resulting from the natural discontinuity system or other fabric features.

In association with this process, there is relaxation around the borehole which may promote the formation of enhanced permeability zones running axially on either side of the borehole. This potential leakage path may or may not result in additional risks of radionuclide migration to the biosphere. The ideal scenario is where the salinity profile and convection system prevent such movement notwithstanding the increase in permeability due to relaxation as the borehole is constructed.

The use of bentonite based fluid for drilling, completion and the deployment mud will be advantageous on promoting infiltration of sealing material into this zones, but the aperture widths may be too small to allow effective penetration. Nevertheless, the driving pressure at depth from the high density mud column together with the increase in pressure as the precompacted bentonite swells, should help to develop an effective seal to this zone.

The effect of relaxation will reduce towards the surface and it is considered that the sealing proposals have the potential to minimize any risk that may remain.

4.13.6 Risks associated with seal formation

Assuming a placement procedure can be developed, the sealing process is relatively straightforward. The principal risk relates to the relationship between the borehole volume and the volume inside the casing or liner. The proposals assume that a barrier with a low hydraulic conductivity can be formed by the combination of the deployment mud and precompacted bentonite. The degree to which this occurs is dependent on minimizing hole overbreak and being able to run as large a diameter high void ratio liner as possible into the hole.

These aspects are difficult to predict with any degree of confidence until a prototype or experimental borehole of the same scale is drilled. However, the reverse circulation air assist drilling system coupled with the realisation that the drilling fluid must be designed to have low fluid loss properties to minimize stress related breakout, suggests that the plan is a feasible proposition and an adequate seal could be achieved.

5. PLUGGING AND SEALING

5.1 General features of the VDH concept with respect to plugging and sealing

The VDH concept will be discussed here in view of the need of sealing and offering a suitable physical and chemical environment of waste canisters in the borehole. The major features of the concept, which are illustrated in Figure 5.1-1 for the special case of "window" cutting of the primary liner, are as follows.

The upper part (0.5 km) will be sealed with very low-permeable plug material (concrete over asphalt) making the interior of the hole almost tight but offering no effective sealing along the rock/plug interface.

The central part (0.5 - 2 km) will be plugged off very effectively by dense Na bentonite having a sealing effect also on the immediate surroundings, i.e. the disturbed zone with enhanced axial conductivity. This effect is obtained by a central core of highly compacted bentonite which expands and consolidates a surrounding bentonite deployment mud. If cemented liners are required locally for rock stabilization, window milling can be made in order to yield effective interaction between the consolidated deployment mud and the rock.

The lower part (2 km) forms the deployment zone with sets of waste canisters separated by very low-permeable bentonite plug material for sealing off the hole at regular, small intervals.

Various diameters of the respective parts of the hole have been discussed and can still be considered. Thus, at one stage of the evolution of the concept the diameter of the upper sealed zone was suggested to be 76 cm, while those of the plugged zone and the 2 km deployment zone were set at 51 and 37.5 cm, respectively. At that stage the diameter and length of the waste canisters were suggested to be 25 cm and 4.3 m, respectively. It was anticipated that 4-5 of them could be firmly connected to form about 20 m long units contained in a cage together with a 1 m high cylindrical block of highly compacted block of Na bentonite at one end. The idea was to bring these units down into the deployment mud by pushing and vibrating them using heavy "pile-driving" equipment connected to the upper end of each canister set unit.

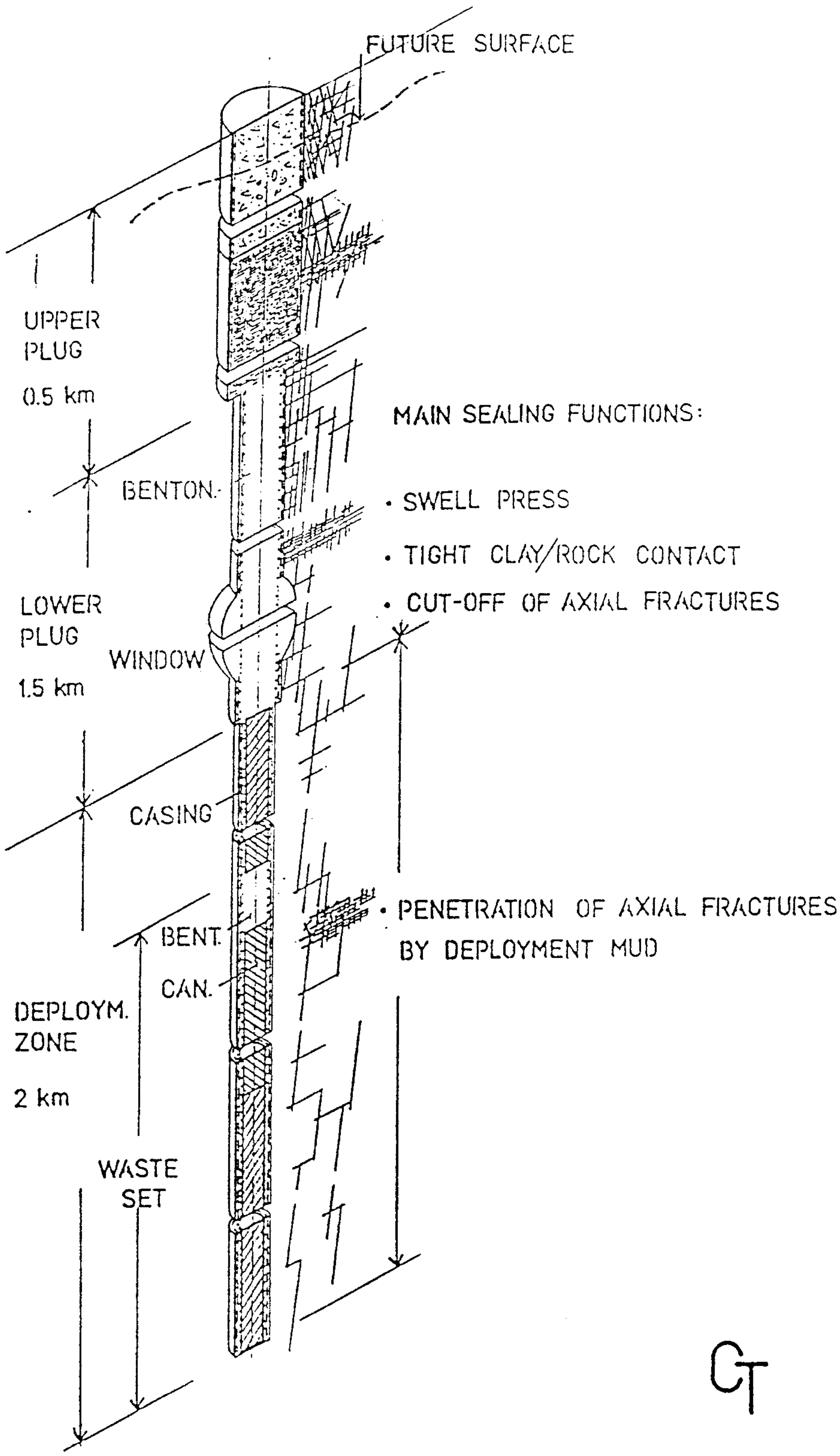


Figure 5.1-1 General view of a VDH concept

5.2 Improved version of the VDH concept with respect to simpler application of canisters and more effective sealing of the deployment zone

5.2.1 General

Although the above-mentioned, intermediate concept Interim report Stage B of the study still appears to be feasible, difficulties are expected in bringing canister sets down through 2 km of deployment mud. Also, the small dimensions of the deployment part of the hole (37.5 cm), allowing for a diameter of only about 25 cm of the bentonite plugs that separate the canister sets, will leave the canisters embedded in a rather pervious deployment mud with negligible sealing ability. These shortcomings suggested that "slim" hole concepts should be abandoned and that larger holes be considered instead. The present concept is in agreement with the large borehole concept put forward in the previous chapter and although further deviation from the proposed borehole diameters can certainly be considered, we will confine ourselves here to outline a version with the following diameters and arrangements:

I. Upper part, 0.5 km

The upper 0.5 km of the plugged zone contains a shallow 30 m mined part with a diameter of several tens of meters, the rest being drilled with a diameter of around 1.4 m. Local rock support in the construction phase in the form of cementation that can be left in the hole or removed by milling in the plugging phase may be required. A "large porosity", cage-type, non-cemented and removable liner made of "navy bronze" or titanium will be used for preventing rock fall in the waste application phase. Plugging will be achieved by asphalt filling and casting of concrete on top of the asphalt.

II. Central part, 0.5 - 2 km

Lower 1.5 km part of the plugged zone will also be drilled with a diameter of 1.3-1.4 m. Temporary stabilization will be made as in the upper part, the porous liner being left in the hole. Plugging is achieved by applying a central core of highly compacted Na bentonite which is embedded in a rather dense bentonitic deployment mud.

III. Deployment part, 2 - 4 km

The deployment zone will be drilled with a diameter of 0.8 m. A "large porosity" cage-type liner of "navy bronze" or titanium with an inner diameter of 0.6 m is run in the hole for preventing rock fall. Sets of 0.5 m diameter canisters separated by blocks of highly compacted Na bentonite are emplaced in a rather dense deployment mud, which is consolidated by the radially expanding blocks.

5.2.2 The deployment zone

5.2.2.1 Mud considerations in the drilling phase

Among the various mud technologies that can be applied for the drilling and deployment activities the use of Na bentonite drilling mud ($p = 1.15$ g/cm^3) and a denser Na bentonite-based deployment mud has been recommended. The rock stability is expected to be satisfactory in the drilling phase, at the end of which clean drilling mud will be left in the hole. Ongoing rock mechanical investigations concerning the effect of stress release and excavation technique on the hydraulic conductivity of the "near-field" rock suggest that steep rock wedges represent critical rock structure components. Thus, one finds, at common principal stress constellations, that natural, steeply oriented fractures that form wedges and are directly or indirectly connected to the hole, become widened by several tens of micrometers by which the pore pressure in them drops instantaneously when the cutter head proceeds downwards. This may cause inflow of drilling mud into the fractures if their apertures are wide enough and if arching does not take place at the fracture openings at the borehole wall. The resulting sealing effect may be considerable if the fractures are kept closed or become closed by a sufficiently high pressure on the borehole wall.

The drilling mud is maintained in the hole above the deployment level during the deployment and plugging operations. It is deplaced in steps by extruding deployment mud from special vessels that are pushed down intermediate to the emplacement of the canister sets.

5.2.2.2 Deployment mud criteria

An ideal deployment mud should be characterized by the following properties:

1. Simple applicability.
2. Low hydraulic conductivity.
3. High shear strength at rest for good lateral support of canister sets and high wall friction for maximum bearing capacity.
4. Low shear strength, i.e. some "fluidity", at its application and at the emplacement of canister sets.
5. Ability to penetrate fractures for self-healing and to resist erosion
6. Very good chemical stability.

Considering first the matter of applicability, an earlier idea of replacing the drilling mud by pumping in deployment mud does not seem to be feasible. Instead it appears to be practical to expell a moderate quantity (5-10 m^3) of thick deployment mud from a vessel to displace drilling mud from below. It is assumed that the expulsion can be made by combining a static pressure and a dynamic impact (through pumping) by which the shear resistance is reduced. This faciliates flow of bentonite-based muds especially if the water content is slightly higher than or at least equal to the liquid limit.

The intended functions can be attained if the deployment mud has thixotropic properties and is reasonably erosion-resistant. Ongoing research indicates that a number of muds with Na smectite clay as one component can be considered, the erosion-resistance, flow properties, and shear strength at rest being improved by addition of quartz filler (powder).

The criterion of low hydraulic conductivity is a major one and it is clear that the increased conductivity that results from adding quartz filler to the clay must be compensated by giving the mud a high bulk density. In conjunction with this one must realize that the salinity may be high at great depths and that this significantly affects the permeability. This is illustrated by the fact that a 30/70 mixture of Na bentonite and quartz filler has a hydraulic conductivity of about 10^{-12} m/s at a bulk density of 1.9 g/cm^3 on percolation with distilled water, while this figure is expected to increase to about 10^{-10} to 10^{-9} m/s when 10% chloride salt solution is used for percolation, the swelling pressure then being in the range of 500 to 700 kPa. The lower conductivity figure is most probable for Na-rich groundwater and the higher when calcium dominates.

Assuming the average conductivity of the near-field rock to be about 10^{-9} to 10^{-8} m/s one finds that the ultimate density of the deployment mud should be at least 1.9 g/cm^3 for this mud composition. The property of the deployment mud to offer good lateral support to the canisters and to yield a high wall friction would also be satisfied by such a composition and density. For comparison, a hydraulic conductivity of 100% Na bentonite is 10^{-9} m/s on saturation and percolation by a 10% $CaCl_2$ solution, when the bulk density is about 1.75 g/cm^3.

As to the "fluidity" of the mud in the application phases, it is clear that thixotropic properties, or a high sensitivity in general soil mechanical sense, are very valuable. A low shear resistance in the application phases calls for a water content of the mud that is somewhat higher than the Atterberg liquid limit, and this inevitably means that the shear strength at rest will be rather low. However, using a relatively low smectite content and adding NaCl to a salt content of 2% of the porewater, it should be possible to confine the water content range for solid/fluid transition to a small interval, i.e. say within 1.05 to 1.1 times the liquid limit. For a 30/70 Na bentonite/quartz-filler mud with the liquid limit 65% this would correspond to a water content of about 70% and a bulk density of about 1.6 g/cm^3 of the deployment mud. It is thus required that such a mud must undergo considerable compression in the holes to get a density of about 1.9 g/cm^3 but we will see later that the "Basic Case" canister/bentonite plug design in the deployment zone actually brings about such compression. For 100% Na bentonite deployment mud acceptable fluid behavior can probably be obtained by adding 10% NaCl solution to a water content of about 100%, which corresponds to an initial density of the mud of around 1.5 g/cm^3.

It should be noticed that while soft clay grouts consisting of mixtures of bentonite and and prepared with a water content that is 1.2 - 1.7 times higher than the liquid limit get their shear strength reduced by about 50 times on mechanical agitation, a water content that is only slightly higher than the liquid limit means that the shear strength is not reduced by more than 10-30%. Still, this would represent such a soft state that the application of mud and canisters would not cause major problems.

The compressibility of bentonite muds is high as illustrated by the behavior of a mud mixture of Na bentonite and quartz filler with an initial bulk density of 1.6 g/cm^3. Thus, pilot oedometer tests using 10% NaCl solution for saturation and percolation indicate that the radial compression of a 15 cm thick annulus of such deployment mud under an effective pressure of around 500 - 800 kPa will be about 5 cm, yielding a net bulk density of 1.9 g/cm^3. Such compression is expected to be caused when contacting the mud annulus with a central core of highly compacted Na bentonite, which expands to an ultimate bulk density of about 1.90 g/cm^3.

Finally, the very important issue of chemical stability needs consideration. The comprehensive study that is being conducted by SKB and many other organizations as to possible changes of bentonite buffer materials in repositories shows that the crystal lattice stability is appreciable at lower temperatures than 120°C, which is the expected maximum temperature at the lower end of 4 km deep VDH's in Swedish crystalline rock. Thus, it can be assumed that at least 50% of the smectite will remain intact after heating to that temperature for a few thousand years and after an additional number of several hundred thousand years at the initial, natural temperature (90-100°C). The influence of metal components (casing, canisters, cage) on the smectite minerals is mainly related to ion exchange processes that can be predicted if these components consist of copper or copper-dominated materials (bronze with less than 10% tin). Dissolution of copper is assumed to be very slow and it is estimated that it will lead both to the formation of solid copper compounds and to release of copper in ionic form. With the expected low concentration of copper ions in the supposedly relatively saline groundwater, Na will probably remain in interlamellar exchange positions while some ion exchange will take place in the electrical double-layers at the outer boundaries of the smectite stacks, possibly yielding some minor particle coagulation associated with a slight increase in hydraulic conductivity.

5.2.2.3 Arrangement of canisters and highly compacted bentonite seals, "The Basic Case"

Although there are several options for the arrangement of canisters and highly compacted Na bentonite seals in the deployment zone, current analyses suggest a concept with the following characteristic features:

- o The copper or titanium canisters have a diameter of 50 cm and are connected to form sets consisting of firmly jointed 4.4 m long canisters separated by 1 m long cylinder of highly compacted Na bentonite with a diameter of 0.50 cm.
- o The sets are contained in well fitting highly porous cages so designed that they can be lifted and pushed down through 4 km of drilling mud and further down through some 10 m of deployment mud.
- o The deployment mud consists of 30/70 bentonite/quartz filler in 10% NaCl solution with a density of 1.6 g/cm^3.
- o The dry density of the highly compacted Na bentonite cylinders is 2.10 g/cm^3.

5.2.2.4 General function

The cylindrical bentonite blocks that separate the canister sets will be surrounded by deployment mud from which the blocks extract water and swell, thereby compressing the mud to an approximately 10 cm thick annulus of 30/70 bentonite/quartz mud with a hydraulic conductivity of 10^{-9} m/s (10% $CaCl_2$ brine groundwater). Equilibrium will be developed when the consolidation (reaction) pressure of the deployment mud equals the swelling pressure of the expanding, highly compacted bentonite cylinders, i.e. a pressure in the range of 500 - 800 kPa. At that stage the diameter of these cylinders will have expanded from originally 50 cm to 60 cm, by which their bulk density will become almost exactly 1.90 g/cm^3, a value that happens to be about the same as that of the compressed mud.

The hydraulic conductivity of the expanded plug is expected to be around 10^{-11} m/s (10% $CaCl_2$ brine groundwater). This calculation of the conditions at equilibrium is based on the assumption that redistribution of water and solids takes place radially only. In practice there will be some, but probably rather insignificant, axial expansion of the blocks and consolidation of the drilling mud in the vertical direction, along the canisters.

A major problem is the mechanical stability of the entire system of canisters, blocks of compacted bentonite, and deployment mud in the course of and after the application of canisters. After complete consolidation of the deployment mud, its shear strength can be estimated at about 200 kPa, applying the figure 15^o for the angle of internal friction, which is quite sufficient for carrying the entire sets of canisters and unconsolidated deployment mud by wall friction.

The development of stable conditions takes time since redistribution of water from the compressed mud to the expanding bentonite blocks is a diffusion process, with an estimated diffusion (consolidation) coefficient of around 10^{-9} m^2/s. Thus, assuming that 80% degree of consolidation is sufficient to make the mud carry the total load by wall friction, about 0.5 year is required to yield stable conditions with no risk of breakage of the cylindrical bentonite blocks or vertical slip along the rock/mud interface. A rough estimate is thus that the operation of applying canister sets should be slow and made in steps such that about 200 meters of canister/block sets are applied per month.

A general picture of the initial arrangement of canister/block sets and of the subsequent mud consolidation process is shown in Figure 5.2-1.

5.2.2.5 Compensation for influence of rock breakouts

The "Basic Case" described in the preceeding text refers to conditions with only insignificant rock breakouts. However, experience shows that breakout does indeed take place from the walls of large-diameter boreholes and the resulting increased width is naturally important in the present context since it reduces the net density and sealing ability of the consolidated parts of the deployment mud. The deep hole drilled at Gravberg (Siljan Deep Gas Project) illustrates the extent to which borehole widening can take place. For this particular granite rock, breakouts were insignificant down to 1500 m depth, while caliper measurements showed an average ratio of maximum/minimum diameters of about 1.3 over about 40% of the borehole length in the interval 1500 - 4000 m. This appears to be much more than in most other rocks and it can be questioned whether such extensive borehole widening will really take place in granite that is suitable for repositories (actually, the average ratio is 1.25 from 1500 to 2000 m, 1.3 from 2200 to 3300 m, 1.4 from 3300 to 3600 m, and 1.3 from 3600 to 4000 m).

In general, the rock structure and the character of the joints are major determinants of potential breakout, the triggering mechanism being stresses induced by the excavation. It seems possible that the mosaic character with a high frequency of hydrothermally healed fractures of the Gravberg rock can explain the extensive borehole widening if one assumes that the healing is not complete and that a number of "Griffith-type" defects remain in the joints. Thus, in contrast to the behavior of the "relaxed" rock matrix of ordinary fracture-rich rock, high stresses can be carried by fracture-filled rock in which comprehensive, brittle failure ("shattering") takes place along largely randomly oriented fracture planes at a critical stress. This can be assumed to yield more isotropic breakout than when the tangential stress exceeds the compressive strength of regularly structured rock.

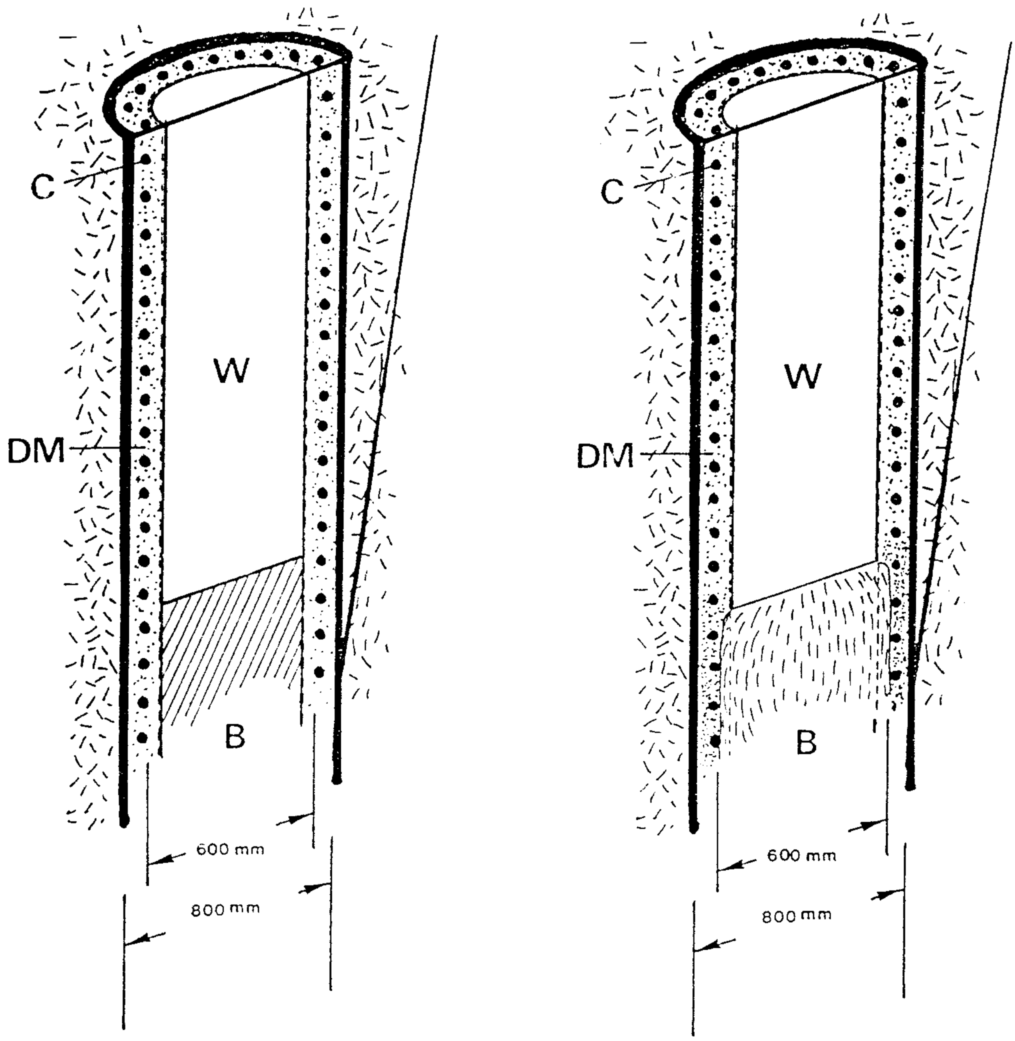


Figure 5.2-1 Arrangement of canister/block set in the deployment zone. Left: Initial state of waste canister (W) and cylinder of highly compacted Na bentonite (B) submerged in deployment mud (DM). Right: State after consolidation of the deployment mud and swelling of the cylindrical bentonite core.

It may be concluded from this that the Gravberg granite represents rather extreme conditions with respect to borehole widening. Still, it is reasonable also in the present context to consider rather frequent breakouts corresponding to a maximum/minimum diameter ratio of as much as 1.5 deeper than 2000 m, and 1.2 closer to the ground surface. While breakouts appear to have occurred over about 40% of the borehole length down to 4000 m depth at Gravberg, it is plausible to assume the fraction to be 15% at maximum in ordinary granite. This suggests that the overall sealing effect will be approximately the same as in the "Basic Case" but it is clear that both the time for consolidation of the deployment mud and its ultimate density and sealing ability depend on how much and how frequently the borehole is widened by breakouts.

Rock mechanics considerations when drilling a deep borehole is discussed in more detail in report Part I, Geological considerations for the Very Deep Borehole Concept.

Taking the maximum/minimum diameter ratio as 1.5 (cf. Fig. 5.2-2), one finds that the net bulk density of 30/70 bentonite/quartz filler mud will be less than 1.7 g/cm^3 at the far ends of the borehole section, while the average density over the entire section will be around 1.8 g/cm^3, provided that the minimum diameter equals the initial 80 cm diameter of the hole. The hydraulic conductivity thereby increases far beyond 10^{-9} m/s for Ca-rich brine groundwater. This can partly be compensated for by increasing the initial density of the deployment mud and the dense bentonite core. Still it appears that the net density of the consolidated mud will be low and that the only possible way of maintaining effective isolation is to increase the bentonite content of the deployment mud. Assuming that it is raised to 100% and that its initial density is 1.5 g/cm^3 in order to obtain a reasonably high fluidity (cf. p 58), the dense bentonite cylinder, which should have an initial dry density of 2.2 g/cm^3, and the deployment mud will combine to form a highly homogeneous clay plug with an ultimate average density of around 1.75 g/cm^3 and a hydraulic conductivity of about 10^{-9} m/s for the worst possible case, i.e. a Ca-rich brine groundwater. The time required for achieving sufficient consolidation and bearing capacity of the deployment mud is longer than in the "Basic Case" and the rate of canister emplacement probably has to be lower. This matter requires laboratory testing and application of suitable rheological models to be settled.

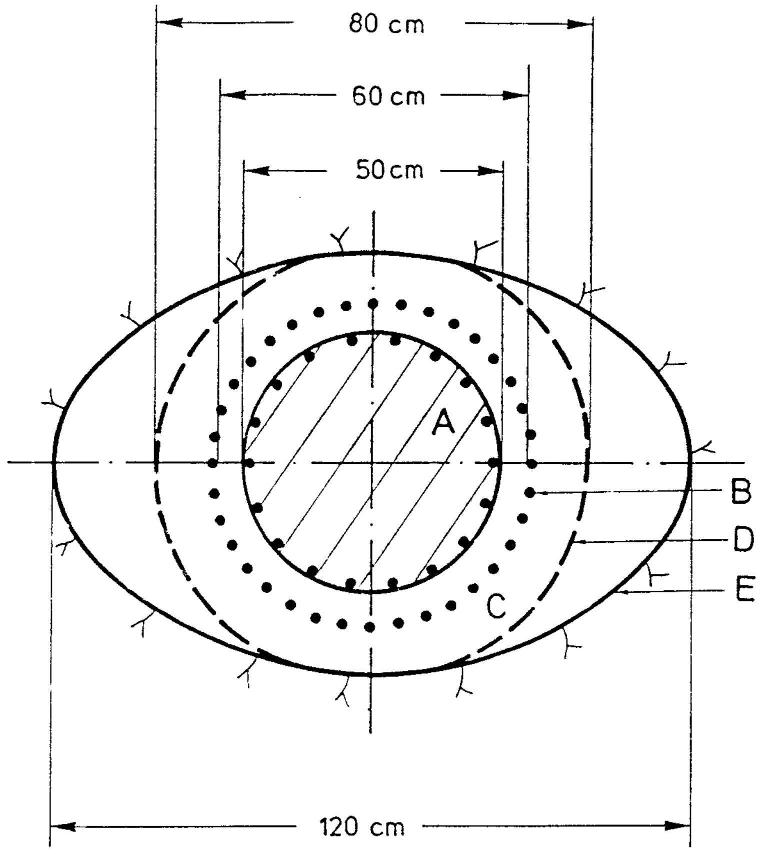


Figure 5.2-2 Assumed shape of the deployment hole at rock breakout. A) Cylinder of highly compacted Na bentonite cylinder. B) Highly porous bronze liner. C) Deployment mud. D) Periphery of hole with no breakout. E) Periphery of hole with breakout.

5.2.2.6 Comments

It is quite obvious that the occurrence of rock breakouts has an impact on the performance of the system of bentonite components. It is clear, however, that there are means of composing the deployment mud so that it can be emplaced from a suitable vessel and so that the net hydraulic conductivity of the system of bentonitic components hardly exceeds the axial (vertical) conductivity of the near-field rock. Very probably, a "standard" deployment mud yielding optimum performance would contain both Na bentonite and quartz filler, presumably with 50/50 composition. The selection of a suitable mixture requires systematic laboratory testing in the first place.

Except where the deployment mud becomes consolidated by the compacted bentonite cylinders it will stay rather soft, contributing very little to the bearing capacity and to the sealing of the system, especially were break-outs take place. However, it can be assumed that the shear strength may be increased by a few hundred percent or possibly more than that due to the microstructural strengthening that follows from the heating. Even the tightness may gain from exposure to heat but the "unconsolidated" mud sections still have a poor sealing power.

Ways of improving the sealing power are offered by increasing the length of the bentonite cylinders from 1 m to 3-5 m, or to increase their number so that each canister becomes located between two bentonite cylinders. Different versions of the concept should be considered in order to find an optimum form once the assumed scenario of consolidation and swelling has been checked experimentally at appropriate temperatures and test conditions. It should be added that the mud density can probably be somewhat higher than assumed here, provided that very effective pumping can be made at the expulsion of deployment mud from the vessels, and that heavy "pile-driving" technique is applied in bringing the canisters down.

It is expected that there will be some vertical movements in the canister/block system due to the strain required to mobilize wall friction, and in conjunction with the radial expansion and slight softening of the compacted bentonite cylinders. Also, there will be a slow settlement of the heavy canisters due to undrained creep and consolidation but it is probably rather unimportant.

5.2.3 The plugged zone

5.2.3.1 Plugging of the lower 1.5 km part of the zone

The sealing of the lower part of the 1.3-1.4 m diameter plug-zone is suitably made in the way suggested in Interim report Stage B of the study, i.e. so that the hole is kept stable by drilling mud of the same composition as in the drilling of the deployment hole, and that highly compacted bentonite blocks are submerged in deployment mud of the same types as in the waste-containing part of the hole. The present concept version implies step-wise replacement of the drilling mud by deployment mud extruded from closed vesels that are pushed down to the level of application. As for the deployment zone a 30/70 mixture of bentonite and quartz filler, stirred up in NaCl solution with a concentration of at least 2% to about 70% water content and a density of 1.6 g/cm^3, is considered as a primary candidate since it yields a fairly high net bulk density and a chance to obtain some self-healing of fractures by an erosion-resistant "grout", and since it is expected to yield a rather high wall friction after consolidation.

Unlike previous versions, the present concept will employ a non-cemented casing for rock support in the plugged zone, with the possible exception of certain local parts where stabilization by use of cement may be required. There will simply be a strong "navy bronze" or titanium, highly porous casing with a required inner diameter of 1.0 m, allowing for almost unrestricted water and mass movement radially as well as vertically. Thus, the cylindrical blocks, which have an initial diameter of 0.9 m and a dry density of 2.1 g/cm^3, will expand radially on the expense of the mud.

The initially 22.5 cm wide annulus of deployment mud will ultimately be consolidated to form a slightly less than 15 cm thick annulus with a bulk density of about 1.9 g/cm^3 surrounding a central core of homogeneous Na bentonite with an ultimate bulk density of about 1.9 g/cm^3, the figures referring to an initial salt content of 2% NaCl. Assuming, in a long-term perspective, the "worst case" of Ca-rich brine groundwater as in the deployment zone, it is expected that the corresponding hydraulic conductivity will be around 10^{-9} m/s of the consolidated 30/70 mud, and 10^{-11} m/s of the bentonite core, the effective pressure at their interface being on the order of 600 - 1000 kPa.

It is obvious from this that the sealing ability of the plug created by the mud and bentonite core is about the same as that of the corresponding clay compound in the deployment zone. However, it is anticipated that the salinity may not exceed that of the oceans in the upper 2 km of granitic bedrock in Sweden, which would reduce the conductivity figures by about one order of magnitude. Although this may be acceptable, it is clear that where rock breakouts take place the tightness may not be sufficient.

The overall mechanical stability of the 1.5 km bentonite block column in the plugged zone is satisfactory even if the application takes place at a much higher rate than in the deployment zone.

5.2.3.2 Compensation for influence of rock breakouts

As in the case of the deployment zone it is expected that rock breakouts will occur although they are assumed to be less frequent and extensive in the plugging zone. It is clear that also minor widening of the borehole will lead to a significant drop in bulk density of the deployment mud and it is concluded that the only practical way of compensating for the associated increase in hydraulic conductivity is to increase the bentonite content of the mud.

Taking the average maximum/minimum diameter ratio as 1.15 (cf. Fig. 5.2-3), and assuming that 100% Na bentonite with a bulk density of 1.5 g/cm^3 is used as deployment mud, one finds, by giving the highly compacted Na bentonite cylinders an initial dry density of 2.2 g/cm^3, that the average ultimate density of the bentonite compound will be around 1.75 g/cm^3. For Ca-rich brine groundwater this will yield a hydraulic conductivity of around 10^{-9} m/s, while for the more probable case of a salinity that corresponds to that of the oceans, the hydraulic conductivity will be in the range of 5×10^{-11} to 5×10^{-10} m/s depending on whether Na or Ca is the dominant cation. In the upper few hundred meters, i.e. just below the concrete/asphalt plug, where fresh- or brackish water conditions usually prevail, the hydraulic conductivity of the matured plug will hardly exceed 10^{-11} m/s.

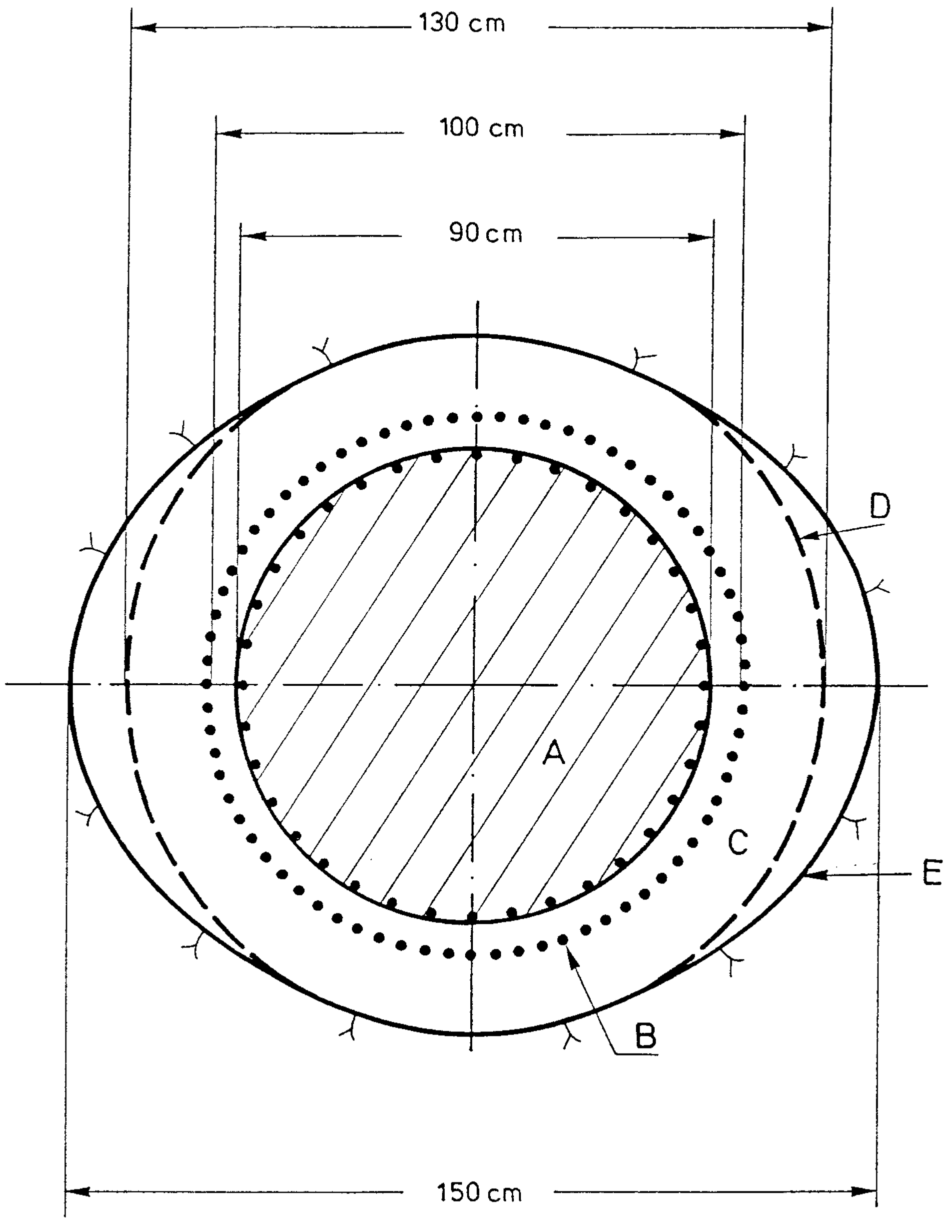


Figure 5.2-3 Assumed shape of the plugging zone at rock breakout. A) Continuous cylinder of highly compacted Na bentonite cylinder. B) Highly porous bronze liner. C) Deployment mud. D) Periphery of hole with no rock breakout. E) Periphery of hole with breakout.

It is clear that rock breakouts strongly affect the "effective" density of the bentonitic compound that plugs off the hole above the deployment zone but that there are means of maintaining a good isolating ability, either by increasing the bentonite content or, possibly, to increase the initial density of the deployment mud. The latter option may be more realistic for the deployment zone but systematic laboratory tests followed by rather large-scale field experiments are called for in order to identify optimum mud compositions and densities.

Like in the deployment zone some vertical movements are expected also in the plugging zone but since the wall friction will be considerable also at an early stage, at least for deployment muds that are rich in quartz filler, and since there are no heavy objects like canisters, such movements are expected to be small. However, this prediction also requires validation by conducting relevant laboratory tests and using applicable rheological models.

5.2.3.3 Sealing of the uppermost part of the plugging zone

The uppermost part of the hole should be plugged with materials that offer effective protection against erosion and abrasion **while maintaining a reasonably tight contact with the rock.** Thus, if a cemented casing has to be used over more than 50% of the upper 500 m length of the plugged part, direct contact between the rock and the plug materials would be required at regular small intervals and this implies that the casing is removed either by milling or by simpler cutting methods.

It is suggested that the lower half of the top zone, i.e. 250 m, is filled with asphalt which has the advantage of staying viscous for very long periods of time, thereby maintaining a close but not watertight contact with the rock and exerting a hydrostatic pressure on the rock of up to 3 MPa at the lower end of the upper zone. The hydraulic conductivity is practically none of this hydrophobic substance and it is expected to perform over millions of years, provided that the composition (malthenes, asphaltenes and resins) can be selected so that it resembles that of natural asphalt contained in crystalline rock fractures.

The top part of the upper zone should consist of concrete for offering protection against erosion and abrasion. While the underlying asphalt is virtually impermeable, the hydraulic conductivity is approximately 10^{-10} to 10^{-11} m/s of somewhat aged concrete but this figure can probably be reduced somewhat if superplasticisers are used, since the water/cement ratio can be kept so low that a certain reserve hydration potential is maintained and thereby some self-sealing and expansion potential. This is expected to retard the degradation rate, which can be further minimized by adding silica fume to the cement and using first-class crystalline ballast material. Still, it must be assumed that the contact between the concrete and the rock is not tight and that water flow along the rock/plug interface can take place easily.

6. STRATEGY FOR SITE SELECTION

6.1 Site Requirements and Selection

Site requirements and handling of the waste on the surface have already been discussed in Stage A. In summary, a fairly flat area is desirable and if all waste is to be deployed at one site the needed area will be in the range of 2 to 10 km^2. This area is comparatively large and even though Sweden is a large country there will obviously be some conflicts about the use of the land.

As a part of this study the number of necessary boreholes for the Swedish highly radioactive spent fuel has been evaluated. Based on canister design, plugging and sealing, and a geological model for Swedish bedrock, approximately 35 holes will be needed if the boreholes are 4000 m deep and have a diameter of 0.8 m in the deployment zone.

Geological prerequisites for a deep borehole repository will not differ that much from what is presented in the KBS-3 study. New techniques will of course be needed for investigations of the deeper parts of the bedrock. Deep geological investigation methods are discussed in Part I of this report.

The above described concept is based on an on-shore location. A deep borehole concept, however, can also apply to other areas for consideration such as an off-shore location in the Baltic. Deep drilling off-shore is a well known and proven technology and the normally shallow water depth of the Baltic will be easy to handle. Some advantages with an off-shore location are listed below:

- Little or no conflicts about the use of the bottom of the sea.
- An off-shore drilling rig is easy to move and a repository could therefore cover a large area.
- Any radionuclides reaching the sea bottom and the biosphere will then be heavily diluted.
- Advantageous hydrogeological preconditions will probably exist with small groundwater movements. This is supported by the analysis of water from the SFR storage where higher salinity water was encountered in the bedrock compared to the sea.

Similar geological conditions could also exist in locations on-shore. For example, boreholes on the island of Gotland show a salinity content between 4 to 8% which is much higher than compared to the seawater today. Available information also indicates the existence of a tight clay horizon within the sedimentary rock column. This zone will help in preventing any radionuclides from reaching the surface.

Of course, negative aspects of an off-shore location should not be neglected. For example, geological investigations will be much more difficult. If it is required to re-enter the repository in the future then an off-shore location is less desirable.

6.1.1 Geological Considerations

There probably exist numerous sites in Sweden which are suitable for a very deep hole nuclear waste repository. One of the advantages of the concept is that it is not critically dependent upon near-surface geological conditions for a site to be suitable. This will allow greater flexibility in choosing the site than what the KBS-3 concept allows. Even though great flexibility exists, there are a number of geological factors which deserve special consideration when choosing the site. These are:

1. Extensive vertical fracture zones.
2. Homogeneous rock.
3. Presence of highly saline water.
4. Degree of anisotropy in the horizontal stresses.
5. Long-term stability of the site.

A borehole should not be drilled in or in close connection to a vertical fracture zone since such a zone would provide a direct high conductivity path to the surface. The entire site should be located well away from any extensive vertical fracture zone. This point may not be as critical as previously thought since results from the hydrogeological modelling show that the presence of a well conducting vertical fracture zone near a borehole actually reduces the vertical flow of water in the surrounding rock. However, when considering long-term stability of the site it will be wise to avoid placing the site too close to any extensive vertical fracture zones.

The ideal location for the site will be in a large homogeneous block of low permeability rock, however, this is not a necessary requirement. As has been found in all deep boreholes in nontectonically active areas, the permeability of the host rock at depths below 2000 m will be very low and even the fracture zones have relatively low permeability at these depths. An area which is obviously highly fractured should be avoided since this means that a smaller proportion of the deployment zone can be used and will require additional boreholes. At present it appears that granites as a whole are less permeable than gneisses and, thus, is the preferred rock type.

As was shown in Chapter 3 the presence of highly saline water at depth will be extremely advantageous for a very deep hole repository. As the review of deep wells in Part I of this report shows, it is most likely that highly saline water will be encountered at depth in crystalline rock. Depths to this saline water are unknown throughout most of Sweden, however, there are indications that it will be found at shallower levels (1-2 km) close to the Baltic compared to inland. Locating the site where highly saline water is present at 1-2 km depth is considered to be more important than locating it at a site with relatively few fracture zones.

Since breakouts are an obvious problem in securing adequate sealing by bentonite in the deployment zone and in the plug zone, it is important to consider the stress conditions when locating the site. Indications are that the breakout situation in the Gravberg-1 borehole is extreme. Other deep boreholes, KTB and Cornwall, show no such breakouts at 2-3 km depth. The KTB pilot borehole is virtually free from breakout below 2 km as are the boreholes at Cornwall. Special attention was given to the drilling fluid at KTB so that this along with the drilling technique used may have resulted in less breakout than would otherwise have been the case. At Cornwall, essentially the same drilling fluid and drilling techniques were used as at Gravberg indicating that geological factors play a role in the formation of breakouts. An area where it is well known that there is large anisotropy in the horizontal stresses should, therefore, be avoided.

The greatest uncertainty in the geological considerations for choosing a site is the long-term stability of the area. It is impossible to guarantee that a site will be stable over a period of hundreds of thousands of years. However, it is generally accepted that when movement takes place at shallower levels in the Swedish shield it is along pre-existing zones of weakness. By not deploying waste along these zones, which are simple to identify, the likelyhood that future movements would displace a borehole where waste is located is extremely unlikely. Even if movements displace the borehole at a position where waste has been deployed, the great depth at which it is deployed will result in that when radionuclides reach the surface they will probably be sufficiently diluted to be harmless.

6.1.2 Logistical Considerations

Due to the flexibility available in choosing the site, as discussed in the previous section, the selection can be based more on practical aspects than on geological aspects. Logistical factors to consider are:

1. Transportation of cannisters to the site.
2. All waste need not to be deployed at a single site.

The very deep borehole concept allows the choice of site to take into account the transportation aspects to a high degree. For instance, a suitable site may be along the coast where all transportation of cannisters to the site would be by sea. This would eliminate the need for specially built rail transport networks and the costs and safety hazards which accompany transport by rail. A choice along the Baltic coast may also be very advantageous from a geological standpoint if highly saline fluid is present at 1 or 2 km depth as may be the case.

If logistics determine that all waste cannot be stored at one site then the very deep borehole concept allows for waste to be deployed at different sites. This may be desirable if the greatest risks are considered to be in the transportation of the waste to the site or if not enough space is available for permitting at a single site.

6.2 Investigation Programmes

Geological investigation programmes have already been described in Part I of this report, however, they are briefly reviewed in this chapter. The investigation programmes are similar to those for the KBS-3 concept except for that more emphasis is put on deep geophysics and less on surface geology investigations.

6.2.1 Pre-site Selection

As discussed in Section 6.1.2, the site selection will probably be based on both geological considerations and logistical considerations. However, it is highly desirable to have a site which has geological properties as outlined in Section 6.1.1. A great deal of work has already been done on locating major vertical fracture zones and these should be fairly simple to avoid when choosing the site. As the relationship between deep geological structures and surface geophysics becomes more established it should be possible to choose a fairly homogeneous rockmass for the site. This process may require acquisition of surface geophysical data and drilling of boreholes down to 4-5 km depth to confirm surface interpretations. Locating the depth to highly saline fluid in crystalline rock by surface measurements is a relatively unknown technique. However, predictions from surface magnetotelluric measurements at Gravberg of less resistive rock in the 6-10 km depth range and recovery of highly saline fluids from these depths are encouraging. With controlled source magnetotelluric surveys it may be possible to estimate the depth to highly saline fluids much more accurately. This will require considerable development of the technique and drilling of boreholes to 2-3 km depth to confirm results. The search for homogeneous rock and highly saline fluid will indirectly give information about the stress field as boreholes are drilled. This will indicate whether the breakouts experienced at Gravberg are a general phenomenon in Swedish rock or are unique to that site. At present, no real problems are envisioned in the site selection process.

6.2.2 Site Investigations Prior to Drilling Repository Holes

These investigations would include detailed surface geological mapping and surface geophysics as described in Part I of this report. Surface geophysics would focus on the rock at surface down to dephts of 6 to 8 km. Drilling of several boreholes down to about 1 km below the deepest repository hole at the site will be necessary to confirm surface interpretations. The surface geophysical techniques envisioned to be employed are:

1. Gravity
2. Magnetics
3. Magnetotellurics
4. Seismic refraction
5. Seismic reflection
6. Seismic monitoring (i.e., earthquake or rock movement detection)
7. Remote sensing
8. VLF surveys

A borehole investigation programme as outlined in Part I would be done in the investigatory boreholes drilled. The geophysical logs envisioned to be run are:

1. Formation Microelectric Scanner (FMS)
2. Digital Sonic
3. Geochemical Logging Tool (includes natural gamma spectrum log)
4. Density
5. Resistivity
6. Four-arm caliper

It is possible to fully core the investigatory holes. This core would then be carefully core mapped and comparisons made with the geophysical logs. It may be required by the authorities that the holes are fully cored. It should be realized, however, that at times it is not possible to recover 100% core, particularly in fractured intervals. In such zones, geophysical logs will have to be relied upon.

It will be important to carry out tests in the investigatory boreholes which give information about the rock some distance away from the boreholes. These investigations would include:

1. Hydraulic Tests
2. VSP and crosshole seismics
3. Borehole gravity
4. Borehole radar

6.2.3 Investigations in the repository holes

As discussed in Chapter 4, a suitable method for investigating the large diameter boreholes is to drill 200 m sections of fully cored pilot holes in advance of the large diameter hole. This will be necessary if core should be obtained since it will not be possible to core in large diameter holes. Drilling of these pilot holes will also allow standard geophysical logs to be run in them. If the drilling fluid used drilling the pilot holes is water then valuable hydrological information can be obtained over 200 m intervals in the repository holes. In the slim hole concept, pilot holes will not be necessary to drill, however, the drilling fluid will have to be changed to water over those intervals where hydrological information is desired. In general, the investigation programme for the repository holes will be quite similar to that for the investigation boreholes. Casing requirements, however, may limit some of the seismic investigations which can be run in the repository holes.

6.2.4 Monitoring after deployment

The following parameters are considered important to monitor at a very deep borehole storage site:

1. Seismic activity
2. Temperature gradients
3. Hydrology
4. Nuclear activity

With the early installation of a seismic monitoring network on the surface it should be possible to locate rock movements at depth at the site. If the network can be calibrated with sources at known positions in the boreholes before they are used for storage it should be possible to locate these movements to accuracies of a few meters in the horizontal plane. VSP while drilling (Rector and Marion, 1989) could play an important role in the calibration. Movements close to the deployment zone in a borehole should not occur since waste is not deployed adjacent to fracture zones. In the event that such movement should occur then it may be desirable to intensify monitoring in that area.

The temperature gradient should be monitored carefully in the investigatory boreholes which were drilled prior to the repository boreholes. Comparison with predictions based on hydrogeological models of the site will allow anomalous flow patterns to be detected. To monitor the temperature gradient adequately may require the drilling of shallow boreholes at the site down to 500 - 1000 meters.

The area should be monitored hydrogeologically for the same purposes as the temperature gradient is monitored. It may, however, be difficult to determine rates at which fluid is entering the boreholes in the area.

7. PREFERRED BOREHOLE CONCEPT

7.1 The 4.0 km large diameter hole concept

The technical and ecnomic evaluation of different borehole options, together with the sealing process and canister design shows clearly that a large diameter borehole to 4 km depth is prefered. Two different systems will be of interest for further studies:

Option A

Large diameter boreholes, 0.8 m in diameter in the deployment zone between 2-4 km depth. 4 BWR or 1 PWR + 2 BWR (SVEA) fuel bundles will be stored in each canister. The canisters will have an outer and inner diameter of 0.5 and 0.39 m, respectively. The maximum temperature in the canister-bentonite interface will be 120°C. A total of 35 boreholes will be required for the Swedish high level radioactive waste.

Option B

The same boreholes as option A, but the fuel elements will be rebundled and the single rods packed together, rod consolidation, in the same type of canisters as above. If this system should be of interest the maximum allowed temperature in the canister-bentonite interface should be increased to 150°C from 120°C. A total of 19 boreholes will then be required for the Swedish high level radioactive waste.

Option A constitutes the main concept put forward in this report. With a similar canister as in the KBS-3 concept and a 2 km rock overburden above the repository the VDH concept should have a higher safety margin towards migrating radionuclides to surface compared to KBS-3. The cost is, however, higher. Option B constitutes a greater potential in this context. If it could be proven that the bentonite seal can resist a temperature of 150°C and that rod consolidation is a feasible system the cost for option B will be in the same range as the KBS-3 concept.

The result from the hydrogeological modelling presented in Chapter 3 shows that canisters with waste should not be deployed in parts of the borehole that is intersected by major fracture zones or other discontinuities with increased permeability. Results from the Gravberg-1 borehole shows that at least 80% of the boreholes could be used for deployment. This figure may be conservative but is nonetheless used in this report for cost estimates. Fractured parts of the borehole will be sealed off with bentonite as presented in Chapter 5. Unstable parts of the borehole may also be stabilized by concrete plugs.

7.2 Canister considerations

7.2.1 Prerequisities

The spent fuel which is to be disposed of will be temporarily stored for about 40 years in CLAB. The total amount of spent fuel is 6000 metric tons uranium from BWR and 1800 metric tons uranium from PWR. The remaining effect is 700 W per metric ton uranium from BWR and 800 W per metric ton uranium from PWR.

The total amount of uranium that should be deployed has been reduced to 7656 metric ton in the report Plan 89. The difference is very small and the old figure is therefore still used in this report.

Larger waste products such as different metal parts are not considered in this report. They can either be molded, canistered and deployed in holes similar to this proposed VDH concept or deployed in special shallower and wider holes with more engineered barriers or in a mined repository.

7.2.2 Fuel assemblies

BWR

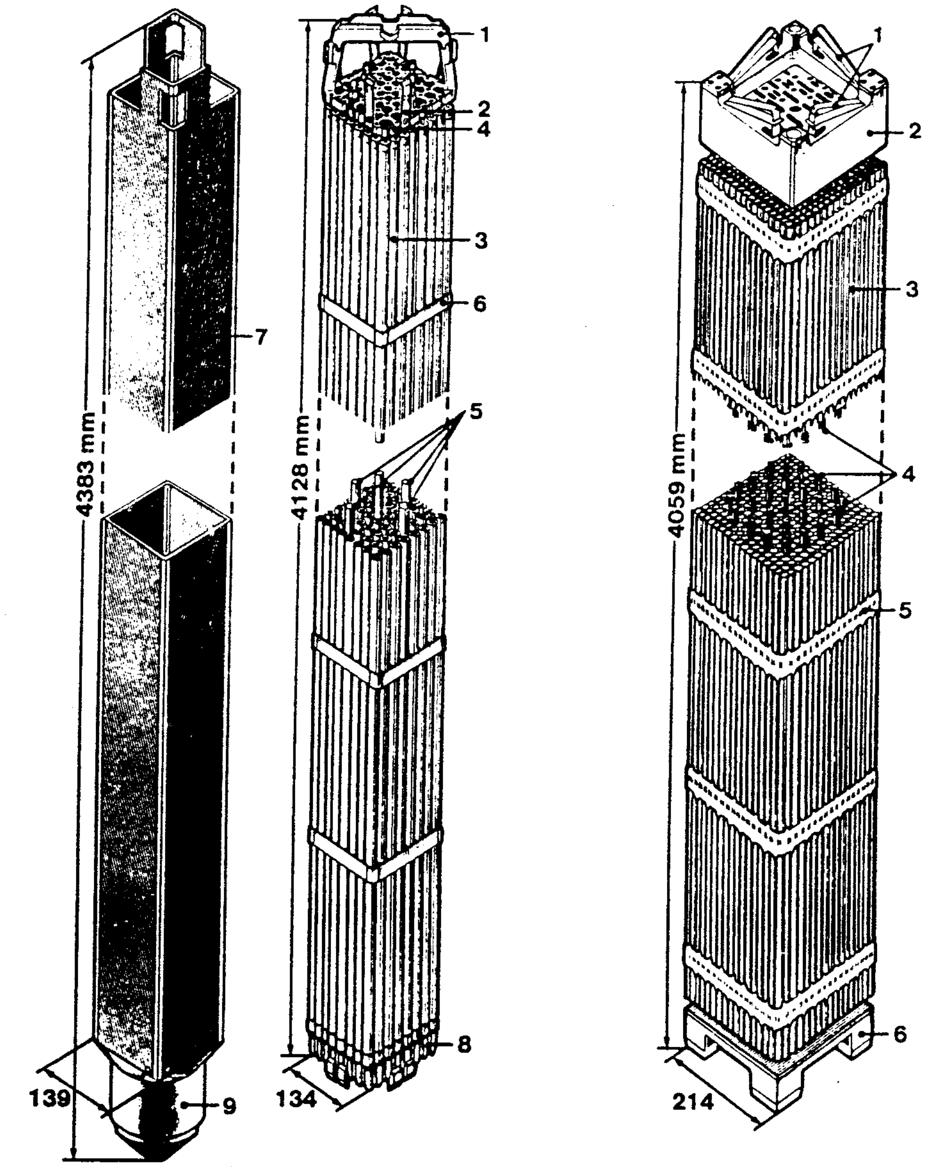
The configuration of a BWR fuel assembly is shown in Figure 7.2-1. Out of the total weight of 297 kg, 178 kg is uranium. Each fuel assembly consists of 63 fuel rods, diameter 12.25 or 11.75 mm and 3998 mm long. Total length of the fuel assembly is 4383 mm and the maximum width is 134 mm.

One variant of the BWR fuel element that is of interest for the VDH concept is the so-called SVEA fuel, see Figure 7.2-2. The SVEA fuel differs from the standard fuel mainly in that the fuel bundle has been divided in four subassemblies. Each subassembly has a width of 67 mm and can be handled separately and placed together with the BWR fuel optimizing available space in each canister.

PWR

The PWR fuel assemblies consists of more but thinner fuel rods than the BWR. As seen in Figure 7.2-1, the PWR fuel assemblies do not have any fuel box as the BWR assemblies have.

Total weight of the PWR fuel assembly is 666 kg of which 464 kg is uranium. Each fuel assembly consists of 264 fuel rods, diameter 9.5 mm and 3852 mm long. Total length of the fuel assembly is 4059 mm and the maximum width is 214 mm.



1 HANDLE
2 SPRINGS
3 FUEL ROD BUNDLE
4 TOP TIE PLATE
5 TIE ROD
6 SPACER
7 FUEL CHANNEL
8 BOTTOM TIE PLATE
9 TRANSITION PIECE

a.

1 SPRINGS
2 TOP NOZZLE
3 FUEL ROD
4 CONTROLROD GUIDE TUBE
5 SPACER
6 BOTTOM NOZZLE

b.

Figure 7.2-1 a. BWR fuel assembly
b. PWR fuel assembly

Mechanical design

The non-boiling water inside the SVEA fuel assembly is introduced through an internal cruciform structure attached to the fuel channel walls. The internal structure and the outer walls are an integral mechanical component.

The integral channel design also ensures a considerable reduction in channel wall creep deformation

The much improved creep deformation properties are utilized to minimize the amount of Zircaloy in the channel walls and also allow for narrowing the outer water gaps, without reducing the control rod blade clearance at the end of channel life

The transition piece to which the SVEA channel is attached can be designed to suit all operating BWRs and full mechanical compatibility is ensured between the SVEA fuel and fuel of earlier design.

The SVEA fuel bundle consists of four sub-bundles separated by the internal structure of the channel.

The design principle of the sub-bundles is the same as before, with separate top and bottom tie plates and spacers of ASEA-ATOM low pressure drop design. The top tie plates of the four sub-bundles are attached to a common handling piece. Although the total SVEA assembly, with sub-bundles and channel, is handled as one unit as in the past, the common handling piece can be removed for individual handling of the sub-assemblies

The fuel rod is of the standard ASEA-ATOM design which has an outstanding performance record.

All materials used in the SVEA assembly are identical to those in the earlier design.

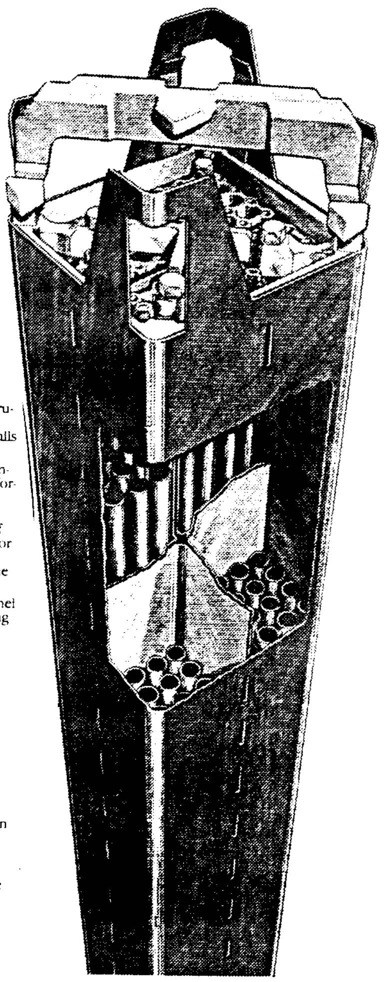


Figure 7.2-2 SVEA fuel assembly (a variant of BWR fuel)

7.2.3 Type of canisters

Two, in principal, different methods for deployment of fuel rods will be discussed in this report. The basic option is to deploy the fuel element as they are in a suitable canister similar to the system presented in the KBS-3 study. This method will not optimize the free space inside the canister, but separate treatment of the fuel rods is not needed. One option also discussed is rod consolidation which implies that the fuel assemblies are dismantled and the fuel rods rebundled. The advantage with this system is that more waste could be deployed in each canister and thus reduce the number of boreholes. One possibility is also that smaller canisters can be used and thus slimmer boreholes.

The size of the canisters will be determined by the following factors:

- Maximum outside diameter should be 500 mm due to available borehole diameter.

- The maximum allowed temperature in the bentonite canister interface should be 120°C.

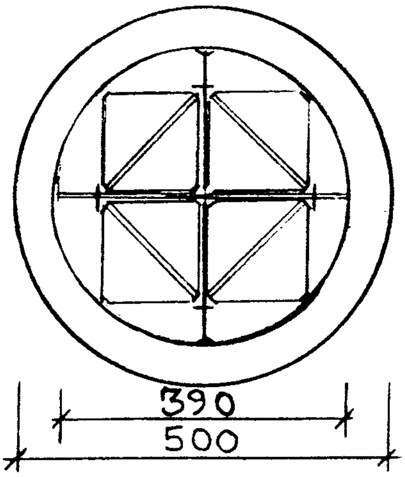
This implies that the maximum temperature increase from the waste can be 50°C considering a temperature gradient of 16.1°C per km and a 4 km deep borehole.

Two types of canisters will be considered. A self-supporting titanium canister with a void filler or a solid copper canister. A canister consisting of an inner part of steel and an outer part of copper has been suggested. The available size is, however, too small and the stresses in the steel membrane will be too high if a self-supporting canister is requested.

Presented canister design is based on the large diameter borehole concept. For slimmer boreholes the same principal design of the canisters will be used.

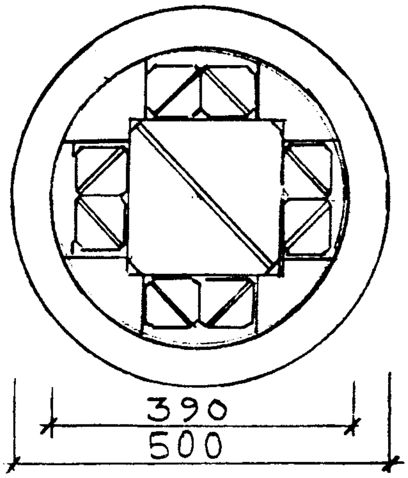
Canister size

Type A, 4 BWR element (maximum width of each element is 134 mm)



Inside diameter	390 mm
Outside diameter	500 mm
Weight uranium	712 kg
Total effect	498 W
Total number of canisters	6488

Type B, 1 PWR element (maximum width 214 mm) and 2 BWR element divided in 8 SVEA subassemblies (maximum width each subassembly 67 mm)



Inside diameter	390 mm
Outside diameter	500 mm
Weight uranium	820 kg
Total effect	620 W
Total number of canisters	3879

In order to limit the temperature in the bentonite seal the type B canister, which is hotter, will be placed in the upper part of the deployment zone.

The total number of the SVEA elements is not known at present, but the final available number will exceed the needed 7758 which is twice the number of PWR elements.

7.2.4 Canister design

The canister will consist of either titanium or copper in order to minimize corrosion and thus hydrogen gas formation. Due to the need of a very strong material and the above prerequisities titanium is chosen for the self-supporting canister.

A metallic material changes its properties with increasing temperature. At 120°C the limit of proportionality of titanium, the canister material chosen here, is reduced to about 2/3 of its value at room temperature. Young's modulus is also reduced some 5-10%, but has only a minor influence on the crippling.

When a metallic material is subject to stresses at temperatures above about half the melting temperature in °K there will be a viscous deformation, so called creep, apart from time-independent deformations. The creep is not judged to be critical at the low temperatures considered here, about 120°C, and the short period of time, a few years, during which the canister has to be intact.

The calculations presented in Appendix 5 should be seen as estimates. If more exact calculations are to be obtained, FEM-model calculations have to be made.

The void in the canister should be filled with some supporting material, thus supporting the canister wall. Cooper & Tough (1984) suggest glass beads as void filling material, while others have suggested lead beads for example. Other possibilities are to cast the void with lead, but the volume of lead is reduced some 2% when it solidifies, thus producing new voids. A cementitious environment is advantageous from a radioactive distribution point of view. A dry cement powder is also advantageous from a gas generation point of view.

Design load at 4 km depth:

- The maximum load during deployment in a mud with a density of 1.15 g/cc will be 46 MPa
- The maximum load during storage will be 45 MPa (the hydrostatic pressure plus the swelling pressure from the bentonite seal).

Self-supporting titanium canister

The result from the estimates presented in Appendix 5 show that it is possible to construct a suitable canister. The stresses in the shell of a canister with an outside diameter of 500 m and a wall thickness of 55 mm will be 209 MPa which is less than the design strength of 240 MPa at 120°C.

The estimates also show that it is impractical to design a self-supporting end-lid. In this report it is assumed that the needed metal spacers between the element will support the lids and thus reduce the needed thickness to 100 mm.

If the canisters are filled with a void filler, the walls can be made thinner or a higher safety margin towards colaps is encountered. A thin wall can, without harm, be deformed more than a thick wall.

The estimates have been made under the assumption that the mud in the borehole during deployment has a density of 1.15 g/cm^3 and that the swelling pressure of the bentonite in the hole is 5 MPa. Any change in mud density will influence the external pressure and thus the stresses in the canister wall. There is a linear proportionality between change in mud density and canister wall thickness.

The only material parameter that has a major influence on the required wall thickness is the design strength. The other parameters, like Young's modulus and Poison's ratio have some minor influence on crippling resistance.

As stated above, the creep of the material has no influence on the canister design for the short period of time that the canisters have to stay intact and any void filler will also reduce the creep.

Pressed copper canister

The fuel element could also be stored in a pressed copper canister with the same size as for the self-supporting titanium canister. This method, hot isostatic pressing of waste, HIPOW, is described in detail in the report Encapsulation and handling of spent nuclear fuel for final disposal, SKBF/KBS Teknisk rapport 83-20.

The result of the HIPOW technique is a solid canister with the fuel bundles embedded in copper, see Figure 7.2-3 below. This solid canister will with no problem stand the high pressure during deployment and the long-time storage. From a safety assessment point of view the HIPOW canisters is preferable compared to the self-supporting titanium canisters with a void filling material.

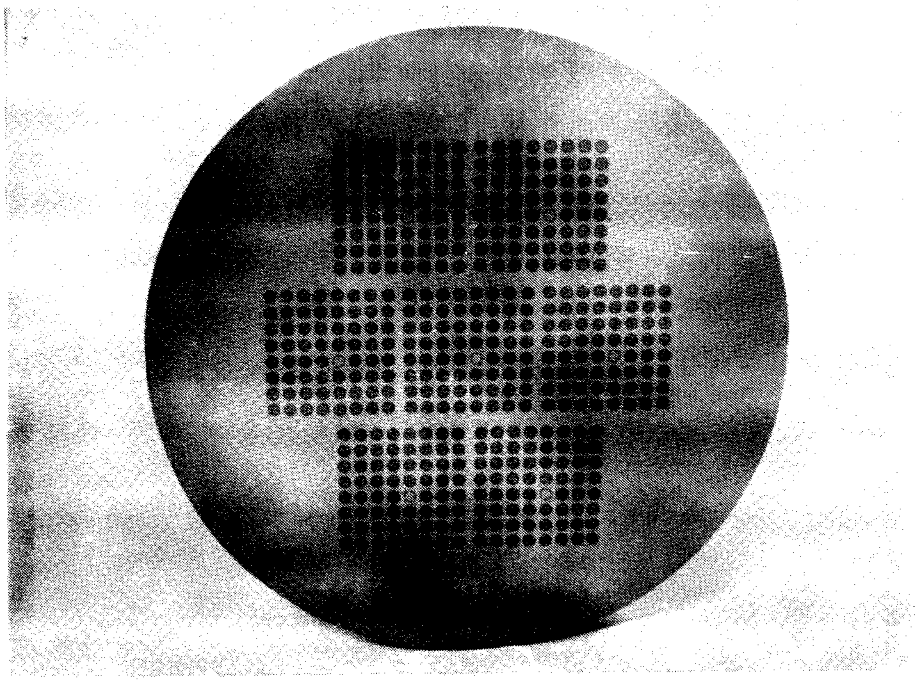


Figure 7.2-3 Diametrical cut of a canister with fuel bundles embedded in copper

7.2.5 Required number of boreholes for option A

The required number of boreholes together with a summary of waste data is presented in the Table 7.2-1 below. Note that it is assumed that waste will be deployed in only 80% of the deployment zone. This figure may be too low and can probably be increased. The 80% figure was chosen based on a conservative evaluation of the Gravberg-1 borehole.

Table 7.2-1 Required number of boreholes and waste data

Weight uranium:

- 6000 metric tons BWR
- 1800 metric tons PWR

Canister Type A 4 x BWR

- Number of canisters	6488
- Weight uranium	712 kg
- Total effect/canister	498 W
- Temperature increase max	39.6°C
- Temperature increase average	29.3°C
- Temperature max bentonite, 4 km	110°C
- Temperature max bedrock, 4 km	100°C

Canister Type B 1 x PWR, 2 x BWR (Svea)

- Number of canisters	3879
- Weight uranium	820 kg
- Total effect/canister	620 W
- Temperature increase max	49.3°C
- Temperature increase average	36.5°C
- Temperature max bentonite, 4 km	119°C
- Temperature max bedrock, 4 km	107°C

Available length for deposition per hole is 2000 m.
80% can be used for waste disposal

- Effective length for deposition		1600 m
- Length of canister including bentonite (4.4 m + 1.0 m)		5.4 m
- Number of canisters/hole		296
- Required number of holes	$\frac{6488 + 3879}{296} =$	35

7.2.6 Rod consolidation

As shown in Table 7.2-1 the maximum temperature in the surrounding bentonite is 110°C and thus close to the maximum recommended temperature of 120°C (canisters with both BWR and PWR element will be placed in the upper part of the borehole). With rod consolidation it will therefore not be possible to deploy that very much more uranium per canister if the temperature criteria, max 120°C in the bentonite, should not be exceeded. It may also be possible to deploy more uranium in the upper part of the borehole but this option is likely to be impractical.

Rod consolidation will reduce the required size of the canisters. The possibility for slimmer boreholes may reduce the total cost but the potential for cost savings must be judged as limited due to the cost for rod consolidation.

If rod consolidation should be a concept of interest the maximum accepted temperature should be aloud to increase to 150°C. With a bottom hole temperature of 150°C at 4 km depth the temperature at 2 km depth will be in the range of 120°C. The temperature limit of 120°C in the bentonite should not be seen as definite. Further research work may prove that a good sealing effect could be obtained even with higher temperatures in the bentonite.

Required number of boreholes, together with a summary of waste data for the rod consolidation concept is presented below in Table 7.2-2. The estimate is based on a maximum temperature in the bentonite of 150°C.

Table 7.2-2 Required number of boreholes and waste data

	BWR	PWR
Fuel rod diameter (mm)	12.25	9.50
Weight uranium each rod, kg	2.845	1.762
Fuel rod length	3.998	3.852
Total weight of uranium, kg, in each canister in order to meet the temperature criterion 150°C	1438.0	1338.7
Number of rods each canister	506	760
Needed canister size, inside diameter mm. The estimate is based on 25% voids	319	304
Outside canister diameter	400	400
Number of canisters	4173	1345
Available length for deposit per hole is 2000 m. 80% can be used for waste disposal		
- Effective length for disposal	1600	1600
- Length of canister including bentonite (4.4 x 1.0)	5.4	5.4
- Number of canisters/hole	296	296
- Required number of holes	$\frac{4173 + 1345}{296}$	= 19

Summary

A suitable canister for the alternative with rodconsolidation will have an outside diameter of 400 mm with an inside diameter of 320 mm.

Minimum required borehole diameter for this alternative will be 700 mm:

Canister	400
Free space between canister and casing 2 x 50	100
Casing and annulus 2 x 100	200
Minimum borehole diameter	700 mm

In the cost estimate presented in Chapter 8 it is assumed that the standard canister with an outside diameter of 500 mm will be used even for the rod consolidation alternative and thus a 800 mm borehole in the deployment zone.

7.3 Advantages with the VDH concept

In a comparison with the KBS-3 concept for storage of nuclear waste there are several advantages with the VDH concept. Some of the advantages which will be briefly discussed in this section are:

1. Geological aspects.
2. Multiple deployment sites.
3. Adaptability to technological innovations.
4. Possible economic advantages.
5. Retrieveability.

Geological aspects

There are several geological advantages with the VDH concept. Among these are the lower permeabilities in the rock at the depths which the waste is to be deployed, a greater rock column serving as a natural barrier (2 km vs. 500 m) and the probable presence of saline water at the depths being considered. From a geological standpoint it is much better to deploy waste at 2 km than at 500 m. Much less movement of water in the rock itself is expected at the greater depth due to lower natural permeabilities and lower fracture intensities. As discussed in Part I, there appears to be an exogenic zone in most areas where deep wells have been drilled where surface waters predominately circulate. At Kola this zone was found to extend down to 800 m, at the KTB site down to 500 m and at Gravberg it appears to extend down to 1200 m. If there is a significant increase in the salinity of the pore water below this exogenic zone then this is an additional factor which the VDH concept can take advantage of insuring that no appreciable quantities of radionuclides are transported to the surface. Another advantage is that the VDH concept is not as dependent upon the near surface geological conditions as the KBS-3 concept allowing a greater flexibility in the choice of the deployment site or sites.

Multiple deployment sites

The KBS-3 concept requires that once a site is chosen that all waste must be deployed at that site. On the other hand the VDH concept allows for waste to be deployed at two or more repositories if needed or requested. This can be a great advantage if land use problems arise or if during the construction of the site it is determined that not all of the site is suitable for deployment as originally thought.

Adaptability to technological innovations

Since the waste will probably be deployed over a long time period it is possible to take advantage of technological innovations in the field of shaft drilling. This increases the possibility of reducing costs for each hole drilled. It is also possible to change to an entirely different concept during deployment if so required. In the KBS-3 concept it will be more difficult to take advantage of technological innovations as they occur.

Possible economic advantages

Although the large diameter borehole option without rod consolidation of the VDH concept is more expensive (see Chapter 8) than the KBS-3 concept, there are a number of economic advantages with the former. First, the initial investment is considerably less since one or two boreholes can be drilled at a time while in the KBS-3 concept most of the repository must be mined before any waste can be deployed. Secondly, the low initial investment in the VDH concept implies that interest costs should be taken into account when comparing the two concepts.

If it is determined that the maximum allowable temperature in the bentonite can be increased to 150^{o} then rod consolidation may result in an economic advantage is gained with the VDH concept. Rod consolidation in the large diameter holes would allow almost twice as much waste to be deployed in each hole reducing the number of holes required from 35 to 19.

The great flexibility in site selection for the VDH concept may also help in finding a location close to a harbour and thus reduce the cost for transportations to a remote area. Cost savings in the order of 3000 MSEK has been mentioned if a suitable location could be find close to CLAB.

Retrievability

It was initially thought that the VDH concept would not allow the canisters to be retrieved once they had been deployed. Further consideration of this aspect of the concept indicates this not to be the case. There is no reason why the plugged section cannot be drilled or washed out with high pressure fluids. Once the canisters have been reached they could be fished out using overshot tools, a standard oilfield practice. This procedure assumes that the canisters are still in-tact.

8 COST ESTIMATE

Figure 8.1-1 below shows a general view of the handling systems for the radioactive waste products from the Swedish nuclear power stations. The VDH-concept discussed in this report replaces SFL 2, Final Storage For Spent Fuel. SFL 2 as outlined in the KBS-3 report consists of a mined repository at approximately 500 m depth. The waste canisters will be placed in drilled shafts, 7.5 m deep and 1.5 m in diameter. A total of about 40 km of tunnels will be needed for the 5600 canisters with waste.

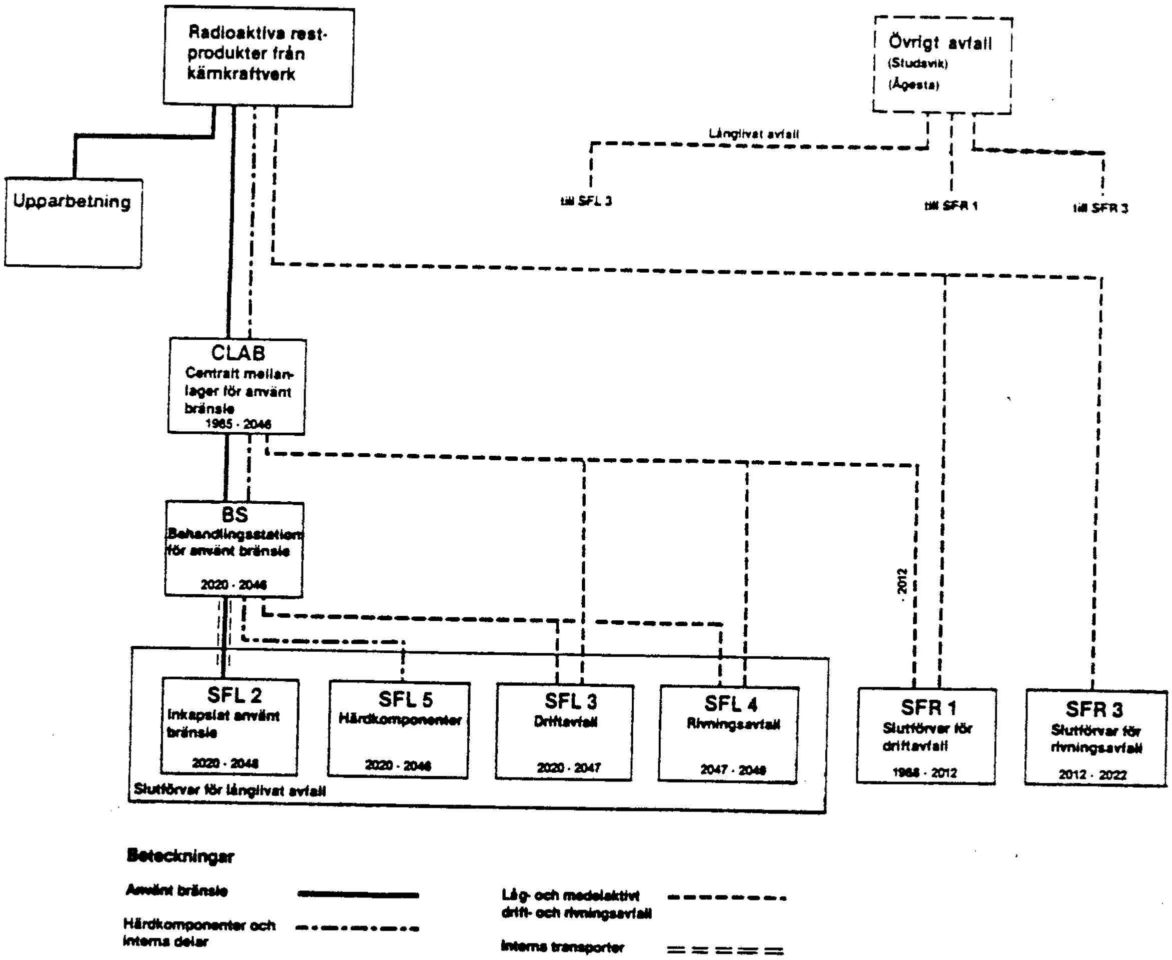


Figure 8.1-1 Handling plan for the radioactive waste products from the Swedish nuclear power stations.

The difference in technology between VDH and SFL 2 will also make some impact on BS, Encapsulation Station. For example, the suggested canisters in the VDH concept will be slimmer with less waste and thus the total number will increase. This report does not include any analysis of BS and the cost is, therefore, assumed to be the same as presented in the SKB report, Kostnader för kärnkraftens radioaktiva restprodukter, juni 1989.

A detailed cost analysis for the VDH boreholes including planning, drilling, deployment and sealing is presented in Appendix 4. This cost estimate must be judged as tentative. Most of the cost figures are based on American experience and a great deal of new approaches in drilling technology are suggested. The drilling cost also includes technical prestudies and detailed design and cost for site investigation. The cost for the Preliminary Study should be excluded since this work will be incorporated in SKB's R and D budget and replace planned work for a mined repository. In order to present a total cost for the VDH concept that with all certainty will not be exceeded the following costs have been added (the same figures have been used for the KBS-3 concept):

- Administration, procurement, QA etc.	15%
- SKB project cost, contractors revenue	8%
- Contingency	15%

Total cost for each borehole, option A large diameter boreholes (0.8 m) to 4 km depth is:

- Drilling cost	251 MSEK
- Waste deployment	137 MSEK
- Administration, procurement, QA etc. 0.15 (251 + 137)	58 MSEK
- SKB project cost, Contractors revenue 0.08 (251 + 137 + 58)	36 MSEK
- Contingency 0.15 (251 + 137 + 58 + 36)	72 MSEK
Total cost each borehole	554 MSEK

The total cost for two of the VDH concepts discussed in this report is presented in the table below.

Table 8.1-1 Total cost for the VDH storage

Concept	Number of boreholes	Cost each borehole MSEK	Total cost MSEK	Present value year 1990, 2.5%, MSEK
Large diameter boreholes. Canisters with complete fuel elements	35	554	19 390	6 850
Large diameter boreholes. Canister with rod consolidation T_{max} 150°C	19	554	10 526	3 719

The above total cost is higher compared to the cost for SFL 2 (KBS-3) that has been estimated to 6314 MSEK. In comparison the total cost for the Swedish waste systesm has been estimated to approximately 43 000 MSEK.

For a total comparison with the KBS-3 concept it is essential to consider the cash flow and thus the present value discounted to year 1990. The construction phase for SFL 2 is planned to start 8 years before the first deployment of waste. This implies that a fairly large part of the total cost is invested early and before deployment. For the VDH concept, however, the drilling, deployment and sealing are integrated in one operation after each other during 3 years. Present value for the VDH concept year 1990 is also presented in Table 8.1-1. The estimate presented in Appendix 6 is based on a 27 years deployment period between 2020 and 2047 and an interest rate of 2.5%.

The total number of drilled boreholes each year will be dependent on the number of drilling rigs used. The numbers could easily match any requested total deployment period, see Table 8.1-2.

Table 8.1-2 Needed time for deployment with different number of drilling rigs

Number of drilling rigs	Borehole per year	Needed time for 35 boreholes (years)	Needed time for 19 boreholes
1 rig for both drilling and deployment	0.4	87.5	47.5
2 rigs for both drilling and deployment	0.8	43.7	23.8
1 rig for drilling 1 rig for deployment (smaller)	0.67	52.2	28.4
2 rigs for drilling 2 rigs for deployment (smaller)	1.34	26.1	14.2

(A deployment period of 27 years has been choosen in this report both for 35 and 19 boreholes.)

For a comparison with the KBS-3 concept it is also necessary to make a cost analysis for the canisters which are included in BS, Treatment Station For Spent Fuel. In this context it is also vital to analyse BS together with the VDH in order to minimize the total cost. For example, in the KBS concept the deployment period of 28 years is based on the capacity to manufacture canisters. Presented below is the total number of canisters for the VDH and KBS-3 concept together with an estimate of needed materials. The writers of this report have not studied the BS and, therefore, should any cost analysis of this part of the operation be desired it must be performed by the SKB specialists.

Table 8.1-3 presents the total number of canisters and the volume for copper and lead for the VDH and KBS-3.

Table 8.1-3 Total number of canisters for VDH and KBS-3 concept

Concept	Weight uranium ton	Number of canisters	Volume copper m^3	Volume free space + void filler
VDH Canisters with complete fuel elements	7 800	10 367	3 755	5 201
VDH Canisters with rod consolidation	7 800	5 518	1 999	2 768
KBS-3	7 656	5 637	5 998	6 471
	7 800	5 743	6 111	6 593

9. SUGGESTED FUTURE WORK

9.1 Continued review of results from deep boreholes in crystalline rock

The drilling of deep boreholes is a rather expensive operation. By reviewing work done at other sites a great deal of information can be obtained quite cheaply. In addition, if it is decided to drill a deep investigatory well it is important to be aware of what other groups have done and what problems they have had. The reviewing done up to now has been of a qualitative nature consisting of basically finding out what wells have been drilled in crystalline rock and what the most important results were. In the future it should be possible to obtain the actual data from some of the wells and begin building a database of results from deep wells in crystalline rock. If further studies of the very deep borehole concept for radioactive waste disposal are being considered then either the current qualitative or a more quantitative review program should be included.

9.2 Drilling related research

The following topics are suggested as being appropriate for the drilling, casing and in hole activities of the proposed concept for deep disposal:

- Seek opportunities to construct or observe the construction of a deep drilled shaft in granite using the drilling method proposed. It may be possible to use such a method in any access shafts for any other planned underground facilities in Sweden.
- Construct models of the proposed lining/casing system and simulate the seal construction in laboratory experiments to help verify the feasibility of the process and if possible measure the change in permeability.
- Prepare detailed design options for the high void ratio liners.
- Construct scale models of the proposed liner designs to assist in the debate on the method.
- Construct a full scale prototype section of liner or casing to the preferred design option. If possible check the loading capacities.
- Support further research work into the phenomenon and control of stress breakout in granite basement rock.

9.3 Plugging and sealing

The pursued study of the VDH concept that is presented in this report tends to support earlier conclusions with respect to the possibility of attaining good sealing of the plugged part and effective cut-off of long-range axial flow and radionuclide migration within the deployment zone. Thus, the use of large diameter boreholes, that is the major new feature of the concept, yields very considerable improvement of the isolation ability of the various clay-based buffers and sealing materials.

The feasibility and various functions of the clay components still need to be investigated both with respect to the material properties and to their thermo-mechanical behavior for evaluation of the potential of the VDH concept in its proposed form. The major issues of the sealing functions are outlined below.

1. How is the fracture network of the near-field rock in deep boreholes affected by stress release, and by the presence and intrusion of drilling and deployment muds?

   Means of investigation: UDEC- and 3Dec-type theoretical analyses.

2. What is the optimum composition of the deployment mud to bring it in a fluid condition at the expulsion, and to stiffen thixotropically at rest, as well as to become sufficiently low-permeable on consolidation?

   Laboratory investigations: Viscometry, shear tests, oedometer tests all with special respect to the influence of salt water and temperature.

   Field tests: Large-scale model tests of application technique. Consolidation tests in rock for experimental determination of rate of maturation of plug compounds with special respect to heat and salinity of the water.

3. Determination of the stability of the system of canister/bentonite sets in deployment mud in the application phase and after plugging with respect to settlement and fracturing of unsaturated bentonite blocks.

   Theoretical analyses: Derivation and application of thermo-mechanical model.

   Experimental: Small scale, accelerated load tests.

   Field tests: Loading tests in large boreholes in Stripa or similar test site.

4. Development of complete model of the thermo-mechanical behavior and long-term performance with respect to chemical alteration.

5. Full-scale testing in deep hole.

9.4 Modelling of water convection

The modelling of water flow in the rock due to deployment of the radioactive waste done up to now is to a certain extent inadequate. The current models are all 2-D and several simplifications have been made. The following studies are proposed:

1. 3-D transient modelling of heterogeneous rock with waste deployed in the interval 2 to 4 km. The modelling would allow the effects of saline water to be incorporated directly.

2. Literature study of flow in porous media versus flow in fractures and if the assumption of flow in porous media in the modelling done is acceptable.

3. Experimental studies to verify if the conclusions drawn from point 2 are valid.

Points 2 and 3 are probably being investigated elsewhere, but it is important to share these results with those involved with the very deep borehole concept since they are also relevant to it.

9.5 Pilot study to determine depth to saline water using electromagnetic methods

Due to the significance that saline water may have on the quality assurance of the very deep borehole concept it is suggested that studies be done to investigate if the depth to highly saline water can be determind by surface measurements. In the Siljan Ring area surface magnetotelluric investigations indicated the presence of more saline fluid in the 6-10 km depth range in the vicinity of Gravberg. Subsequent drilling and testing of the Gravberg-1 borehole resulted in that highly saline fluids were produced from depths below 6 km which agrees very well with the surface predictions. The magnetotelluric method uses natural sources which have rather long wavelengths so the resolution is quite poor. By using controlled higher frequency artificial sources the resolution can be improved considerably. It is recommended that field tests be carried out in an area where the salinity is known to increase rapidly at a distinct depth to test the viability of mapping salinity boundaries in crystalline rock. If the results are positive then the method should be employed at the location of a 3.0 km investigatory borehole.

9.6 A 3.0 km borehole to test the geological assumptions

The very deep borehole concept is based upon the rock below 1-2 km being significantly less fractured than the rock above it. Surface seismic investigations and the drilling of the Gravberg-1 borehole indicate this to be the case in most parts of Sweden. A very brief comparison of the macrofracture density between the Siljan Ring area and Äspö indicates the rock in the upper 1000 meters at Äspö to be as or more fractured than the rock at Siljan (Juhlin 1990). Surface investigations focusing on the rock from 1000-5000 meters coupled with the drilling of 3.0 km deep borehole at or near Äspö would give extremely important results concerning the validity of the geological assumptions. If the fracture density decreases dramatically with depth at Äspö then serious consideration must be given to the very deep borehole concept. A deep borehole at Äspö also has the advantage in that the depth to highly saline water can probably be determined since it is located on the Baltic Coast where the depth is expected to be shallower than inland (Nurmi et al, 1988). Even if the very deep borehole concept is not considered as a viable alternative to the KBS-3 concept or other concepts deep boreholes are still important. For instance, if other concepts are considered or it is decided that KBS-3 should be placed at a deeper level the returns from drilling a deeper investigatory hole will be very valuable. Factors such as fracture density and salinity of the groundwater will greatly influence circulation patterns of the water in the vicinity of any repository. It would appear to be of utmost importance to determine if decreased fracture density and increased salinity with depth is a general feature of the Baltic Shield and at what depths the transitions occur.

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1-D MODELLING OF WATER CONVECTION

1. 1-D MODELLING OF WATER CONVECTION

1.1 Flow along the borehole

1.1.1 General

The heat from the radioactive waste will lower the density of water by approximately 0.64 kg/m^{3}K, see Figure A3.1-1. If it is assumed that the borehole is a good hydraulic conductor (due to breakouts), the total potential deficit along the heated part of the borehole will become as below.

$$\Delta h = L \cdot \frac{0.64 \cdot \Delta T}{\rho_{T_O}}$$

This potential deficit generates a convection cell around the borehole as shown in Figure A3.1-2. The following assumptions have been made in the calculations. (K = Hydraulic conductivity of the bedrock.)

$K = 1 \cdot 10^{-10}$ m/s
$d = 4000$ m
$L = 2000$ m
$r_w = 0.122$ m

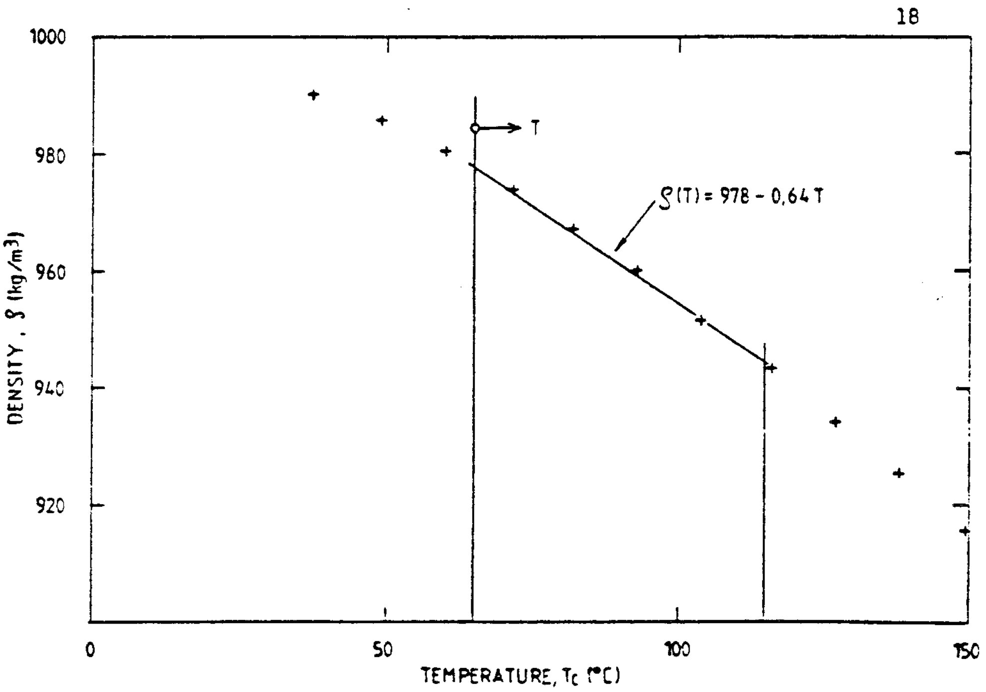


Figure 1.1-1 The density of water

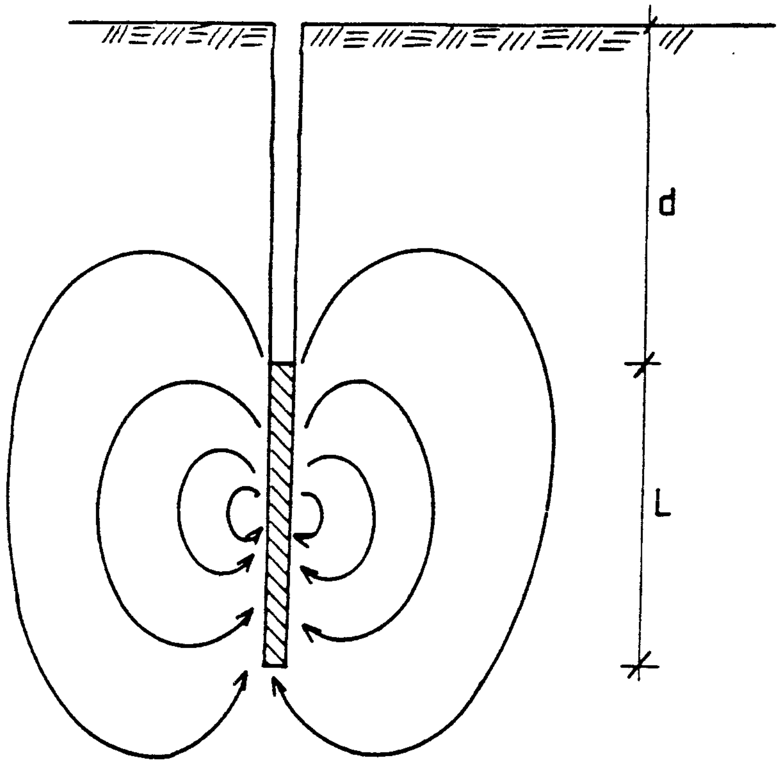


Figure 1.1-2 Principal configuration of the convection cell around the borehole

At a depth of 5 km, the natural temperature can be estimated to about $T_0 = 90^oC$. This gives a density of 977 kg/m^3 at a depth of 5 km. The temperature increase in the borehole is dependent on the spacing between the boreholes and the heat generation rate. Here we assume that the temperature increase is small, as for one borehole. T is assumed to be 30^oC. The potential difference along the borehole can be expressed as below where z is the coordinate along the borehole starting from the middle of the borehole and is positive upwards, see Figure A3.1-3.

Formula $h = \frac{0{,}64 \times 30}{977} \cdot z = 0{,}0197 \cdot z = a \cdot z$

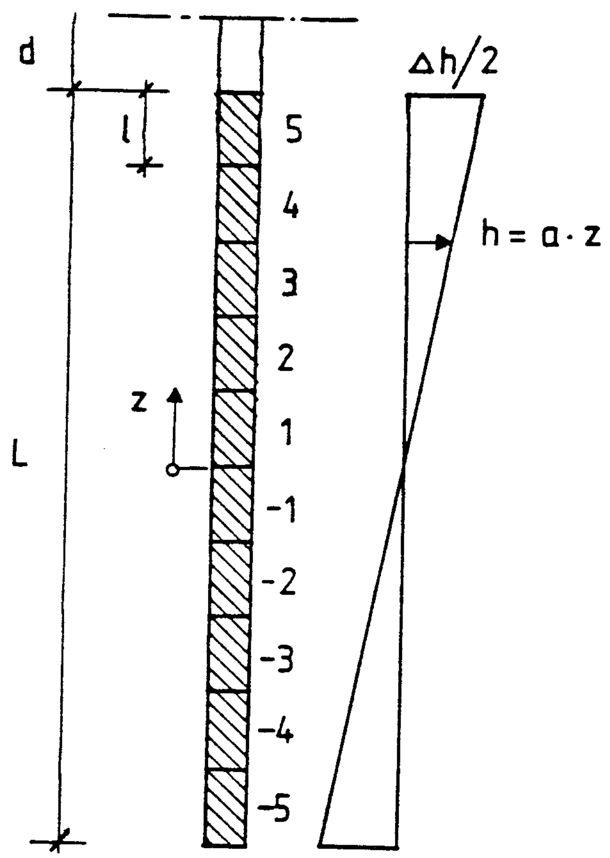


Figure 1.1-3 Principal figure of potential difference (h) along the borehole and the element division -5 to 5 along the borehole used in the calculations.

The amount of water flowing to and from the borehole can now be calculated. From Figure 1.1-3 it can be concluded that

$Q_n = -Q_{-n}$

where n stands for the element indices in Figure 1.1-3. It is therefore only necessary to study one half of the borehole. The potential for each element along the borehole can be written as

(1.1)

Equation (1.1) can now be used for each element in Figure 1.1-3 which gives a system of equations from which Q_n can be calculated.

1.1.2 One convection cell

In this case L = 2000 m and the element division is in accordance with Figure A3.1-3 which gives l = 200 m. After some simplifications the equation system is as follows:

$$900 \cdot b = \frac{Q_1}{1600} - \frac{Q_1}{2000} + \frac{Q_2}{1200} - \frac{Q_2}{2400} + \frac{Q_3}{800} - \frac{Q_3}{2800} + \frac{Q_4}{400} - \frac{Q_4}{3200} + \frac{Q_5}{27.01} - \frac{Q_5}{3600}$$

$$700 \cdot b = \frac{Q_1}{1200} - \frac{Q_1}{1600} + \frac{Q_2}{800} - \frac{Q_2}{2000} + \frac{Q_3}{400} - \frac{Q_3}{2400} + \frac{Q_4}{27.01} - \frac{Q_4}{2800} + \frac{Q_5}{400} - \frac{Q_5}{3200}$$

$$500 \cdot b - \frac{Q_1}{800} - \frac{Q_1}{1200} + \frac{Q_2}{400} - \frac{Q_2}{1600} + \frac{Q_3}{27.01} - \frac{Q_3}{2000} + \frac{Q_4}{400} - \frac{Q_4}{2400} + \frac{Q_5}{800} - \frac{Q_5}{2800}$$

$$300 \cdot b = \frac{Q_1}{400} - \frac{Q_1}{800} + \frac{Q_2}{27.01} - \frac{Q_2}{1200} + \frac{Q_3}{400} - \frac{Q_3}{1600} + \frac{Q_4}{800} - \frac{Q_4}{2000} + \frac{Q_5}{1200} - \frac{Q_5}{2400}$$

$$100 \cdot b = \frac{Q_1}{27.01} - \frac{Q_1}{400} + \frac{Q_2}{400} - \frac{Q_2}{800} + \frac{Q_3}{800} - \frac{Q_3}{1200} + \frac{Q_4}{1200} - \frac{Q_4}{1600} + \frac{Q_5}{1600} - \frac{Q_5}{2000}$$

$$b = 2\pi \cdot 10^{-10} \cdot a$$

$$\begin{bmatrix} 0.125 & 0.417 & 0.893 & 2.188 & 36.75 \\ 0.208 & 0.75 & 2.083 & 36.667 & 2.188 \\ 0.417 & 1.875 & 36.593 & 2.083 & 0.893 \\ 1.25 & 36.190 & 1.875 & 0.75 & 0.417 \\ 34.593 & 1.25 & 0.417 & 0.208 & 0.125 \end{bmatrix} \cdot \begin{bmatrix} Q_1 \\ Q_2 \\ Q_3 \\ Q_4 \\ Q_5 \end{bmatrix} = \begin{bmatrix} 5.654 \\ 4.398 \\ 3.142 \\ 1.885 \\ 0.698 \end{bmatrix} \cdot 10^{-4} \cdot a$$

The solution of this equation system gives the water flow Q_n:

$$\begin{aligned} Q_1 &= 0.0145 \cdot 10^{-4} \cdot a \\ Q_2 &= 0.0439 \cdot 10^{-4} \cdot a \\ Q_3 &= 0.0740 \cdot 10^{-4} \cdot a \\ Q_4 &= 0.1061 \cdot 10^{-4} \cdot a \\ Q_5 &= 0.1452 \cdot 10^{-4} \cdot a \end{aligned}$$

$$Q_{TOT} = 0.3837 \cdot 10^{-4} \cdot a$$

With a = 0.0197, Q_{TOT} becomes $7.56 \cdot 10^{-7}$ m^3/s or 65 l/day. Q_{TOT} flow along the borehole where z = 0.

1.1.3 Six convection cells

If bentonite plugs are inserted between the radioactive waste in the borehole as shown in Figure 1.1-4, the water flow will decrease. The hydraulic conductivity of the bentonite is approximately 10^{-12} m/s. The assumption for the calculations below is that the bentonite plugs tighten the borehole (no water flow along the borehole through the bentonite plug). No consideration has been given to the fact that there might be flow in the bedrock between the different parts of the borehole sectioned off by the bentonite due to a potential difference between these parts.

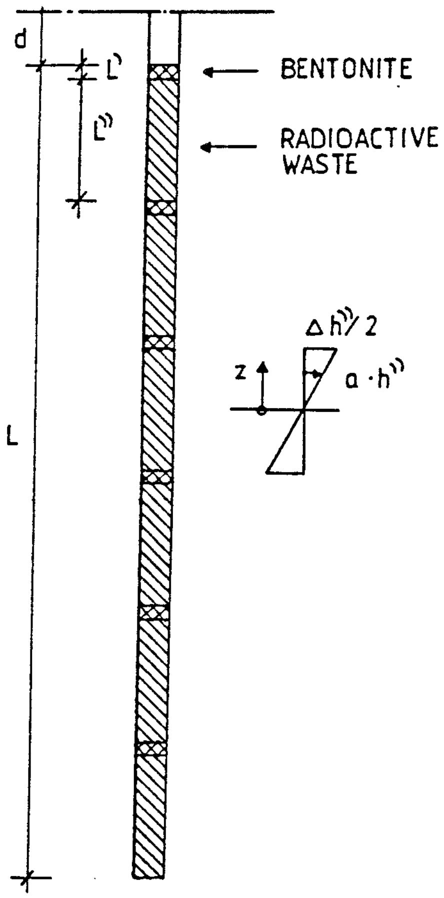


Figure 1.1-4 Principal configuration of the borehole with bentonite plugs

In this case the geometry is defined as below, see Figure 1.1-4.

d = 3000 m
L = 3000 m
L' = 50 m
L" = 450 m

L" is divided into 10 elements which gives $l = 45$ m in equation (1.1). Solving the equation system gives the result below.

$Q_1 = 0.88 \cdot 10^{-7} \cdot a$

$Q_2 = 2.84 \cdot 10^{-7} \cdot a$

$Q_3 = 4.53 \cdot 10^{-7} \cdot a$

$Q_4 = 6.54 \cdot 10^{-7} \cdot a$

$Q_5 = 9.11 \cdot 10^{-7} \cdot a$

$Q_{TOT} = 2.390 \cdot 10^{-6} \cdot a$

This gives $Q_{TOT} = 4.71 \cdot 10^{-8}$ m^3/s = 4.1 l/day.

1.1.4 Conclusions

When the borehole is a good conductor, the maximum water flow along the borehole becomes 65 l/day when the free flow length along the borehole is 2000 m according to Section 1.1.2. In Section 1.1.3 the free flow length was decreased to 450 m, which made the maximum water flow along the borehole decrease to 4.1 l/day. It can be concluded that bentonite plugs will decrease the flow rate, but it is difficult to say whether these flow rates are a problem or not.

These water convection estimates should be repeated when the final design for sealing within the deployment zone is settled. A design with a continuous bentonite seal is foreseen. This design will reduce the flow rates considerably. For a more accurate water convection estimate a better hydraulic model for the rock is needed. The hydraulic conductivity used is based on measured data from Gravberg-1, which probably represent the most permeable zones. The average hydraulic conductivity will definitely be lower.

A detailed estimate should also include the effect from one or more fracture zones with higher hydraulic conductivity crossing the borehole storage. The cost of such modelling will be fairly high and it has not been possible to implement it in this study.

1.2 Flow when the borehole is continuously sealed with bentonite

It may be possible to seal the borehole with bentonite. If breakouts and fractures close to the borehole are filled with bentonite the borehole will no longer be a good conducter as described in Section 1.1. (The hydraulic conductivity of bentonite is assumed to be 10^{-10} - 10^{-12} m/s and the bedrock 10^{-10} m/s.) Because of this, the flow pattern will change compared to that in Section 1.1. Outside the borehole heat from the radioactive waste will lower the density of the water and thus cause water convection. In Figure 1.2-1 one stream tube in a convection cell is shown. The water flow in this streamtube is estimated below.

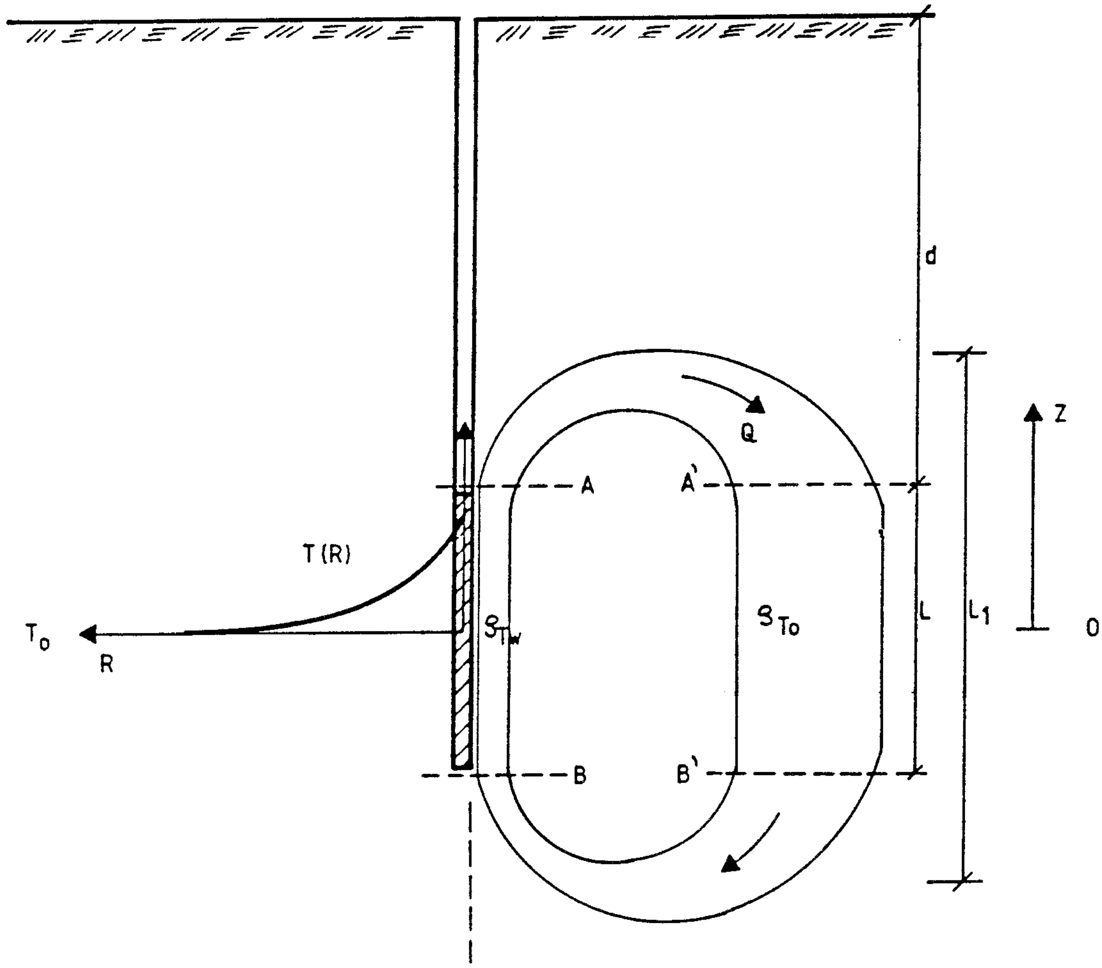


Figure 1.2-1 Principal configuration of a stream tube in a convection cell and the radial temperature T(R). Radioactive waste is shown by the shaded part of the borehole.

The waterflow Q in the stream tube for Z = O is

$$Q = K_w \cdot i_w \cdot A_w = K_o \cdot i_o \cdot A_o \tag{1.2}$$

K: hydraulic conductivity

i: gradient

A: area

index w: part of stream tube close to borehole

index o: part of stream tube with Z > O and far away from borehole

The gradient i_w can be estimated in the following way. The temperature within the stream tube between A and B is T_w and between A' and B' T_o. Between A and A' and between B and B' the temperature decreases from T_w to T_o. If the stream tube is closed at A, the pressure at B can be calculated to be

$$P_B + \rho_{Tw} \cdot g \cdot L = P_A + P_{A-A'} + \rho_{To} \cdot g \cdot L + P_{B-B'}$$

$$P_B = g \cdot L(\rho_{To} - \rho_{Tw}) + P_A + P_{A-A'} + P_{B-B'} =$$

$$= g \cdot L \cdot \Delta\rho + P_A + P_{A-A'} + P_{B-B'} \tag{1.3}$$

If the stream tube and the temperature field are symmetric around Z = O then

$$P_{A-A'} = P_{B-B'} \tag{1.4}$$

The magnitude will be

$$P_{A-A'} = \int_A^{A'} \rho(T) \cdot g \cdot dz < (\rho_{To} - \rho_{Tw}) \cdot (L_1 - L) g/2 =$$

$$= \Delta\rho \ (L_1 - L) g/2 \tag{1.5}$$

If we assume that $K_w \ll K_o$ then the gradient between A and B can easily be calculated.

The hydraulic head (h) at points A and B is

$$h_A = \frac{P_A}{\rho_{To} \cdot g} + Z_A$$

$$h_B = \frac{P_B}{\rho_{To} \cdot g} - Z_A$$

$$\Delta h = h_B - h_A = \frac{P_B - P_A}{\rho_{To} \cdot g}$$

$$\Delta h = L \cdot \frac{\Delta\rho}{\rho_{To}} + \frac{P_{A-A'} + P_{B-B'}}{\rho_{To} \cdot g} = L \cdot \frac{\Delta\rho}{\rho_{To}} + C \qquad (1.6)$$

From equation 1.5, C can be estimated to

$$C < \frac{\Delta\rho}{\rho_{To}} (L_1 - L) \qquad (1.7)$$

C is dependent on the size of the convection cell and the temperature change between A - A' and B - B'. The temperature around the borehole decreases rather fast with increasing R or $Z>Z_A$ or $Z<Z_B$. Therefore C should be much less rather than just less than in the expression in equation 1.7. In the calculations below, C is approximated to C = O.

The gradient between A and B can then be estimated to

$$i'_w = \frac{\Delta h}{L} = \frac{\Delta\rho}{\rho_{To}} \qquad (1.8)$$

The gradient i'_w should be greater than the true gradient i_w because K_w K_o and the total hydraulic head is distributed along a length >L.

$$i_w < i'_w \qquad (1.9)$$

The water flow in the stream tube will then become

$$Q = i_w \cdot K_w \cdot A_w \qquad (1.10)$$

$$q_w = Q/A_w = i_w \cdot K_w < i'_w \cdot K_w = \frac{\Delta\rho}{\rho T_o} \cdot K_w$$

From Section 1.1 the following data can be read

$$\frac{\partial \rho}{\partial T} = 0.64 \text{ kg/(m}^3 \text{ K)}$$

$$\rho_o = 977 \text{ kg/m}^3$$

$$T_o = 67^oC$$

$$\Delta T_w = T_w - T_o = 30^oC$$

$$K_w = 10^{-10} \text{ m/s}$$

These data give the water flow:

If the temperature increase around the borehole is about 30°C the water flow near the borehole due to convection should not become more than about 0.06 l/ (m^2/year).

1.3 Conclusions

When the borehole is a good conducter, the maximum water flow along the borehole becomes 65 l/day when the free flow length along the borehole is 2000 m as in Section 1.1. In Section 1.2.2 the free flow length was decreased to 450 m, which made the maximum water flow along the borehole decrease to 4.1 l/day. It can be concluded that bentonite plugs will decrease the flow rate.

If the entire borehole is sealed with bentonite, it would seem possible that flow paths close to the borehole due to breakouts will be more or less eliminated. But there will still be a water flow in the bedrock due to convection. The flow close to the borehole has in section 1.2 been estimated to 0.06 l/(m^2/year).

It can then be concluded that the effective sealing of the borehole will have a great influence on the magnitude of the water flow close to the borehole. The flow rate will become very low if the hydraulic conductivity close to the borehole is as low or lower than the hydraulic conductivity of the surrounding bedrock.

BOUNDARIES FOR THE RISE OF THE HALOCLINE BY THERMAL CONVECTION

BOUNDARIES FOR THE RISE OF THE HALOCLINE BY THERMAL CONVECTION

Gunnar Gustafson, VIAK AB

Introduction

The storage of spent nuclear fuel in deep boreholes gives a thermal convection around the boreholes because of the heat generation in the fuel. If there is a halocline in the groundwater the convection cell may cause an upward bulge on the halocline or make the halocline to burst. If the bulge is stable this prevents the thermal flow to reach the surface and thus provides an additional barrier. In this paper some bounding calculations for the rise of the halocline due to thermal convection are presented.

The stability of the halocline

Because of the difference in density between fresh water at surface and a saline groundwater at depth a stabile stratification will be the result. Since fresh and saline water are completely miscible in theory a complete mixing between the fluids will be result of molecular dispersion at infinite time. However, experience has shown that a stable stratification is the rule and that the halocline can be considered as a consistent boundary layer.

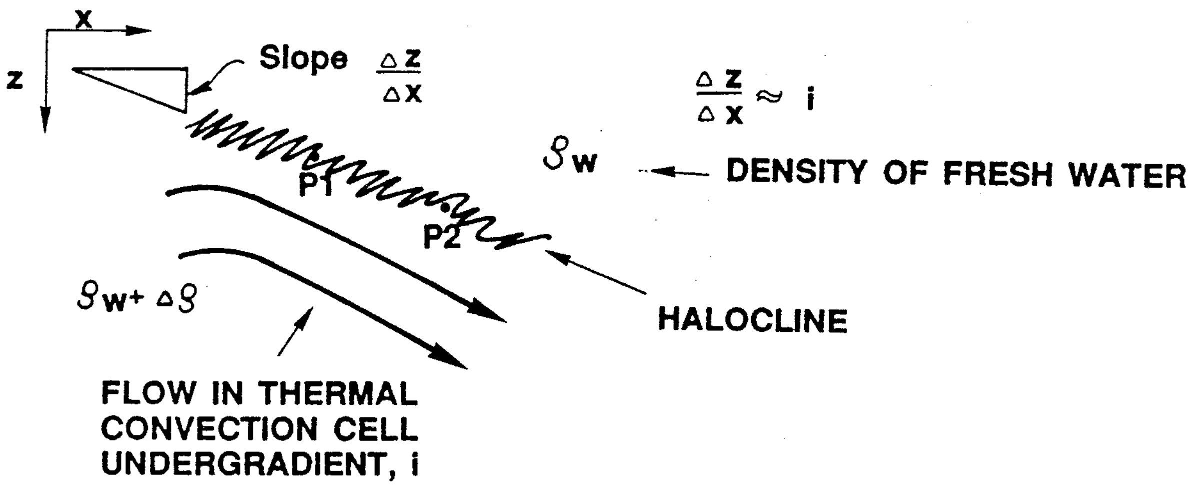


Figure 1 Equilibrium at the halocline

In *figure 1* the conditions at the halocline are shown. At point P1 (x,z) the halocline is assumed to be in equilibrium as well as at P2 (x+Δx,z+Δz). This gives the equilibrium equation at P2:

$$p_1 + \Delta z(\rho_\infty + \Delta\rho)g - i \cdot \Delta x(\rho_\infty + \Delta\rho)g = p_1 + \Delta z \cdot \rho_\infty \cdot g \tag{1a}$$

$$\Delta z/\Delta x = i(\rho_\infty + \Delta\rho)/\Delta\rho \tag{1b}$$

The gradient is in turn determined by the groundwater flux, q, and the hydraulic conductivity, K_x, as:

$$i = \frac{q}{K_x} \tag{2}$$

The hydrostatic approximation

If the flow parallel to the halocline is restricted by at decrease of the conductivity, K_x, it is evident that the gradient of the halocline also grows. If, K_x is reduced to zero it means the steepest possible boundary layer and consequently the maximum bulge of it. This is equivalent to a hydrostatic rise of the layer where only vertical movements of the groundwater are allowed. which thus can be used as a bounding calculations, see *figure 2*.

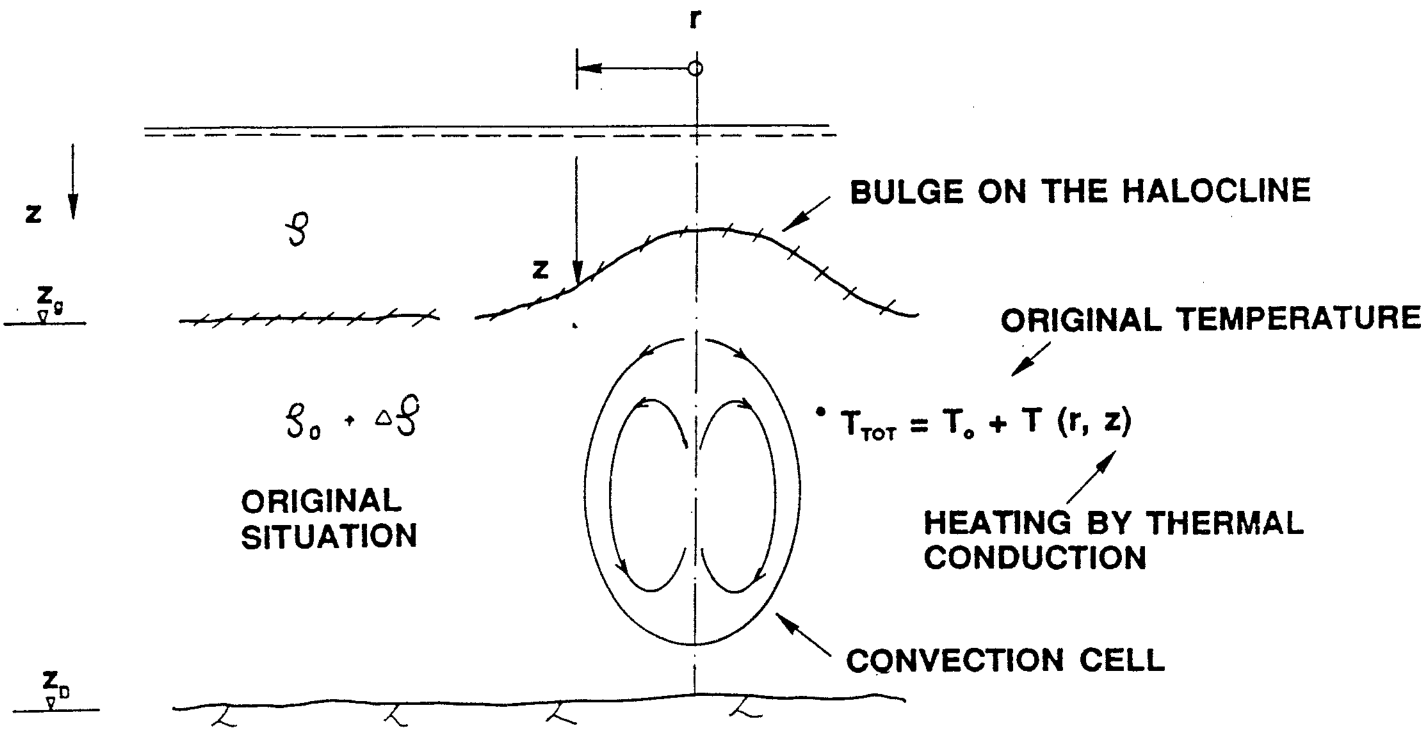


Figure 2 The derivation of the bulge of the halocline

The temperature in the system T_{TOT} is assumed to be determined by the original temperature, T_o, and a heating, T(r,z), determined by heat conduction from the deposition only. It is also assumed that the density of the water can be determined by:

$$\rho_\infty = \rho_o - T(r,z) \cdot R \qquad \text{(fresh water)} \qquad (3a)$$
$$\rho_\infty = \rho_o + \Delta\rho - T(r,z)R \qquad \text{(saline water)} \qquad (3b)$$

Pressure equivalence at some depth, Z_D, below the volume of influence from the deposition borehole gives:

$$p_D/g = \int_o^{z_D} \rho_\infty \, dz \qquad (4a)$$

$$z_g\rho_o + (z_D - z_g)(\rho_o + \Delta\rho) = \int_o^{z} (\rho_o - T \cdot R)dz + \int_z^{z_D} (\rho_o + \Delta\rho - T \cdot R)dz \qquad (4b)$$

$$(z_g - z) = \frac{R}{\Delta\rho} \int_o^{z_D} T(r,z)dz \qquad (4c)$$

Thus the maximum rise of the halocline $(z_g\text{-}z)$ can be determined by equation (4c).

An approximate solution to the temperature integral

The influence on the temperature by thermal conduction can approximately be determined by superposition of point sources, *figure 3.*

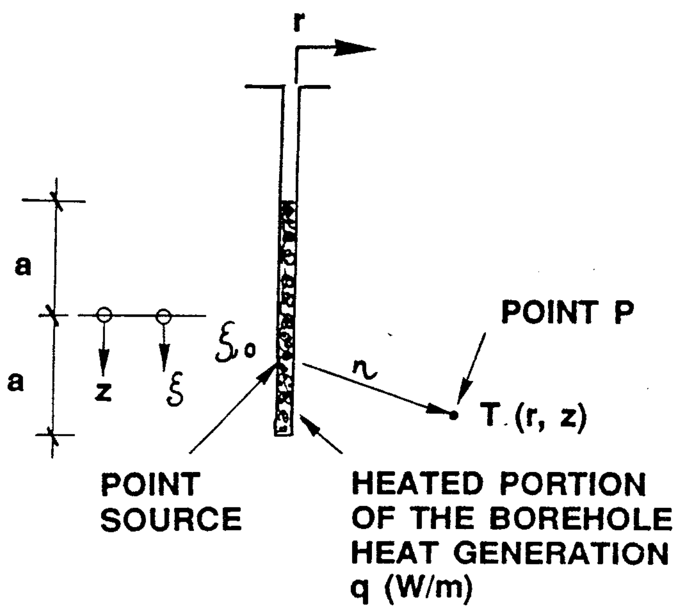


Figure 3 The thermal influence from the borehole.

The strength of each point source is assumed to be:

$$dq = q \cdot d\xi \tag{5a}$$

This give an influence on the point P as:

$$dt = \frac{q d\xi}{4\pi\lambda\eta} = \frac{q d\xi}{4\pi\lambda \;\; (z-\xi)^2 + r^2} \tag{5b}$$

where λ is the heat conductivity of the rock. Thus the integrated temperature influence will be:

$$T(r,z) = \int_{-a}^{a} \frac{q d\xi}{4\pi\lambda \cdot \sqrt{(z-\xi)^2 + r^2}} = \frac{q}{4\pi\lambda} \ln\left(\frac{z+a + \sqrt{(z+a)^2 + r^2}}{z-a + \sqrt{(z-a)^2 + r^2}} \right) \tag{5c}$$

Now the thermal integral can be determined as:

$$\int_{0}^{z_D} T(r,z)dz = \frac{q}{4\pi\lambda} . I \tag{6a}$$

$$I = \int_{0}^{z_D} \ln\left(\frac{z+a + \sqrt{(z+a)^2 + r^2}}{z-a + \sqrt{(z-a)^2 + r^2}} \right) dz \tag{6b}$$

Equations (6a) and (6b) are derived under the assumption of a steady taste. The solution of the transient correspondence is rather complicated. However, it can be shown that the radius of thermal influence can be approximated to be

$$r_T = \frac{2.25\lambda t}{\rho c} \tag{7}$$

In this equation t is the heating time and ρc is the volumetric heat capacity. In order not to overestimate the integral the integration limits can be set to +/-($a+r_T$)

Calculation results

The rise of the halocline was calculated for a number of situations, see *table 1.* The basis is a deployment zone with an initial heat generation of 100 W/m and a borehole radius of r = 0.4 m. The decline of the heat generations follows the table given on page 83 in SKB 88-56. The calculations were made with a density difference of $\Delta\rho$ = 25 kg/m^3, corresponding to ocean water and $\Delta\rho$ = 100 kg/m^3 corresponding to a brine. Calculations were also done for three distances from the borehole axis.

If the results are scrutinized it is clear that the potential rise of the halocline at high heat generation is fairly large. However, it is also clear that since the heat generation decreases rapidly already after about 100 years reasonable figures are obtained. Furthermore there are definite limits for how far the halocline can migrate during this period. It is also unclear how much on the safe side the hydrostatic calculations method is.

In short, however, this small study indicates that density stratification of the deep groundwaters around a deposition borehole for spent nuclear fuel will be an efficient barrier for spreading of radioactive constituents after a period of about 100 years.

Table 1 Calculated bounds for the rise of a halocline above a deposition borehole for some different times and distances.

λ = 3.0 W/mK, ρc = 2.2·10^6 J/m^{3}K, R = 0.64 kg/m^{3}K, r_w = 0.4 m

Density diff. $\Delta\rho$(kg/m^3)	**Heating time t, (years)**	**Radius of influence r (m)**	**Heat generation q (W/m)**	**Rise of halocline (z-z$_t$) (m) Distance r=0.4 m**	**r=4 m**	**r=40 m**
25	0	1	110	2454	1766	1069
	10	32	50	2143	1542	939
	100	100	18	781	567	349
	∞(40.000)	2000	0.3	15	11	8
100	0	1	110	614	442	267
	10	32	50	537	385	235
	100	100	18	195	142	87
	∞(40.000)	2000	0.3	4	3	2

TENTATIVE SPECIFICATION FOR DRILLING RIG AND ANCILLARY EQUIPMENT FOR LARGE DIAMETER OPTION

TENTATIVE SPECIFICATION FOR DRILLING RIG AND ANCILLARY EQUIPMENT FOR LARGE DIAMETER OPTION

The following tentative inventory for a suitable drilling rig and associated equipment is intended as a guide only. This schedule is for the 4000 m deep large diameter option. It is not a complete list, but it does give a good indication of the type of rig and equipment that will be required for such a programme.

The full equipment requirements can only be determined after the borehole design is finalised. In addition, actual equipment will depend on availability at the time and the changes in technology that will undoubtedly take place over the period from now until any field operations commence.

Derrick or mast	: 2 000 000 lb nominal capacity derrick or mast with minimum 1 500 000 hook load capacity with 12 lines. Racking board to rack 2000 m of 13-3/8 in drill pipe plus 2000 m of 5 in drill pipe. Derrick will be required to handle flat bottom bits and/or tapered bits up to 2 400 mm diameter and 25 m sections of 1 300 mm casing as well as 12 m lengths of 2 000 mm diameter casing.
Substructure	: Compatible with derrick or mast furnished with floor designed to accept tools and materials required for the work programme. Minimum set back load 1 000 000 lb plus 1 500 000 lb rotary table load simultaneously. Substructure must have movable rotary and spreader beams to facilitate handling large diameter hole tools.
Catwalk and V-door	: As required for rig preferably complete with pipe handling device.
Crown block	: Minimum 7-sheave, grooved for 2 in wire rope 900 ton working capacity.
Travelling block	: Minimum 6-sheave grooved for 2 in wire rope 750 ton working capacity.
Hook	: 750 ton with elevator links of the same capacity.
Rotary table	: Minimum rating 250 000 ft lb torque with a speed range of 3-30 rpm.
Drawworks	: single drawworks with minimum 2 500 hp rating with 1 500 000 lb hoisting capacity.

Drawworks power	: DC traction motors through SCR units from AC power generated either on site or from utility power (mains).
Auxiliary brake	: Dynamic electric brake with controls or hydromatic brake with controls matching the drawworks.
Pipe tongs	: B-J Type DD or equivalent tongs for 13-3/8 in drill pipe and 3 in manual and power tubing tongs for making up and breaking out reverse circulation air line.
Handling tools	: Two set side door 750 ton capacity elevators for 13-3/8 in drill pipe complete with elevator links. 750 ton spider and slips for 13-3/8 in drill pipe and 16 in drill collar mandrel. 3 in side door tubing elevator for air line complete with links. Spider and slips for 3 in air line tubing.
Drilling line	: 2 in diameter, 6 x 19 Seale construction, preformed, right lay, new, EIPS quality, IRC (6-strands, 19 wires per strand with independent wire rope core, extra improved plough steel, 198 ton breaking strength).
Weight indicator	: Suitable temperature compensated accurate weight indicator with tie-down anchor of a type and kind suitable to the indicator and it's intended use.
Automatic driller	: Capable of holding a 300 000 lb bit load to within 1000 lb. Built in automatic safety shut-off feature.
Winches	: Two 3-ton minimum air or hydraulic. One dedicated man riding winch.
Centrifugal pumps	: Four complete with electric drive motors each rated at 600 gal/min at 50 lb/in^2 for mixing and circulating mud in the surface system.
Drilling recorder	: Five pen recorder minimum with charts to record weight-on-bit, rotary torque, pump and air pressure and flow rate. This limited instrumentation system should be augmented by a computerised drilling recording system with or without a mud logging service depending on the requirements.

Rotary hose	: 80 ft (25 m) x 4 in ID and associated standpipe for dual string drilling.
Power tools	: Air hammer and hydraulic torque wrench suitable for drilling flange bolts.
Air compressors	: Rotary screw air compressors as required to drill with reverse air assist. Capacity 3000 scf continuous operation at the elevation of the location with booster capacity, if necessary, to furnish air as required to 350 lb/in^2. Air manifold lines and fittings as required to transmit air from the compressors to the drilling rig. Air volume measuring equipment and two pen recorders.
Water storage	: Minimum 1000 bbl (160 m^3) in steel tanks.
Winterisation	: Rig covers, heating equipment, boiler and all necessary equipment to assure operation can continue in temperatures down to -40°C.
Dose pump	: Positive displacement for adding small quantities of liquid chemical or water to air system up to 10 gal/hr at 500 lb/in^2.
Rig lighting	: Lights and lighting distribution system for rig and location as required.
Welding equipment	: 240 A welding machine with cutting equipment and all necessary welding supplies to service the rig and change bit cutter saddles.
Buildings	: Doghouse, toolhouse, office, crew welfare facilities and residential accommodation units for toolpusher and other supervisory personnel.
Drill pipe	: 13-3/8 in OD upset drill pipe Range 2 or 3. The length variation in the drill pipe shall not be more than plus or minus 6 in shoulder to shoulder and the minimum ID at the tool joints shall be 11-3/4 in. The pure tension capacity of the drill pipe shall be not less than 2 000 000 lb and the pure torque capacity not less than 450 000 ft lb. Aluminium thread protectors shall be installed on each end of each drill pipe joint.
	: 4500 m 5 in Premium Class Grade E minimum 5 in 19.50 lb/ft drill pipe with all necessary handling tools.
Rotary hose	: Two 12 in ID hoses with 12 in flanges both ends 50 ft (15 m) long.

Swivel	: 1000 ton rotary swivel for direct or reverse circulation with left hand pin compatible with kelly.
Kelly	: Square kelly with left hand box on upper end compatible with swivel pin and tool joint pin on lower end compatible with drill string. The kelly shall have a minimum of 42 ft (12.80 m) and a minimum ID of 11-3/4 in. Kelly strength shall be equal to that of the drill pipe.
Saver sub	: Four saver subs to match the drill pipe, two 5 ft long, one 10 ft long and one 15 ft long. Each sub shall have aluminium thread protectors on each end.
Drill collars	: Drill collar mandrel 12 in ID x 16 in OD with 13-3/8 in tool joint box up (compatible with kelly and drill pipe) with 60 in flange down. Upper portion of mandrel stem tapered to provide elevator shoulder. Drill collar to be 50 ft long and equipped with 25 donut weights 60 in OD x 16-5/8 in ID x 18 in high with doughnut clamps. An alternative drill collar is a segmented lead filled flanged drill collar 60 in OD x 12 in ID with flanged drill pipe lifting sub having a 13-3/8 in tool joint box up.
	: Drill collar for small hole 24 in OD. Drill collar shall consist of 12 in OD x 8 in ID mandrel stem with a 24 in flange down and 13-3/8 in tool joint box up. The mandrel shall be tapered at the top to provide an elevator shoulder. Drill collar mandrel to be 50 ft long and equipped with 21 doughnut weights 24 in OD x 12-1/2 in ID x 24 in long with a hold down clamp. An alternative drill collar is a segmented lead filled drill collar with 24 in flanges for the bit body and stabilisers with a flanged lifting sub with a 13-3/8 in tool joint box up.
Stabilisers	: Slip on stabiliser with 60 in flanges top and bottom for lead filled drill collar if one is used. Stabiliser to be dressed with 12 in x 24 in long hard faced smooth rollers.
	: Two slip in stabilisers with 24 in flanges top and bottom for the lead filled drill collar if one is used. Stabilisers to be dressed with 7 in x 12 in long hard faced smooth rollers.

Bits	:	54 in bit body with integral reamer stabiliser dressed with 12 in x 24 in long rollers with tungsten carbide inserts on the lower half of the rollers. Bit body may be either flat bottom or conical in shape. Bit body shall be dressed with saddles for the cutters selected.
	:	32 in bit body (size depending on final hole and casing dimensions) with integral reamer stabiliser with 7 in x 12 in long rollers with tungsten carbide inserts in the lower half of the rollers. Bit body may be either flat bottomed or conical in shape. Bit body shall be dressed with saddles for the cutters selected.
Cutters	:	Tungsten carbide insert cutters with conical shaped profile such as Smith Type 7 cutter for each size of bit.
Fasteners	:	Nuts and bolts appropriate for the bit body, drill collar and stabilisers.
Air line	:	25 joints 3-1/2 in OD EUE 8RD left hand threaded J-55 or K-55 Range 2 tubing and pup joints 10, 5, 3 and 1 (2 No) ft in length.
Casing elevators	:	Elevators for special primary casing and deployment zone liner (2 for each size) with spreaders and slings rated at 1 000 000 lb.
Strong back	:	Base for suspending casing compatible with rig rated at 1 000 000 lb.
Drill pipe tong	:	Automatic tong compatible with drill pipe tool joints capable of making and breaking the drill pipe to the manufactures recommended torque (for example Varco).

APPENDIX 4

TENTATIVE COST ESTIMATES

SKB STUDY STAGE C : STORAGE OF NUCLEAR WASTE IN VERY DEEP BOREHOLES FILE : JBSKBCOS

COST ESTIMATE FOR 4000 m DEPTH LARGE DIAMETER BOREHOLE

CODE	DESCRIPTION	COBBS $	OTHER $	FACTOR	TOTAL $	TOTAL m SEK	NOTES
	DRILLING TO TD						
01	Preliminary study	19925	130000		149925	22.17	
02	Site investigation	28440	1000000		1028440	6.75	Cost provision allocated to one borehole
03	Detailed design	56500	56500	2.00	113000	0.74	
04	Site development		500000		500000	3.28	Provision. Note SII costs too low for Sweden
05	Rig mobilisation	78000	78000	2.00	156000	1.02	Cobbs + extra for Sweden
06	Rig demobilisation		100000		100000	0.66	Overhaul or demobilisation within Sweden
07	Rig operation	3609195	1804598	1.50	5413793	35.53	Assumes $12445 per day for 435 days
09	Rig modification		200000		200000	1.31	Ex SII cost estimate + 50% for overhaul
10	Special tools	5467420	1366855	1.25	6834275	44.85	Cobbs + 25% for shipping etc
11	Power/fuel		610000		610000	4.00	Rig + winterising + compressors at $1400 per day
12	Air compressors	300000	75000	1.25	375000	2.46	
13	Bits and cutters	1077554	1077554	2.00	2155108	14.14	There will be high bit wear
14	Tool rental		250000		250000	1.64	Provision. Special tools purchased
15	Drilling fluids	45039	22520	1.50	67559	0.44	Prudent uplift to Cobbs estimate
16	Casing	4443000	8886000	3.00	13329000	87.47	Very rough estimate for special copper alloy
17	Casing welding	162990	325980	3.00	488970	3.21	Uplift to cover copper alloy welding
18	Cementing	2467920	-2221128	-0.90	246792	1.62	Provision for minor cementing
19	Survey services	50000	12500	1.25	62500	0.41	
20	Equipment purchase		100000		100000	0.66	Provision. Special tools shown separately
21	Wellhead equipment		25000		25000	0.16	Provision
22	Coring ahead		3384550		3384550	22.21	Rig time for 100 days + equipment and bits
23	Logging ahead		300000		300000	1.97	Provision
24	Testing during drilling		100000		100000	0.66	Provision
25	Miscellaneous services		500000		500000	3.28	
26	Water supply		100000		100000	0.66	
27	Transport		200000		200000	1.31	
28	R and D		250000		250000	1.64	Provision allocation to each borehole
29	Engineering		100000		100000	0.66	Provision allocation to each borehole
30	Site supervision	382000	286500	1.75	668500	4.39	Includes additional 100 days for investigation
31	Management		500000		500000	3.28	Provision
32	Site operation		480000		480000	3.15	Based on similar daily cost to SII estimate
33	Admin/legal		250000		250000	1.64	
34	Insurance		100000		100000	0.66	
35	Contingency	1234059	1234059	2.00	2468118	16.20	Cobbs allowed 10% of total US equivalent
	TOTAL DRILLING	19422042	22184487		41606529	273.04	

WASTE DEPLOYMENT AND SEALING

36	Rig operation	4542608	4542608	29.81	Deployment and sealing assumed to take 365 days
37	Power/fuel	438000	438000	2.87	Fuel + winterising assumed as $1200 per day
38	Special tools	100000	100000	0.66	Provision
39	Special services	100000	100000	0.66	Provision
40	Completion fluid	150000	150000	0.98	
41	Protective liner	6664500	6664500	43.74	
42	Cannister deployment	100000	100000	0.66	
43	Deployment mud	1000000	1000000	6.56	
44	Deployment zone seals	1200000	1200000	7.88	
45	Upper seals	1500000	1500000	9.84	
46	Surface seal	500000	500000	3.28	
47	Miscellaneous services	200000	200000	1.31	
48	Water supply	100000	100000	0.66	
49	Transport	100000	100000	0.66	
50	R and D	200000	200000	1.31	
51	Engineering	100000	100000	0.66	
52	Site supervision	1000000	1000000	6.56	
53	Site operation	365000	365000	2.40	
54	Management	365000	365000	2.40	
55	Admin/legal	100000	100000	0.66	
56	Insurance	100000	100000	0.66	
57	Contingency	2000000	2000000	13.13	
	TOTAL WASTE DEPLOYMENT	20925108	20925108	137.32	
	OVERALL TOTAL PER BOREHOLE		62531636	410.36	

NOTES

Cobbs cost estimate based on US domestic prices. Adjustment factor shown
Additional cost of mobilising rig from Europe $250000
Additional cost of mobilising rig from North America $500000
Assumes use of DDP Rig 20 suitably modified

SUMMARY

Large diameter volume	cub m	566	Assuming deployment over 2000 m of 600 mm diameter hole
Drilling costs	$	41606529	
Deployment costs	$	20925107	
Total costs	$	62531636	
Unit drilling costs	$/cub m	73567	
Unit deployment costs	$/cub m	36999	
Unit overall costs	$/cub m	110566	

SUMMARY OF TIME SCHEDULE AND COSTS PER DAY

Drilling time	days	535
Deployment time	days	365
Total time	days	900
Per day drilling cost	$/day	77769
Per day deployment cost	$/day	57329
Per day overall cost	$/day	69480

SKB STUDY STAGE C : STORAGE OF NUCLEAR WASTE IN VERY DEEP BOREHOLES FILE : JBSKBX

COST ESTIMATE FOR 5500 m SMALL DIAMETER BOREHOLE

CODE	DESCRIPTION	SII $	OTHER $	FACTOR	$	TOTAL m SEK	NOTES
	DRILLING TO TD						
01	Preliminary study	0	150000		150000	0.98	
02	Site investigation	0	1500000		1500000	9.84	
03	Detailed design	0	100000		100000	0.66	
04	Site development	265000	265000	2.00	530000	3.48	Provision. SII estimate too low for Sweden
05	Rig mobilisation	460000	-310000	-0.67	150000	0.98	Assuming DDP Rig 20 used
06	Rig demobilisation	0	100000		100000	0.66	Overhaul or demobilisation within Sweden
07	Rig operation	1556000	2411800	2.55	3967800	26.04	Assumes $12445 per day for 319 days
09	Rig modification	200000	200000	2.00	400000	2.63	Ex SII cost estimate + 100% to cover overhaul
10	Special tools	0	200000		200000	1.31	
11	Power/fuel	171000	275310	2.61	446310	2.93	Rig + winterising + compressors at $1400 per day
12	Air compressors	0	100000		200000	1.31	Provision for top hole
13	Bits and cutters	688000	516000	1.75	1204000	7.90	Based on budget quotes
14	Tool rental	700000	350000	1.50	1050000	6.89	
15	Drilling fluids	246000	61500	1.25	307500	2.02	
16	Casing	140000	280000	3.00	420000	2.76	Assumes provision for special materials etc
17	Casing welding	0	50000	1.50	50000	0.33	Provsion for surface casing
18	Cementing	170000	42500	1.25	212500	1.39	
19	Survey services	295000	73750	1.25	368750	2.42	
20	Equipment purchase	160000	40000	1.25	200000	1.31	
21	Wellhead equipment	25000			25000	0.16	
22	Coring ahead		3384550		3384550	22.21	
23	Logging	300000			300000	1.97	Provision
24	Testing during drilling		100000		100000	0.66	Provision
25	Miscellaneous services	285000	213750	1.75	498750	3.27	
26	Water supply	50000	50000	2.00	100000	0.66	
27	Transport	125000	125000	2.00	250000	1.64	
28	R and D	250000			250000	1.64	
29	Engineering	100000			100000	0.66	
30	Site supervision	216000	324000	2.50	540000	3.54	
31	Management	0	500000		500000	3.28	
32	Site operation	200000	180000	1.90	380000	2.49	
33	Admin/legal	90000	157500	2.75	247500	1.62	
34	Insurance	70000	35000	1.50	105000	0.69	
35	Contingency	50000	1850000	38.00	1900000	12.47	Provsion of 10%
	TOTAL DRILLING	6812000	13325660		20137660	132.15	

WASTE DEPLOYMENT AND SEALING

36	Rig operation	4542608	4542608	29.81	Assumes 365 days to deploy waste
37	Power/fuel	438000	438000	2.87	Assumes $1200 per day
38	Special tools	100000	100000	0.66	
39	Special services	100000	100000	0.66	
40	Completion fluid	60000	60000	0.39	
41	Protective liner	315000	315000	2.07	
42	Cannister deployment	100000	100000	0.66	
43	Deployment mud	250000	250000	1.64	
44	Deployment zone seals	500000	500000	3.28	
45	Upper seals	600000	600000	3.94	
46	Surface seal	250000	250000	1.64	
47	Miscellaneous services	200000	200000	1.31	
48	Water supply	100000	100000	0.66	
49	Transport	100000	100000	0.66	
50	R and D	200000	200000	1.31	
51	Engineering	100000	100000	0.66	
52	Site supervision	1000000	1000000	6.56	
53	Site operation	365000	365000	2.40	
54	Management	365000	365000	2.40	
55	Admin/legal	100000	100000	0.66	
56	Insurance	100000	100000	0.66	
57	Contingency	2000000	2000000	13.13	
	TOTAL WASTE DEPLOYMENT	11885608	11885608	78.00	
	OVERALL TOTAL PER BOREHOLE		32023268	210.15	

NOTES

SII cost estimate based on 180 day 6000 m borehole. Costs adjusted for larger diameter 5500 m borehole
Additional cost of mobilising rig from Europe $250000
Additional cost of mobilising rig from North America $500000
Assumes use of DDP Rig 20 or rig on similar basis

SUMMARY

Small diameter volume	cub m	123	Assuming deployment over 2500 m of 250 mm diameter hole
Drilling costs		$ 20137660	
Deployment costs		$ 11885608	
Total costs		$ 32023268	
Unit drilling costs	$/cub m	164075	
Unit deployment costs	$/cub m	96840	
Unit overall costs	$/cub m	260915	

SUMMARY OF TIME SCHEDULE AND COSTS PER DAY

Drilling time	days	419
Deployment time	days	365
Total time	days	784
Per day drilling cost	$/day	48061
Per day deployment cost	$/day	32563
Per day overall cost	$/day	40846

APPENDIX 5

CANISTER DESIGN-ESTIMATES

CANISTER DESIGN - ESTIMATES

Data

Maximum deployment depth	4000 m
Density of canister material (titanium)	4540 kg/m^3
Density of drilling fluid during deployment and sealing	1150 kg/m^3
Maximum load during deployment	46 MPa
Maximum load during storage	45 MPa

Nomenclature

l	=	Length of canister
ν	=	Poisson's ratio
ρ_b	=	Density of drilling mud
z	=	Deployment depth
p	=	External pressure
p_s	=	Swelling pressure of bentonite
E	=	Young's modulus
I	=	Moment of inertia
a	=	Mean diameter of canister
a_o	=	Outer diameter of canister
a_i	=	Inner diameter of canister
σ_d	=	Design strength
σ_{cr}	=	Critical stress
F_{max}	=	Maximum load at design strength
F_k	=	Euler crippling load
σ_k	=	Stress at F_k
σ_{cyl}	=	Stress in canister shell
h_{min}	=	Minimum thickness of canister wall
M	=	Moment
t	=	Thickness of end-lid
t_{min}	=	Minimum required thickness of end-lid

The following estimates have been made:

External pressure at deployment depth.

$$p = b \cdot g \cdot z + p_s \tag{1.1}$$

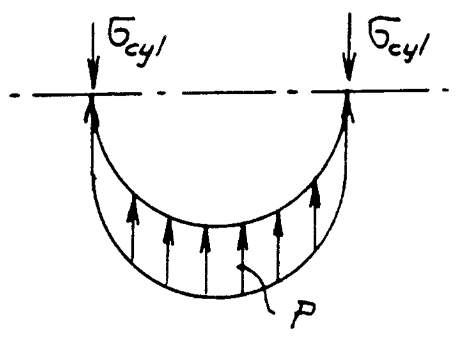
Buckling of a cylindrical shell under the action of uniform axial pressure has been calculated according to Timoshenko & Gere (1961).

$$\sigma_{cr} = \frac{E \cdot h}{a_o \cdot \sqrt{3(1-\nu^2)}} \tag{1.2}$$

If $\sigma_{cr} < \sigma_d$ buckling, due to external pressure, will occur.

Axial buckling (crippling) will not occur due to a deployment period of one year. With this deployment rate the mobilized wall friction will roughly balance the rate of load increase.

The stress in the shell has been calculated under the assumption of an ideal cylinder. The stress is a lower limit of the canister material design strength.



$$\sigma_{cyl} = \frac{p \cdot a_o}{h}$$

$$\Rightarrow \quad h_{min} = \frac{p \cdot a_o}{\sigma_d} \tag{1.3}$$

The required thickness of the canister wall has also been calculated according to the Swedish Tryckkärlsnormer (code for pressure vessels) (1978).

The minimum thickness of the canister wall, h_{min}, has to meet two conditions:

$$h^{I}_{min} \geqslant \frac{2 \cdot a_o}{100} \cdot \sqrt[3]{p \cdot k \cdot 10} \tag{1.4}$$

where k = 1.05 for steel at 150°C and 1.83 for copper. No value is given for titanium.

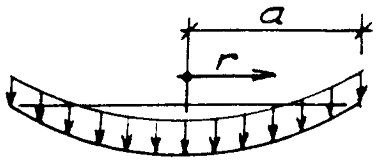
$$h^{II}_{min} \geqslant \frac{2 \cdot a_o \cdot p \cdot 10}{20 \cdot \frac{\sigma_d}{3}} \tag{1.5}$$

safety-factor

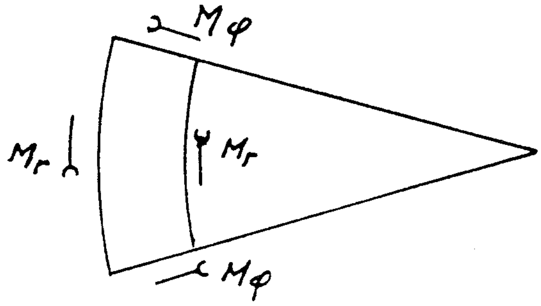
In this case, formula (1.5) is always the critical one.

If the safety-factor 3 in formula (1.5) is reduced to 1, formula (1.5) and (1.3) are equal.

Finally, the minimum thickness of the end-lid has been calculated, assuming that the design strength may be obtained.



$$a = \frac{a_o + a_i}{2}$$



$$M_r = -\frac{p\cdot(3+\nu)}{16}(a^2 - r^2)$$

$$M_\varphi = -\frac{p}{16}\cdot\left(a^2(3+\nu) - r^2(1+3\nu)\right)$$

$$\sigma = \frac{6M}{t^2}$$

$$r = 0 \Rightarrow M_r = M_\varphi = -\frac{p(3+\nu)}{16}\cdot a^2$$

$$\sigma_r = \sigma_\varphi = \sigma$$

$$r = a \rightarrow M_r = 0$$

$$M_\varphi = -\frac{2p}{16}\cdot a^2(1-\nu);$$ (ALWAYS LESS THAN WHEN $r=0$)

$$t_{min} = \sqrt{\frac{6M}{\sigma_d}}$$

DEEP HOLE WASTE DISPOSAL

Titanium - canister, an example

Data

Length of canister	4400 mm
Outer diameter of canister	500 mm
Inner diameter of canister	390 mm
Design strength	240 MPa
Young's modulus	100000 Mpa
Poisson's ratio	0.22
Thickness of end-lid	225 mm

Calculations

Pressure at deployment depth incl. swell-pressure	46 MPa
Buckling of a cylindrical shell under the action of uniform axial pressure. According to Timoshenko & Gere "Theory of elastic stability", chap. 11.3	7102 MPa
No buckling	
Stress in shell (ideal cylinder)	209 MPa
Minimum canister wall thickness if max = d	48 mm
Swedish "Tryckkärlsnormer", Section 8.1.2.4	
Minimum canister wall thickness	287 mm
Minimum canister wall thickness (no safety factor)	96 mm
Maximum stress in a 225 mm thick end-lid	217 MPa
Minimum required thickness at design strength	214 mm

It is impractical to have a 214 mm thick end-lid. In this report it is assumed that the needed metal spacers between the element will support the lids and thus reduce the thickness to 100 mm.

ESTIMATE OF PRESENT VALUE

Estimate of the present value for the basic option with a large borehole to 4 km depth discounted to year 1990

The estimate of the present value for the basic option with a large borehole to 4 km depth without rod consolidations, discounted to year 1990 below is based on an interest rate of 2.5%

Year	Cost per year MSEK	Nuvärdes-faktor 2.5% ränta	Discounted cost year 1990 MSEK
2018	668.6	0.4887	326.74
2019	668.6	0.4767	318.72
2020	668.6	0.4651	310.97
2021	668.6	0.4538	303.41
2022	668.6	0.4427	295.99
2023	668.6	0.4319	288.77
2024	668.6	0.4214	281.75
2025	668.6	0.4111	274.86
2026	668.6	0.4011	268.18
2027	668.6	0.3913	261.62
2028	668.6	0.3817	255.20
2029	668.6	0.3724	248.99
2030	668.6	0.3633	242.90
2031	668.6	0.3545	237.02
2032	668.6	0.3458	231.20
2033	668.6	0.3374	225.59
2034	668.6	0.3292	220.10
2035	668.6	0.3211	214.69
2036	668.6	0.3133	209.47
2037	668.6	0.3057	204.39
2038	668.6	0.2982	199.38
2039	668.6	0.2909	194.50
2040	668.6	0.2838	189.75
2041	668.6	0.2769	185.14
2042	668.6	0.2702	180.66
2043	668.6	0.2636	176.24
2044	668.6	0.2572	171.96
2045	668.6	0.2509	167.75
2046	668.6	0.2448	163.67
Present value year 1990			6849.61

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Near-distance seismological monitoring of the Lansjärv neotectonic fault region Part II: 1988
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Conny Holmqvist, Hans-Edy Mårtensson
Seismological Department, Uppsala University, Uppsala
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Ebbe Eriksson, Stefan Sehlstedt
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Radionuclide release from the near-field in a WP-Cave repository
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TR 89-05
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Transport of escaping radionuclides from the WP-Cave repository to the biosphere
Luis Moreno, Sue Arve, Ivars Neretnieks
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Individual radiation doses from nuclides contained in a WP-Cave repository for spent fuel
Sture Nordlinder, Ulla Bergström
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TR 89-07
SKB WP-Cave Project
Some Notes on Technical Issues
Part 1: Temperature distribution in WP-Cave: when shafts are filled with sand/water mixtures
Stefan Björklund, Lennart Josefson
Division of Solid Mechanics, Chalmers University of Technology, Gothenburg, Sweden
Part 2: Gas and water transport from WP-Cave repository Luis Moreno, Ivars Neretnieks
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