SKB

TECHNICAL REPORT

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Deep drilling KLX 02

Drilling and documentation of a 1700 m deep borehole at Laxemar, Sweden

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August 1994

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DRILLING AND DOCUMENTATION OF A 1700 M DEEP BOREHOLE AT LAXEMAR, SWEDEN

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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.

Information on SKB technical reports from1977-1978 (TR 121), 1979 (TR 79-28), 1980 (TR 80-26), 1981 (TR 81-17), 1982 (TR 82-28), 1983 (TR 83-77), 1984 (TR 85-01), 1985 (TR 85-20), 1986 (TR 86-31), 1987 (TR 87-33), 1988 (TR 88-32), 1989 (TR 89-40), 1990 (TR 90-46), 1991 (TR 91-64) and 1992 (TR 92-46) is available through SKB.

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Keywords: Core drilling, wireline system, DTH predrilling, preparations, organization, set up, equipment, air-lift support, fluid control, drilling parameters, air-lift pumptests, hole completion, core inspection, drilling statistics, technical evaluation

ABSTRACT

In this report the preparation and execution of the deep core drilling KLX 02 is described.

The hole was drilled with the wireline method, NQ dimension (Ø 76 mm), to a final depth of 1700.5 m.

Prior to core drilling a Ø 215 mm pilothole was predrilled to 200 m with controlled hammerdrilling (DTH). In this hole casing and air-lift equipment was installed with the aim to support the circulation of drilling fluid.

During core drilling there was a measurement of major drilling parameters and drilling fluid in and out of hole. As a fluid tracer uranine was used.

Each 300 m of core drilling air-lift pumptests were performed. After completion a flow-meter log was run to finalize the project phase.

It can be concluded that both the predrilling and core drilling methods used proved to be successful. No severe technical problems occurred. However, potential risks have been pointed at in the report.

The air-lift system functioned only partly and has to be modified for further use. Also the technique for monitoring of drilling parameters needs improvement as does the method for air-lift pumptests with packer.

The organisation model for planning and realization functioned satisfactory and can be recommended for similar future projects.

SAMMANDRAG

I denna rapport beskrivs förberedelser och genomförande av den djupa kärnborrningen KLX 02.

Borrningen utfördes med wire-line-metoden, dimension NQ (Ø 76 mm) till ett djup av 1700.5 m.

Innan kärnborrningen gjordes en förborrning Ø 215 mm till 200 m djup med styrd hammarborrning (DTH). I detta hål installerades foderrör och mammutpumputrustning med avsikten att underlätta cirkulationen av spolvatten.

Under kärnborrningen mättes de viktigaste borrningsparametrarna samt spolvattenflödena in i och ut ur hålet. Som spårämne användes uranin.

Var 300:e meter utfördes pumptester med mammutpumpning. En mätning med flödeslog avslutade projektetappen.

Borrningstekniskt visade sig både förborrnings- och kärnborrningsmetoden vara framgångsrik. Inga svårartade problem uppstod. Vissa potentiella risker har dock påtalats i rapporten.

Den kontinuerliga mammutpumpningen fungerade endast delvis och måste modifieras inför framtida användning. Även tekniken för insamling av borrningsparametrar behöver förbättras, liksom pumptesttekniken med luft och manschett.

Organisationsmodellen som använts för planering och genomförande har fungerat tillfredsställande och kan rekommenderas för liknande framtida projekt.

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SUMMARY

The object of the deep core drilling KLX02 was to demonstrate a feasible drilling technique for investigations at depth down to 1500 m.

Due to earlier failures with the conventional method the Wire Line System was chosen and a rig with drillers experienced in WL coring was contracted. The hole was drilled from 200.8 m to a total depth of 1700.5 m with standard NQ dimension (Ø 75.7 mm).

The time needed was roughly 900 hours for core drilling and tests during drilling. This was approximately 200 hours less than scheduled. The average rate was just under 2 m/h, core-trips and time for shutdowns included. The average bit life was roughly 70 m and 22 bits were used. The number of core trips was 302, giving an average core length of 5 m. The core recovery was close to 100 % (0.6 m loss).

To minimize contamination by drilling fluid entering the rock the hole was air-lift pumped. This was performed by using a 200 m deep predrilled hole, Ø 215 mm, in which the air-lift equipment and core drilling casing were installed.

Predrilling was successfully carried out in two steps using the DTH-method modified for straight hole drilling.

During core drilling inclination and drilling parameters were recorded. Furthermore, the amount of flushing water entering and leaving the hole were recorded.

At certain levels short pumping tests were performed using a packer and air-lift technique.

Finally the hole was completed to allow further investigations.

From a technical drilling point of view the WL system proved to be very successful. However, at certain levels high torque became a potential risk due to unsatisfactory hole cleaning and the fact that no lubricating fluid additive was allowed to be used.

For future similar projects it is recommended that the air-lift system should be modified to avoid sedimentation of cuttings in the pre-drilled hole.

The collection of drilling data parameters did not function perfectly. For future applications technical and operative improvements are needed.

The system for recording the flow rate out of hole partly failed due to sedimentation of cuttings and system defects. However, the system can be put into working order by simple modifications.

Four out of five planned air-lift pumping tests were performed in an acceptable way. However, the system used with an unprotected packer to be sunk down the hole proved to be risky. In future projects the standard WL-packer system is recommended.

Finally, the procedure concerning planning and the operative organization used through the project proved to function satisfactorily. The same approach is recommended in future similar projects. However, due to the high speed of data flow during drilling, the field personnel should be reinforced to a level that ensures a continuous high standard of data collection.

SELECTED DRILLING DATA	
GENERAL	TECHNICAL
Name of hole: KLX 02	*Predrilling rig:* Tamrock D10K
Location: Laxemar, Oskarshamn Municipality, Sweden	*Predrilling dimension:* DTH ø215 mm
Position: X 66768.59 Y 49224.23 Z +18.31m	*Predrilling depth:* 201 m
Inclination/Deviation: 85°N20°E/±1 °	*Core drilling rig:* Longyear Hydro 55
Operator: Swedish Nuclear Fuel and Waste Management, Stockholm, Sweden	*Core drilling dimension:* WL NQ (ø75.7 mm)
Main drilling contractor: Suomen Malmi OY, Espoo, Finland	*Cored interval:* 201-1700.5m
Subcontractor predrilling: Sven Andersson AB, Uppsala, Sweden	*Core recoverey:* 99.9 %
Predrilling operation date: 6-12 October 1992	*Core drilling rate:* 1.84 m/h
Subcontractor core drilling: Longyear Inc., Canada	*Average core length:* 5.0 m
Core drilling spudding date: 15 October 1992	*Number of runs:* 302 times
Completion date: 3 December 1992	*Diamond bits used:* 22 pcs
	Average bit life: 68 m

1. INTRODUCTION

The deep drilling project at Laxemar, KLX 02, is the second attempt to investigate deep rock properties, hydrochemical conditions and hydraulics at the border zone to the *Äspö Hard Rock Laboratory (HRL)* (see overview map, Fig. 1.1).

The first trial was carried out in 1990, when it was planned to deepen KLX 01 from 703 m to 1500 m. However, this attempt was discontinued at the depth of 1078 m due to severe technical problems and, finally, an overdrawn budget (1).

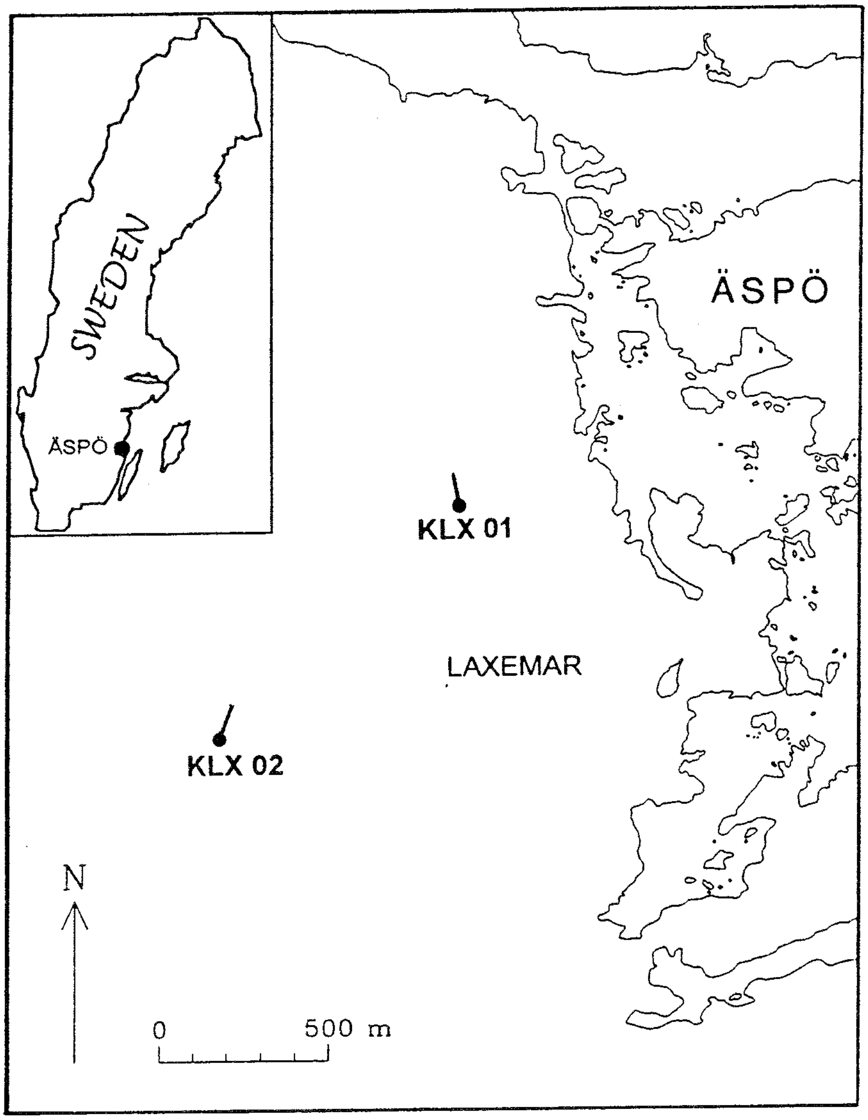

Figure 1.1 Location of Äspö and the deep boreholes KLX 01 and KLX 02 at Laxemar

Based on the experience from KLX 01, one of the main objectives was to demonstrate a drilling technique feasible to safely reach the target depth of 1500 m and still permit different forms of field tests both during and after drilling.

In order to fulfil this goal the project was planned and carried out according to a comprehensive QA (Quality Assurance) programme. This consists of a manual for planning purposes (2) and detailed activity programmes and manuals for the performance (3).

The grand project was split into two stages, drilling and scientific investigations. This report describes only the activities and results carried out within stage 1, mainly drilling and documentation during drilling.

2. ACTIVITY PLAN

Prior to drilling and in the stage of negotiation with the drilling contractor an activity plan was elaborated. The final revised version is shown in Figure 2.1.

The plan was used as guidance for site geologists and the drilling contractor and covers all topics involved in site preparation, drilling and measurement while drilling (MWD). It also points out items in project management.

The overall time schedule was preset, based on expectations concerning time needed for different issues and the general goal to have all drilling activities finished prior to Christmas 1992.

In order to achieve the narrow time goal, the schedule was made as short as possible between site preparation, predrilling and core drilling. Normally the time gap between these steps should be considerably longer to have a safe operation. However, in this case the advantages of a tight schedule were considered to outweigh technical and financial risks.

The conceptual time needed for the main items, core drilling and MWD, was estimated at 1600 hours based on the following assumptions

- rigging up	75	hours
- core drilling	1000	hours
- deviation control	25	hours
- pumping tests etc.	250	hours
- shutdowns for repairs etc.	250	hours
	1600	hours

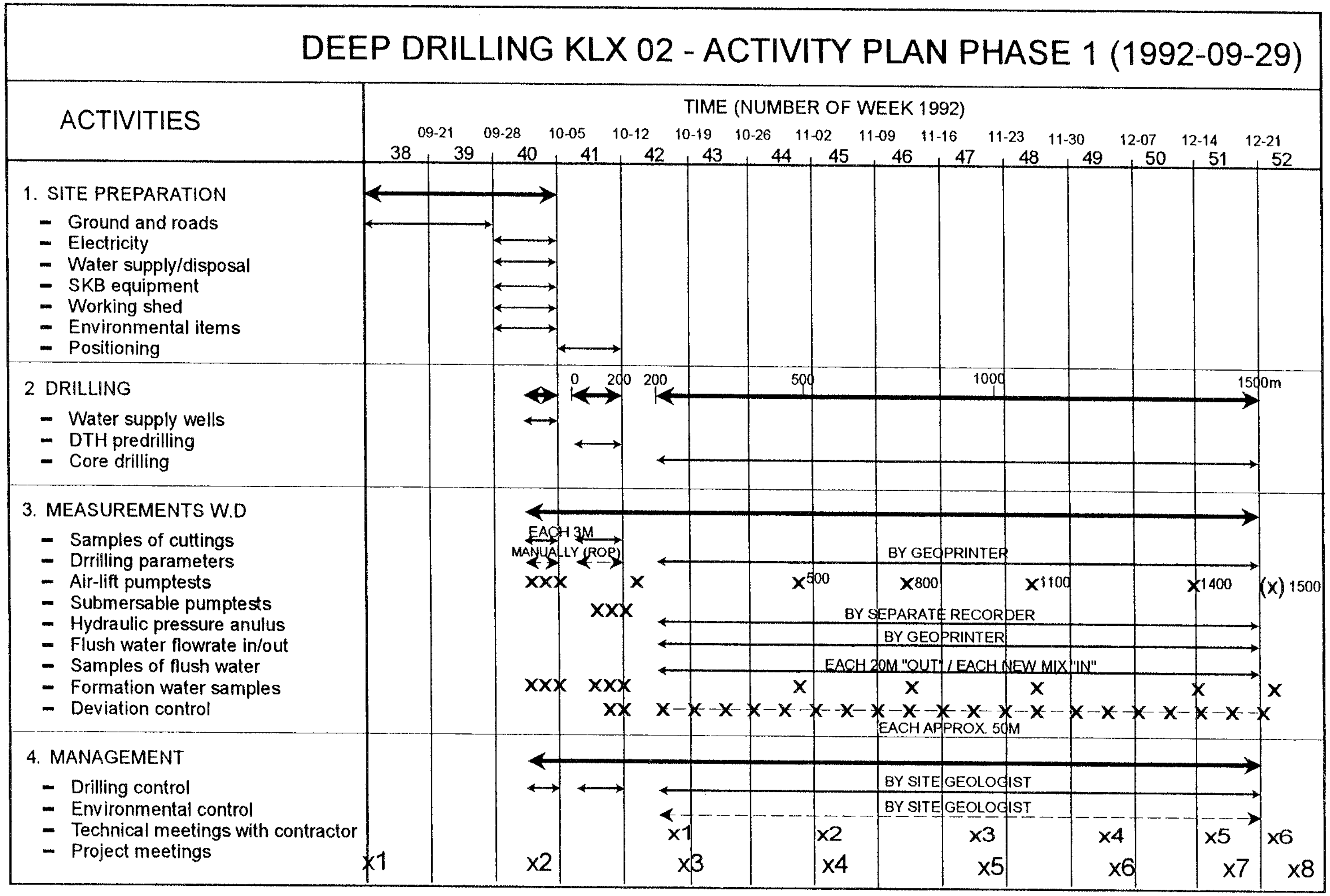

Figure 2.1 Activity plan used in the project

3. ORGANIZATION

The project organization was established by following international practice within the drilling branch.

As operator, SKB formed an operative project group with the following composition and brief responsibilities.

- Project manager (purchase and quality assurance)
- Drilling supervisor (drilling progress and evaluation)
- MWD supervisor (measurement progress and evaluation)
- Site geologist (measurement performance and evaluation)
- Technical assistant (performance data collection and quality assurance)

During the preparatory stage there was also a site preparation supervisor, who was also responsible for working out contracts with landowners and local authorities.

The main contractor, SMOY, formed the following operational organization.

- Project manager (general responsibility)
- Site foreman (drilling supervision)
- Assistant site foremen (drilling supervision)
- Toolpusher (first main driller)
- Assisting tool pusher (second main driller)
- Helpers (drillers' assistants)

During operation a few representatives of each group had scheduled meetings to continuously deal with technical and administrative questions. These meetings were held each fortnight prior to regular SKB project group meetings.

The operative organization is illustrated in Figure 3.1.

The day-to-day communication and documentation was kept by the contractor handing over daily reports to SKB. These are to be found in "Technical documents" (5).

Minor technical problems were solved directly at the drilling site by the SKB authority entrusted to the site geologist.

Major concerns or deviations from the programme were first discussed in the SKB project group before measures were taken.

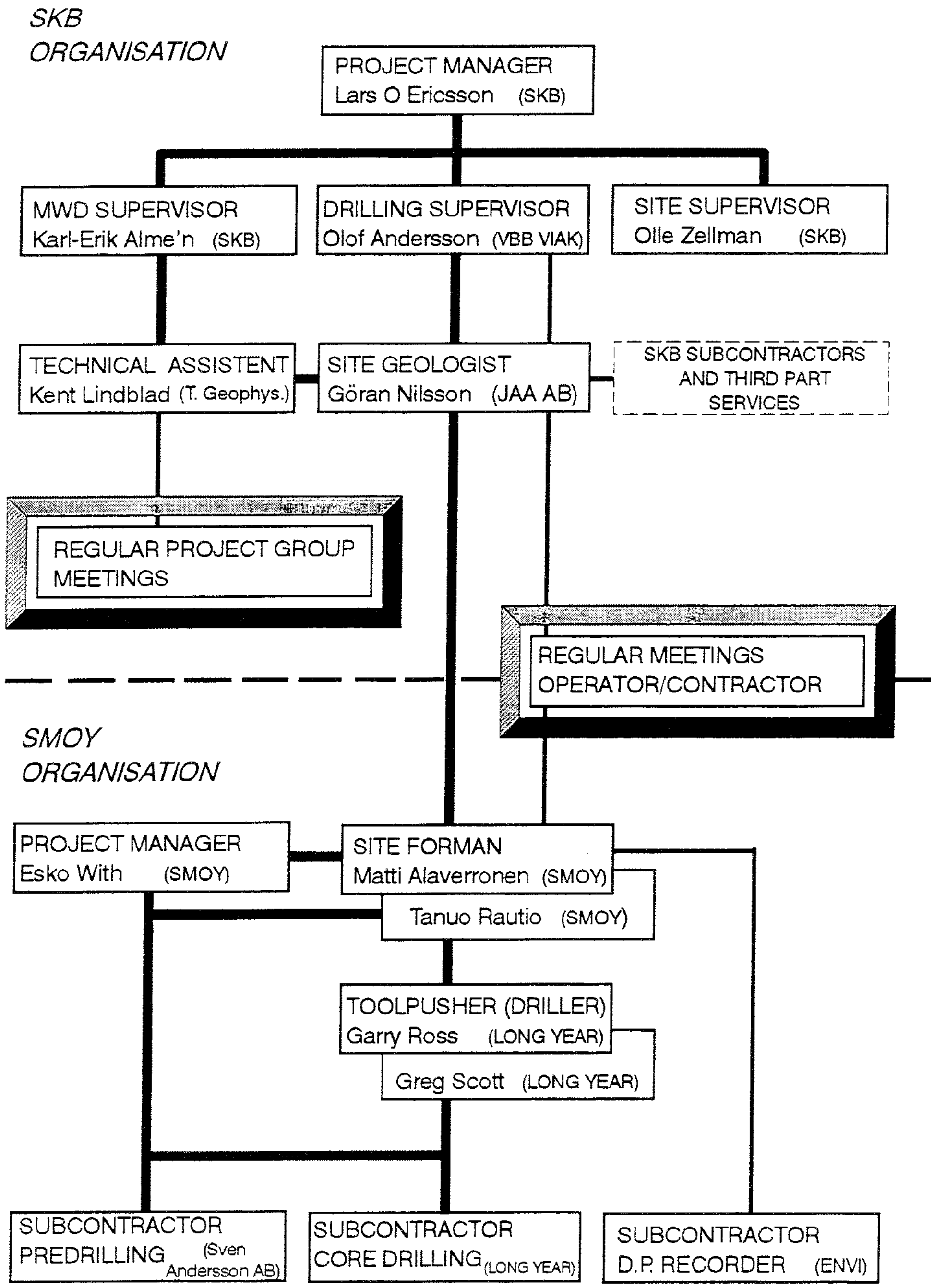

Figure 3.1 Operative project organisation

Notes from SKB project group meetings and meetings with the contractor have been filed in 'Administrative documents' (4), together with daily shift reports.

4. PREPARATIONS

4.1 Drilling site

The drilling site was prepared according to programme (3).

A surface area of approximately 500 m^2 was cleared of trees and bushes. The topsoil was then excavated and finally the surface was levelled with coarse gravel and on top of that a sand layer was placed.

An access road was built to connect the drilling site with the existing road. The latter was finally widened for storage of stand-by equipment, parking, etc.

4.2 Power supply

Electricity was mainly used for lighting and for driving a compressor.

A temporary electric cable was drawn from a nearby transformer initially 230 V, 60 A. Due to overloading the electric power was later increased to 3 x 50 A.

4.3 Water supply wells

Three water supply wells were drilled according to the programme (3). They were located at VLF-anomalies at a distance of approximately 200-250 m from the drilling site (see map Figure 4.1).

The wells were drilled by Sven Andersson AB using a Tamrock D10 rig.

Through the overburden and a few metres into the rock, ODEX W140 equipment was used. The casing Ø 168 mm was then sealed with cement.

The open hole drilling was done with a down-the-hole hammer with the dimension Ø 140 mm.

During drilling, samples of cuttings were taken and the rate of penetration was measured in minutes per drill pipe (min/3 m).

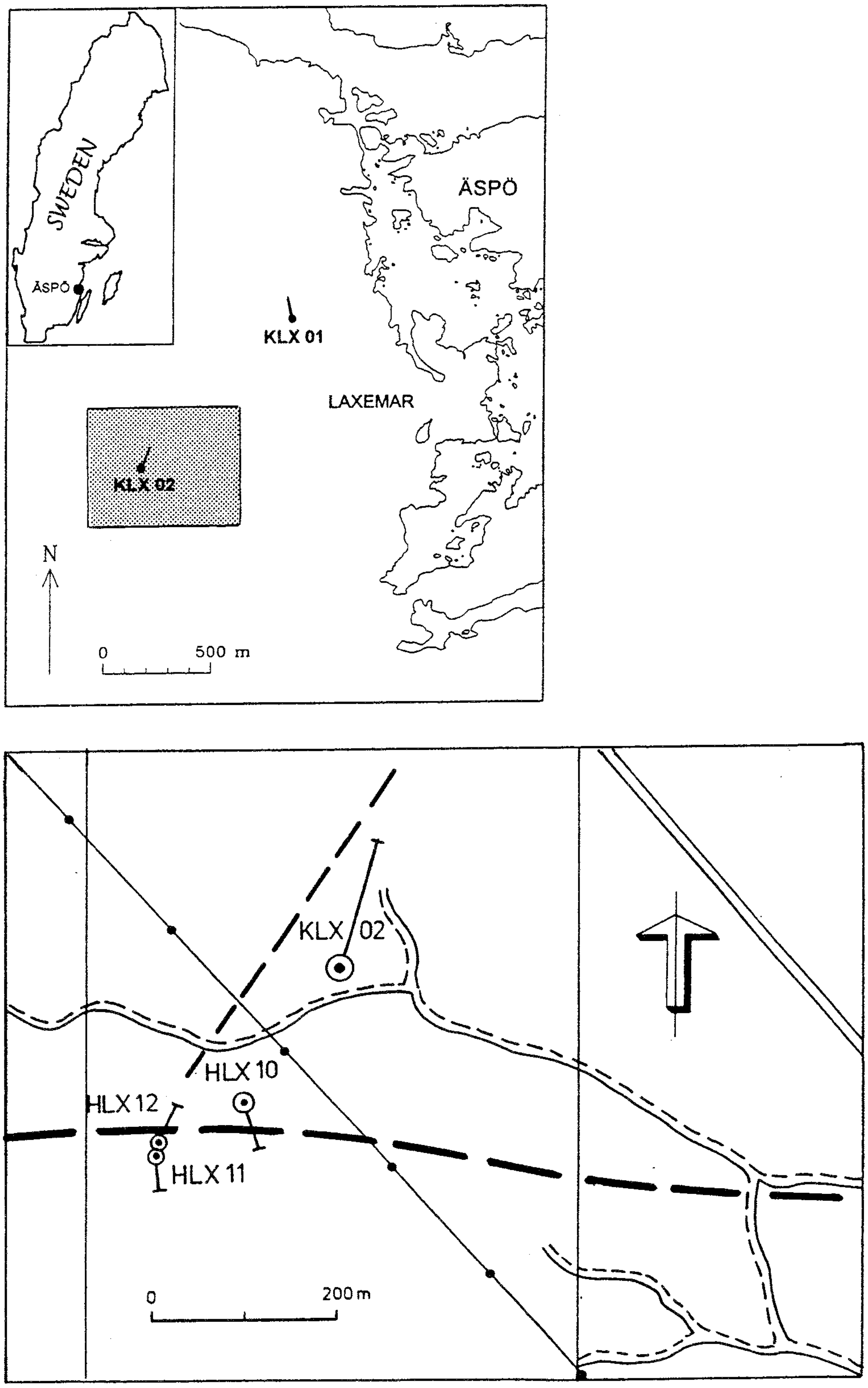

Figure 4.1 Location of the water supply wells HLX 10-12. Dotted lines are VLF-anomalies

Well capacity was estimated in steps by air-lift pumping through drill pipes. At the final capacity test water samples were taken and analysed in the field.

The main well data are shown in Table 4.1. More detailed data have been field in 'Technical Documents' (5) and the drillers logs are shown in a raw data report (8).

Based on the results HLX 10 was chosen to be the main supply well and HLX 12 to be a reserve well.

Table 4.1 Main data, water supply wells

Well number	Final depth (m)	Inclination	Estimated capacity (m^3/h)	pH, hardness (°dH), iron (mg/l) and chloride (mg/l)
HLX 10	85	70° S10°E	10-15	8.0/3/2.9/23
HLX 11	70	85° S5°W	0.1-0.2	No analyses made
HLX 12	31	70° N5°E	3-5	8.0/6/0.4/23

4.4 Flushing water system

The system for supply and disposal of flushing water was constructed according to programme (3).

A submersible 4" pump, designed for 6 m^3/h was installed in HLX 10 and a back-up pump in HLX 12.

From HLX 10 a 2" Teclan tube was laid to the drilling site entering at a set of 3x5 m^3 containers. A flow meter and a stop valve were installed on the tube.

The system for flushing water disposal from KLX 02 was constructed to allow continuous measurement of the return water.

A standpipe above the hole, but beneath the rig floor, was installed and on top of that a flat rubber packer was placed for sealing against the drill rods.

Initially a tube was drawn from the standpipe to a set of two 5 m^3 containers. On the tube a flow meter was installed for continuous recording of flow rates and the accumulated flow.

The two containers were linked together by an overflow hose. The first container served as a trapper for cuttings and the second one as reservoir for disposal.

In the second container a disposal pump set was connected to a Ø 2" tube. The tube was placed along a road slightly downhill of the drilling site and ended in a smaller excavated ditch at a distance of approximately 200 m.

From here, the return water entered a natural ditch with its outlet in the Baltic Sea some 2 km away from the drilling site.

Due to clogging problems with the flow meter and leakages at the standpipe, the disposal system was modified after some weeks of drilling.

Leakage water was collected using a rubber canvas around the standpipe. The water then flowed to an excavated sand trap sealed off with a plastic cover and from there to the disposal containers.

The latter flushing water system is illustrated diagrammatically in Figure 4.2.

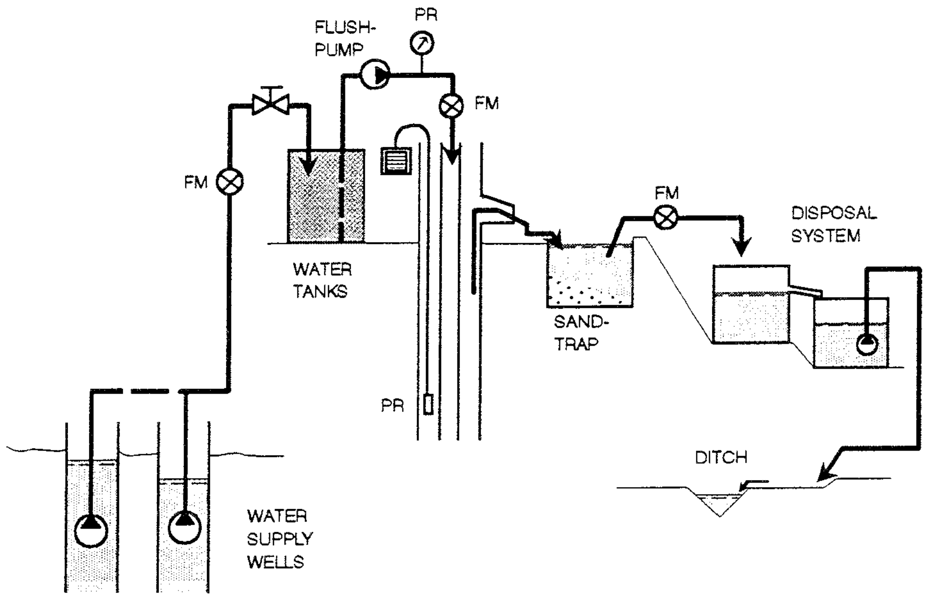

Figure 4.2 Overview of the flushing water system

5. PREDRILLING PERFORMANCE

5.1 Equipment and method

The aim was to drill approximately 200 m to provide space for the installation of air-lift pumping equipment and a double supporting casing for the core drilling. The air-lift pumping system, principally shown in figure 5.1, was introduced during the pre-investigation of the Äspö Hard Rock Laboratory project and proved to be very successful (6).

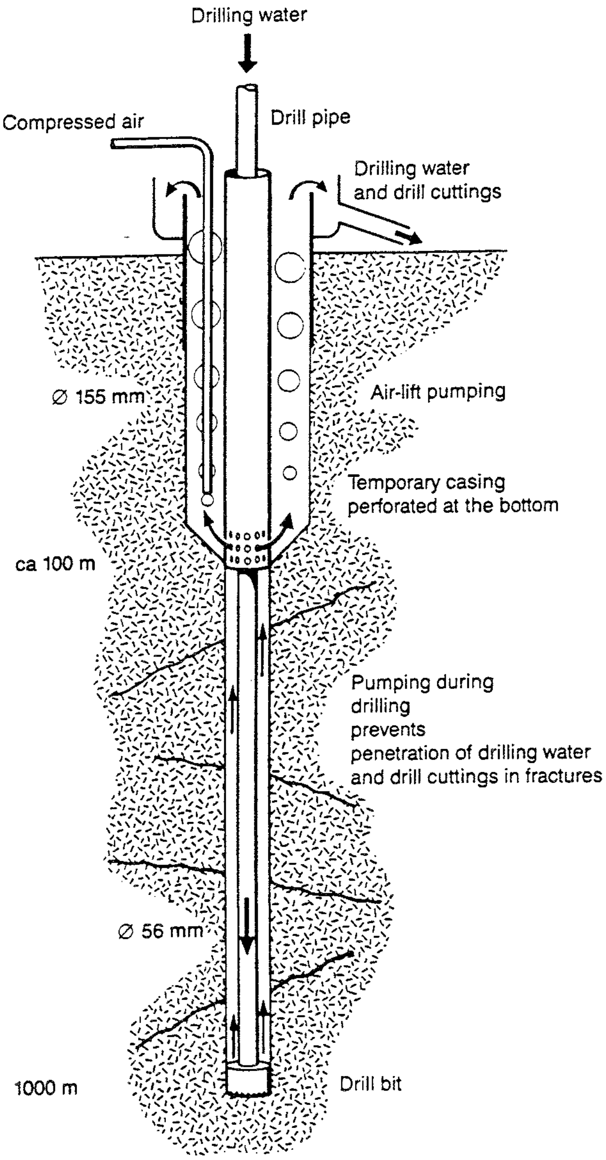

Figure 5.1 Principle of telescope-type drilling technique with air-lift pumping for retrieval of drilling water and cuttings (6)

For the drilling a Tamrock DIOK was used and an Atlas Copco 20 bar 20 m^3/min compressor.

The lifting capacity of the rig was 60 kN and the torque 45 Nm.

The drilling was performed in two steps. First a Ø 165 mm pilot hole was drilled and then this was reamed to Ø 215 mm.

In both cases the drilling was performed with a 5" down-the-hole-hammer (the DTH-method) and with 4½"standard rods.

To have a straight 85° pilot hole, several sets of stabilizers were used especially at lower parts of the drill string. For the same reason the string had a low centre of gravity as a result of using a heavy rod on top of the hammer. Further, the load on the bit was kept low as was the working pressure on the hammer.

The bit used for the pilot hole was a standard hammer bit with insert buttons and a flat curved surface.

Only air was used for flushing and the removal of cuttings.

The reaming to 215 mm was performed using a reamer bit on top of a Ø 165 mm pilot bit - the latter for steering.

To achieve proper removal of cuttings foam was added to the compressed air.

Prior to reaming a conductor casing was set by drilling a 2.5 m deep Ø 305 mm hole into the bedrock. The casing Ø 254 o.d. was then set and fixed using standard cement. The top of the casing was then used as fix point for further drilling.

5.2 Drilling progress

The equipment was brought to drill site and rigged up on one day (5 Oct. '92).

The drilling work was started on 6 Oct. 1992 and progressed according to the description shown in Figure 5.2.

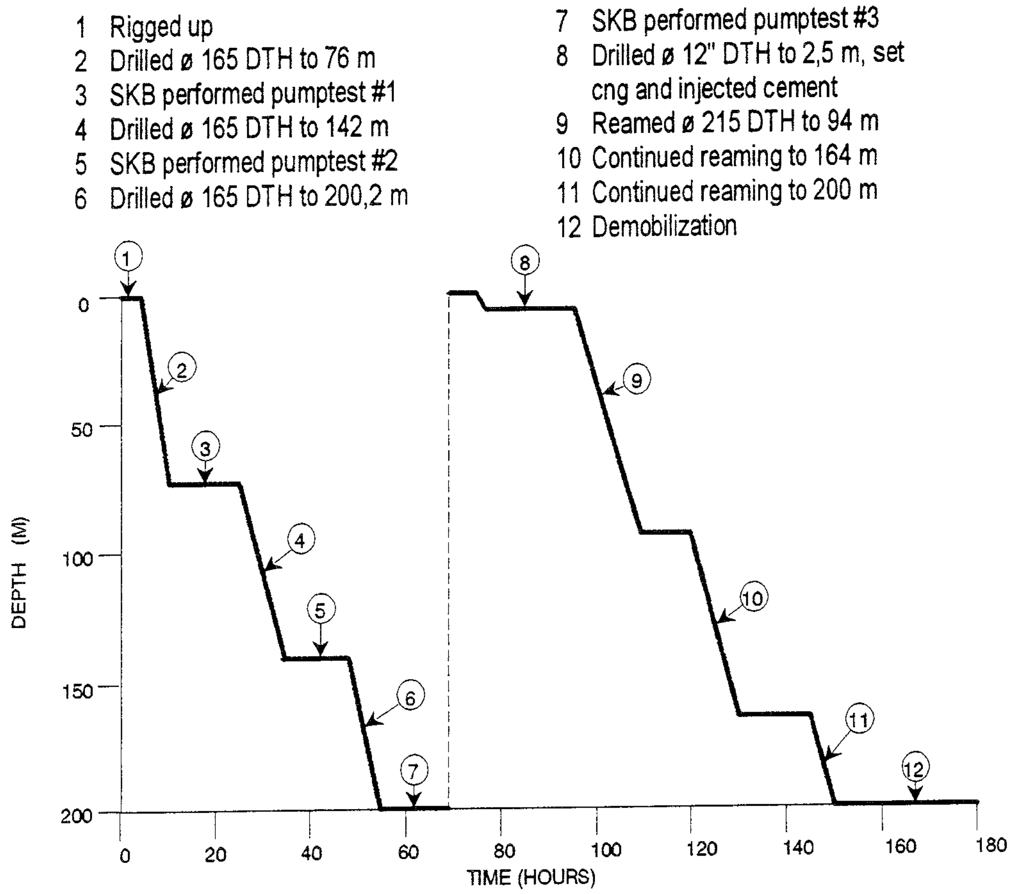

Figure 5.2 Predrilling progress in overview

The pilot hole was drilled to approx 201 m. Then the inclination was measured and since the results showed a deviation less than 1° the decision was taken to continue according to programme.

The reaming landed 0.2 m above the pilot hole depth as planned.

The predrilling part of the programme ended on 12 October 1992. The rig was released and demobilization was finished on 13 Oct. 1992.

5.3 Tests during drilling

During the pilot hole drilling samples of cuttings were taken each 3 m and the rate of penetration was measured manually each 0.2 m.

The cuttings have later been analysed referring to rock type. The result is given in Appendix A.

The rate of penetration is given in figure 5.3. The graph points at relatively higher penetration rates at several levels of which the most pronounced one is situated at a depth of roughly 165 m.

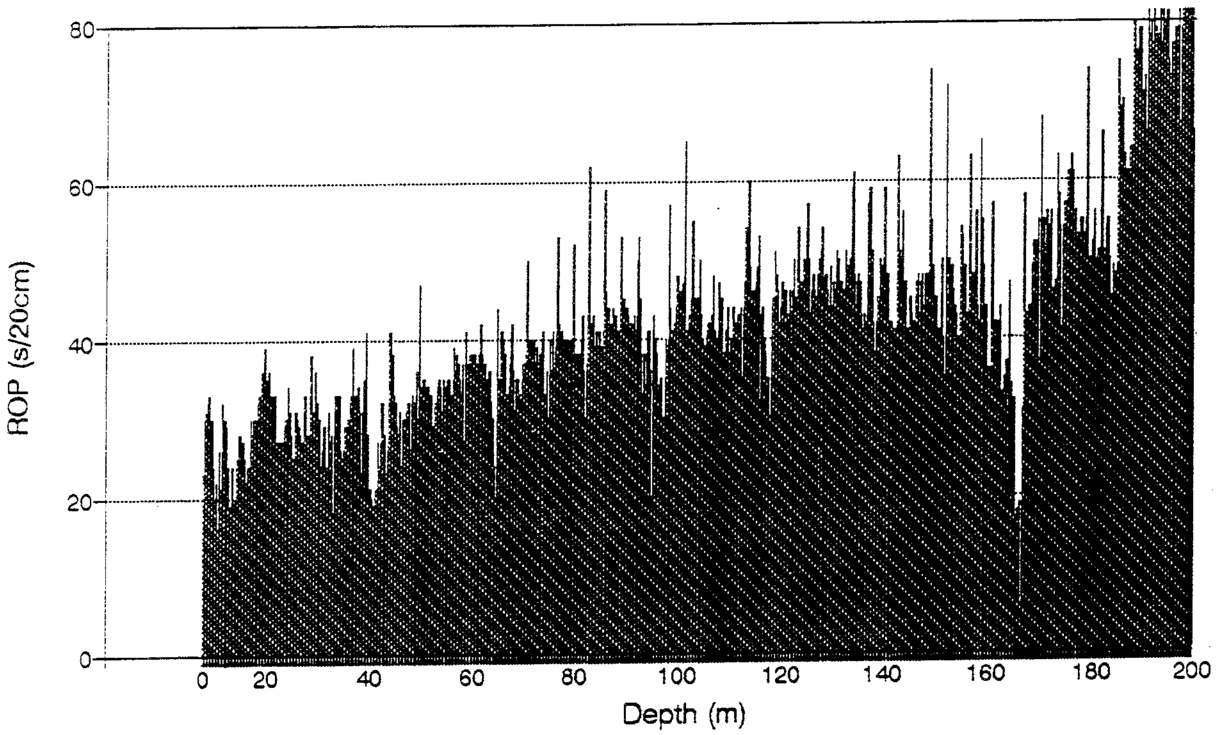

Figure 5.3 The rate of penetration as seconds per 0.2 m measured while drilling the Ø165 mm pilot hole

Furthermore three pumping tests with a submersible pump were made at the depths 76, 142 and 200.2 m.

A planned pumping test after reaming was cancelled due to lack of time. However, this test was carried out after completion of the hole.

Plots of the pumping tests are given in "Technical documents" (5). Key data is shown in Table 5.1.

Table 5.1 Summary of data from pumping tests during predrilling.

Depth (m)	Hours of pumping	Flow rate end of test (m^3/h)	Draw-down end of test (m)	Specific **) capacity ($m^3/h \cdot m$)
76	1.6	2.4	22	0.11
142	7.7	6.9	60	0.11
200.2	8.0	9.0	70	0.16
200.2*)	42.0	8.0	80	0.14

*) performed after core drilling
**) after 1 hour of pumping

As can bee seen from the table there is an increase in the specific capacity at 200 m. However, in the second test, performed after the drilling, the capacity decreased slightly.

5.4 Evaluation

The most critical point in the predrilling operation was the possibility of not having a straight pilot hole.

The contracted demand was set to a maximum of 1° deviation from the starting angle (85°) and, from experience, this is known to be difficult to achieve. However, the drilling method used, with stabilizers, adapted drilling parameters, a slightly oversized rig and skilful drillers, proved to be very successful.

A second concern was the probability of hitting highly permeable fracture zones making it impossible to continue DTH drilling without cement injection or sealing with a casing.

However, the flow never peaked to a critical level but it was high enough to reduce some of the air-lift draw-down effect (see also in Section 7.5). That problem was recognized at an early stage and a seal-off casing was discussed but not employed.

There were two reasons for not casing off the hole. No properly sized casing was on site and with the core drilling rig already on site it would have delayed the programme and considerably increased the cost. The other reason was that even if a casing were to be installed, the space between the core drilling support casing and the sealing casing would be critically limited for safe installation of the air-lift equipment.

Inclination measurements, sampling of cuttings, rate of penetration measurements were all performed according to programme without any remarkable deviations or problems.

The rate of penetration (Figure 5.3) indicates several weaker zones. The most pronounced one is located at approximately 165-170 m and represents most probably a fracture zone.

From the general slope of the ROP-curve there are strong indications that water entrance to the hole takes place at the levels of 40, 167, and at 185-190 m. The slow rate of penetration at the end of drilling indicates that the level 185-190 m has a high influence on the capacity growth shown in Table 5.1.

The decrease of capacity shown in the second pumping test covering the whole section might be the result of cuttings entering the fractures during core drilling. Such conditions existed when drilling the section approx. 300-1100 m (see further in Section 6.3).

6. CORE DRILLING PERFORMANCE

6.1 Rig set up and technical data

The rig and side equipment used were shipped from Canada to Gothenburg and then transported by truck to the drilling site.

The rig was a skid mounted, hydraulically driven Longyear Hydro 55, constructed for wire-line core drilling.

The skids were put on a cross-timber construction, leaving the rig floor approx 0.7 m above the ground.

The mast was 18 m high. It was attached to the drill aggregate by sliding bolts and raised to the drilling angle. It was then leaned back on two supporting legs. At the level 12 m above rig-floor the mast had a derrick platform. From here to the top swivel head there was a rod slipper board. Figure 6.1 gives an idea of rig size and drillsite arrangements.

The prime motor was a Caterpillar 3306T, turbo-charged diesel, with a rated power of 250 hp (149 kw).

There were two hydraulic systems, one for power delivery to the spindle and main hoist, the other for driving the retraction system, chuck and feed cylinders. The hydraulic systems were rated for 20.7 and 8.3 MPa respectively.

Figure 6.1 The wire-line rig at place showing the rig-floor shelter, the mast and the supporting legs. Notice the man on the derrick platform. Photo K-E Almén.

The main hoist with a Ø 19 mm cable had a rated pulling strength of approx 120 kN.

The wire-line hoist had a Ø 4.8 mm 3800 m long cable. The maximum pulling strength was some 15 kN and the adjustable line speed maximum was approx. 240 m/min. It had a manually operated band brake.

The swivel head had two gripping cylinders 101 mm i.d. The spindle i.d. was 98.4 mm and the feed length 1067 mm. The maximum rated thrust was set to 100 kN and the lifting capacity for the swivel head 110 kN.

The transmission system had mechanical spur gears with five different speeds. All gears were reversible.

The rated rotation speed and maximum bit torque each gear is given in Table 6.1.

Table 6.1 Rotation interval area and maximum bit torque at 20 MPa thrust using different gears.

Gear	Rotation area (rpm)	Max torque (Nm)
1	20 - 230	3900
2	45 - 470	1900
3	75 - 720	1250
4	95 - 960	950
5	130 -1250	700

The hydro chuck was self adjustable to handle all rods and casings from 54 to 121 mm o.d. The lifting capacity was in the order of 200 kN under a non-rotating load.

All lifting facilities taken together under non-rotating conditions gave the rig a total lifting capacity of approx. 330 kN and a thrust capacity of approx 100 kN.

6.2 Drilling method

For coring, the wire-line (WL) system was used. The main feature of this system is the reduction in rod pulling. When the core barrel is filled, an inner tube containing the core is detached from the core barrel assembly. The tube and the core are then pulled to the surface by a wire dropped down the line of drill rods. A latch or 'overshot assembly' which snaps onto the top of the inner tube is used for this purpose. The inner tube is then rapidly hoisted to surface inside the drill rod string.

After the core is removed, the inner tube is dropped down into the outer core barrel and drilling is started again. To save time a double set of inner tubes are normally used.

The drill rods are normally pulled only when it is time to change the diamond bit. The main hoist is then used for lifting the rods and the hydro chuck for holding and breaking the rods.

The length of each broken string section was in the KLX 02 case 12 m, and the strings were placed standing from the rig floor up to the derrick platform.

Normally grease is put on the rods to reduce the friction from the wall of the borehole, or a lubricating additive to the flushing water. In this case, due to a risk of contamination grease was not allowed and only pure water was used as the flushing media.

The dimension was a standard NQ which means Ø 69.9/60.3 mm drill rods and Ø 75.7/47.6 diamond core bit. The rods were in lengths of 3.0 m and had standard flush-coupled thread-joints with 3 threads per inch. The weight was approx 8 kg/m.

A standard double tube core barrel was used, six metres long, threaded over half its length, and with a reaming shell slightly above the bit. At some occasions the lower 3 m part of core barrel had thin special designed flow channels to reduce the flow resistance in the interspace between the wall and barrel.

The bits used were diamond impregnated ones with straight wall and flat curving surface.

For back-up reasons rods, barrels and bits for BQ drilling were on site. In case of severe drilling problems, the concept was to use the NQ rods as casing and continue drilling with BQ equipment to full depth.

Flushing water was marked with a tracer (Uranine) in a set of 2x5 m^3 mud tanks. The tracer concentration was 100 μg/l water. The flushing pump used was a 535 FMX Triplex pump rated for 130 l/min at 1 MPa and 60 l/min at 5 MPa.

To decrease the bottom hole pressure an air-lift pumping system was installed prior to core drilling (see further next chapter).

6.3 Support casing and air-lift system installation

On recommendations from the contractor a double supporting casing was installed in the predrilled hole.

The inner casing was a standard NW flush coupled tube of string Ø 88.9/76.2 mm. This was drilled down approx 1 m with a HQ core bit Ø 96.0/63.5 mm, then pulled again for bit removal. It was then put back, perforated with Ø 10 mm holes on the lower 9 m. The distance between the holes was approx 100 mm.

The outer casing HW Ø 114.3/101.6 mm was also perforated over the lower 9 m. One set of centralizers were welded onto the casing surface each 15 m, giving the casing an effective diameter of 210 mm.

The casing string was then set directly on the bottom of the reamed predrilled hole.

Along with the outer casing five plastic tubes were set, fixed with tape. Two of the tubes Ø 22 mm ended 90 m below the surface, two others, Ø 22 and Ø 28 mm, at 160 m and the fifth tube, Ø 28 mm, at 190 m. The first four were for air-lift purposes and the fifth one for draw-down monitoring. All of them were designed for 20 MPa internal pressure. The system in overview is shown in Figure 6.2.

To start with an electrically driven compressor designed for 12 MPa and 2 m^3/min was attached to the uppermost tubes. Later a diesel 20 MPa, 20 m^3/min compressor was used, linked to the tubes at 160 m and on some occasions to the tube at 190 m.

For monitoring the hydraulic head in the hole, a 10 MPa pressure transducer was placed in the lowermost tube. Pressure data was then recorded by a logger with one reading each 5 minutes.

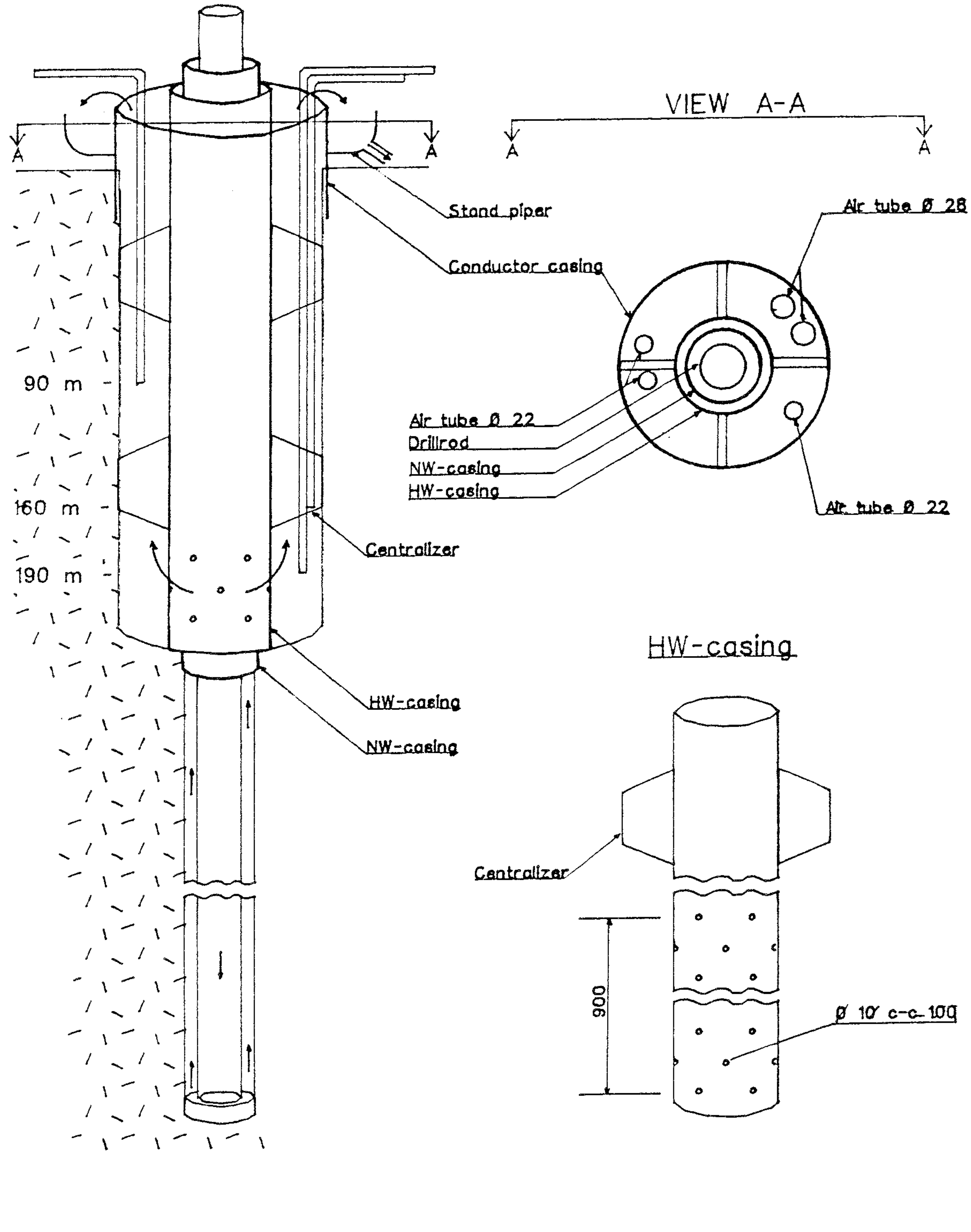

Figure 6.2 The support casing and the air-lift system design

6.4 Drilling progress

The progress of core drilling was recorded on daily drilling reports (5). For visualization, the actual depth as a function of time (working hours) were continuously plotted on a graph also showing main events during drilling. An overview of progress is shown in Figure 6.3.

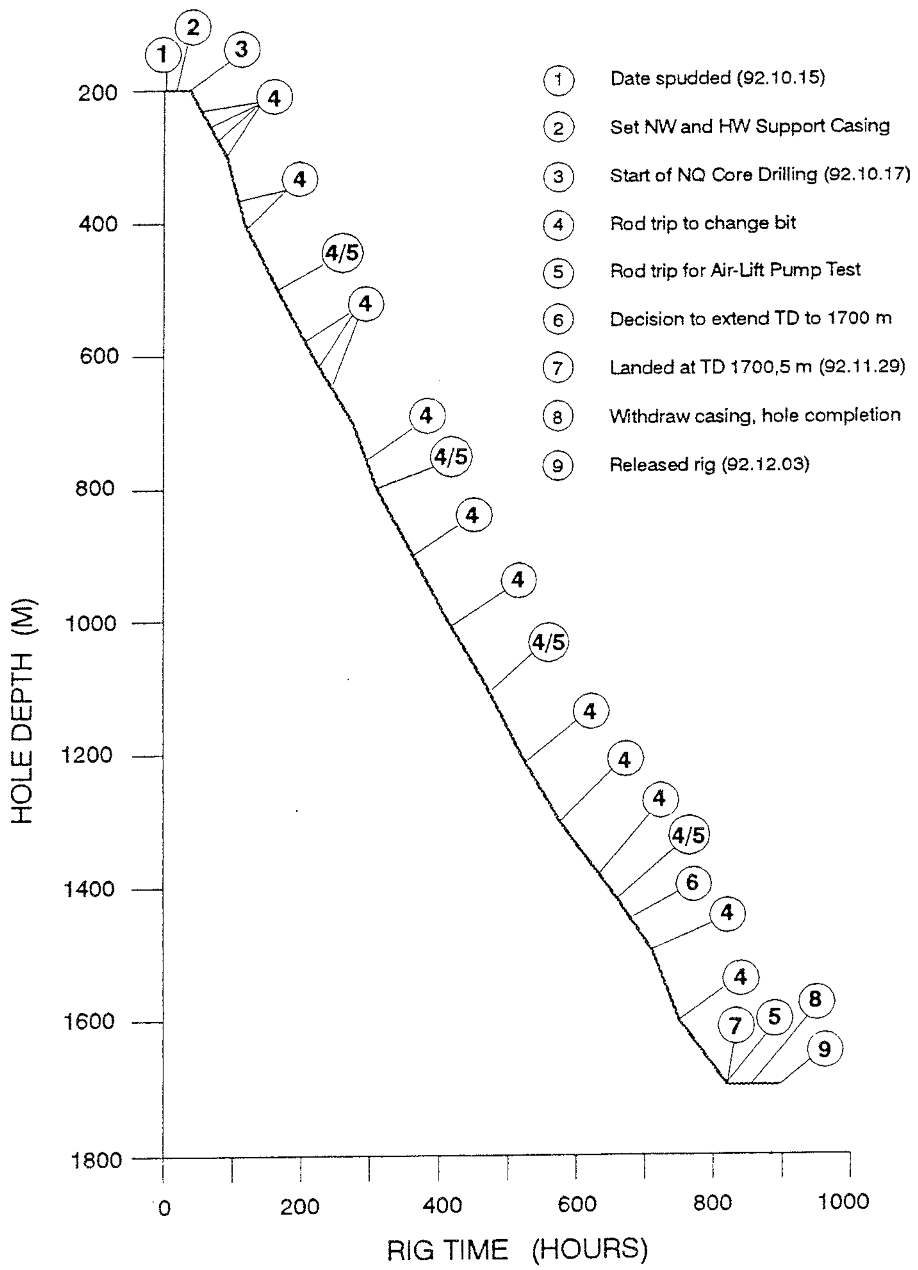

Figure 6.3 Overview of the core drilling progress

Starting from the hour of spudding the effective drilling time needed to reach 1700 m was approximately 825 hours. Approximately 25 working hours were spent setting casing and installing air-lift equipment. Roughly 65 hours of work was used for completion of the hole and 95 rig rental hours for air-lift pumping tests.

Initially, some minor drilling problems occurred and bit changes were frequent. However, quite soon the drilling rate progressed to an almost straight line which was maintained all the way down.

For repairs only seven shutdown hours were used, split into the following items:

-	swivel	1 h
-	hydraulic hose	3 h
-	broken rod	1.5 h
-	broken cable	1.5 h

Another shutdown for seven hours occurred due to insufficient electric power supply resulting in a loss of flushing water.

At 250 m there was a considerable loss of flushing water into the hole probably due to open rock fractures. Further down the return water continued at a much lower rate than normal and this caused hole-cleaning problems on several occasions. The flow pressure and the rotation torque increased to almost critical limits and at the level of approx. 900 m a decision was taken to change the compressor to a bigger size.

The new compressor (20 bar/20 m^3/min) was attached to the two lowermost air-lift tubes and from the level 1050 m it was run at full capacity. In this way the cleaning effect was greatly increased and the rest of the drilling could be performed without further circulation problems.

6.5 Inclination measurements

From 200 to 500 m inclination was measured using an inclinometer at about each 50 m and from 500 to 1700 m it was measured using the acid method at about each 100 metres.

As can be seen in table 6.2 the inclinometer measurements show an almost straight dip very close to the initial at 85°. The deviation is roughly ±1°, or at the level of the instrument's accuracy.

Table 6.2 Results of inclination measurements during drilling. From 590 m the acid method was used.

Depth	Incl.		Depth	Incl.		Depth	Incl.
0	85,5		210	85,3		420	84,3
10	85,2		220	85,4		430	84,2
20	85,2		230	85,6		440	84,2
30	85,5		240	85,4		450	84,0
40	85,3		250	85,3		460	84,2
50	85,2		260	85,1		470	84,0
60	85,3		270	85,4		480	84,0
70	85,2		280	85,3		490	84,0
80	85,3		290	85,5		500	84,0
90	85,2		300	85,3		590	85
100	85,2		310	85,1		750	85
110	85,1		320	85,1		950	85
120	85,2		330	85,3		975	84
130	85,2		340	85,1		1100	84
140	85,2		350	85,1		1200	85
150	85,2		360	84,6		1302	85
160	85,3		370	84,6		1400	85
170	85,0		380	84,5		1500	84
180	85,3		390	84,7		1600	84
190	85,2		400	84,5		1700	83
200	85,3		410	84,4			

The acid method is a manual measurement of an etched line inside a glass tube. The acid used was 48 % hydrofluoric acid. The tube was lowered inside the drill rods and left there for approximately 20 minutes for etching. Then the angle of the line was measured using a protractor. The method gives a rough but safe measurement with an accuracy of ± 3°. The result indicates a continuous straight hole to full depth.

6.6 Drilling parameters and flushing water control

Drilling parameters with respect to

- Rate of penetration (ROP)
- Weight on bit (WOB)
- Rate of rotation (RPM)

and flushing water with respect to

- Entering pressure (FWP)
- Flow rate into hole (FRI) and
- Flow rate return (FRR)

were continuously monitored by means of a recorder, called Geoprinter.

As a sensor for ROP a spring-muted wire was attached to the swivel-head giving rate and depth values.

For WOB recording two pressure sensors were placed one at each side of the swivel feed cylinder. In this way the differential pressure was registered. For reduction of the drill string weight, the driller had to correct manually each core trip by zero initial adjustment prior to drilling.

For RPM monitoring an optical sensor read each turn the rods made as a function of time.

FWP was recorded from a pressure transducer placed on the tube between the mud pump and the swivel. At the same spot the flushing water flow rate (FRI) was measured by a magnetic flow meter while flushing water return (FRR) was measured by the same type of flow meter on the disposal line (see Figure 4.1 above).

Rotation torque was not automatically measured since the transmission system was mechanical. However, the torque could be visually checked on a meter by the driller.

Data was collected and memorized in the recorder with six inlet channels and a storage capacity of approximately 50 m of drilling.

The stored data were collected once a day together with a printout. The data was then processed using notes from the drilling contractor. This version was then plotted on graphs for a preliminary evaluation (7).

Several monitoring problems occurred. The most important ones were

1. Actual depth and therefore also ROP was not accurately measured due to rod-slipping against the gripping cylinders. However, levels for slipping were registered and some corrections have been made in the data processing stage.

2. Since the main wire hoist was used to hold the drill string at levels below approx. 600 m (to keep the load on the bit smaller than the holding capacity of the swivel), the data on thrust (WOB) from this level and downwards are not accurate.

3. On some occasions the driller failed to print the actual depth or printed wrong depths. In some cases he also forgot to zero for thrust.

4. Due to a mechanical damage on a cable RPM values down to approximately 900 m are not accurate.

5. Due to clogging in the flow meter and later also a mechanical failure the return water FRR was not properly registered at levels above about 1200 m. However, manually performed measurements give accurate values on the daily flow rates.

The data coverage is in the order of 90%. Another approx. 10% is of unreliable quality due to the slipping problem, missreadings and problems with editing. Still some 80% of the hole length is covered by data that might be used for further evaluations.

To give a rough understanding of the drilling and flow parameters as a function of depth, the graphs were used for calculating mean values each hundred metres of drilling. The results are shown in Table 6.3.

Table 6.3 Rough mean values of drilling and fluid parameters as a function of depth

Depth (m)	ROP (m/min)	WOB (kN)	RPM (r/min)	FWP (MPa)	FRI (l/min)	FRR[1] (l/min)
200-300	- [4]	- [4]	(500)[2]	30	50	30
300-400	0.125	35	(400)	40	45	10
400-500	0.15	30	(500)	40	50	10
500-500	0.15	30	(450)	40	50	10
600-700	0.15	- [3]	(500)	45	45	10
700-800	0.15	-	(450)	50	40	10
800-900	0.20	-	(400)	55	45	<10
900-1000	0.175	-	500	60	45	<10
1000-1100	0.175	-	500	50	50	140
1100-1200	0.15	-	500	50	50	140
1200-1300	0.125	-	400	55	50	140
1300-1400	0.15	-	400	55	50	140
1400-1500	0.175	-	350	60	50	140
1500-1600	0.20	-	400	60	60	140
1600-1700	0.175	-	350	65	55	140

1) Corrected according to manually measured values
2) Uncertain values
3) Not applicable below 600 m
4) No true values due to slipping

As a comment to the table, the rate of penetration (ROP) indicates the hardness of the rock. However, the true drillability is not reflected since the load on the bit (WOB) was not registered properly.

Further, it is quite visible that the flushing water pressure (FWP) increased step-by-step with depth due to a continuously growing flow resistance. However, what are not visible in the mean values are several high pressure peaks, especially from the 850-1050 m section. These peaks are connected to levels with low return water flow and hence bad removal of cuttings. The effect of more compressor power from 1050 m (see further section 6.4) greatly improved the flowrate out of hole (FRR) from less than 10 to 140 l/min.

Finally, since no torque was measured, the speed of rotation (RPM) might be looked upon as a torque indicator. In this respect low values indicate high torque and vice versa. Hence, in upper levels with still low rod friction against the walls, high torque (low RPM) will indicate friction due to bad hole cleaning. Since good cleaning was achieved from approx. 1050 m and downwards high torque below this level is more connected to an increased rod friction (vibration at some levels) and to some extent high bit-load when harder rocks were encountered.

In the raw data report (8), some drilling parameter graphs have been selected for more detailed analyses.

6.7 Core handling and first inspection

After each core-trip, the 6 m barrel was laid out and split into halves. Each half was then carried into the rig shelter and traditionally emptied by letting the core slide directly into a core box.

In fractured rocks core pieces were puzzled together to give the best fit. This was performed by the driller.

The core boxes were marked and carried to a temporary container at the drilling site. After inspection and first geological classification by the site geologist the boxes were transported to a central core store.

The result of the first core inspection is to be found in reference (8) and a shematic description of the main geological features is given in table 6.4.

Recently the cores have been detailed examined by the use of a computerized system named Petro Core, see further reference (6). The result is presented in Appendix A.

Tabel 6.4 Main geological features based on the preliminary core inspection

Depth (m)	MAIN GEOLOGICAL FEATURES
0-200	Predrilled hole by DTH hammer. Cuttings indicate diorite at top. From approx 70 m granite. Several fissures or fracture zones
200-470	Mainly granite, light red-light grey. Medium grained. Greenstone beds at 350-380 m. Single smaller fracture zones
470-500	Mainly granodiorite, grey and medium grained. Several intrusions or beds of greenstone. Rare in fissures and fractures
800-1120	Mainly granite, redgrey-red and medium grained. Repeated occurrence of greenstone. Frequently tectonized and fractured
1120-1450	Mainly granite or granodiorite, light grey-grey. Medium grained. One bed of greenstone (1370-1405). Extreamly rare in fissures and fractures
1450-1560	Mainly granodiorite, light grey-grey. Medium grained. Extreamly rare in fissures and fractures
1560-1700	Mainly granodiorite as above. Singel beds of granite. Partly commonly fractured and tectonized

7. CORE DRILLING EVALUATION

7.1 Drilling efficiency

The core drilling efficiency is normally measured as a rate of penetration (m/h), trips and shut-downs for repairs included.

As such, the efficiency reflects several parameters. The most important ones are

- Hole depth
- Rock properties
- Standard of equipment
- Skill of drillers
- Weather conditions

In this case the equipment was of excellent standard as was the skill of the drillers. Combined with reasonable weather conditions the efficiency turned out to be considerably higher than expected. The mean value landed at 1.84 m/h compared with the expected 1.3.

According to the drillers an optimal value would be in the order of 2.2 if grease or mud was allowed to be used and if the initial knowledge of how to control the drilling parameters to the rock properties had been available.

As can be seen in the graph (Figure 7.1) the first hundred metres of drilling involved the lowest efficiency, reflecting an initial learning process. Further down the efficiency varied between 1.6 and 2.2, with lower values indicating hard rock properties.

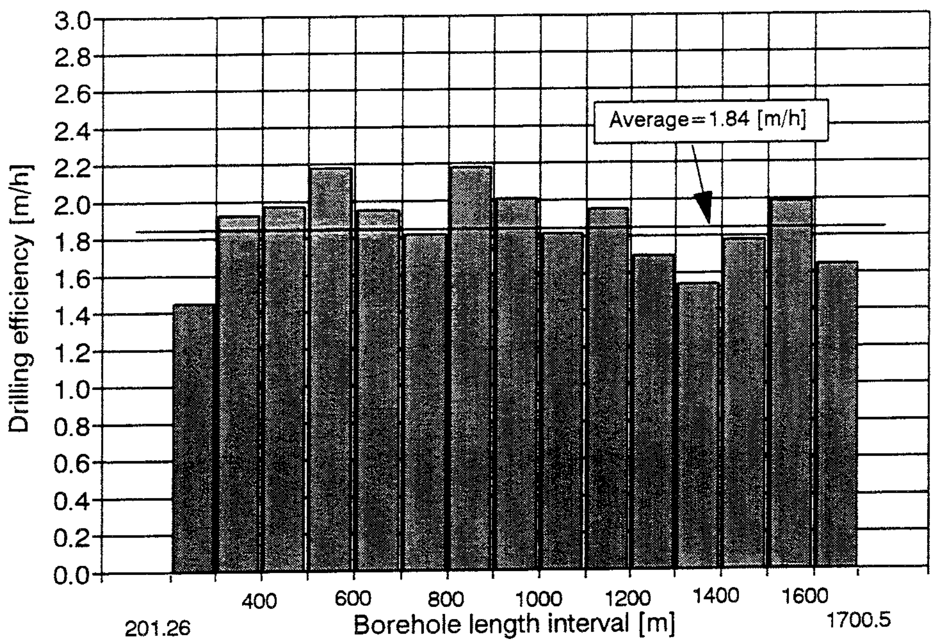

Figure 7.1 Drilling efficiency as a function of depth

One would expect a much more pronounced general decrease of efficiency as the hole got deeper due to higher torque and more time for core removal. However, a statistical analysis of time consumption between each core-trip indicates an almost equal time lap no matter what depth (see Figure 7.2).

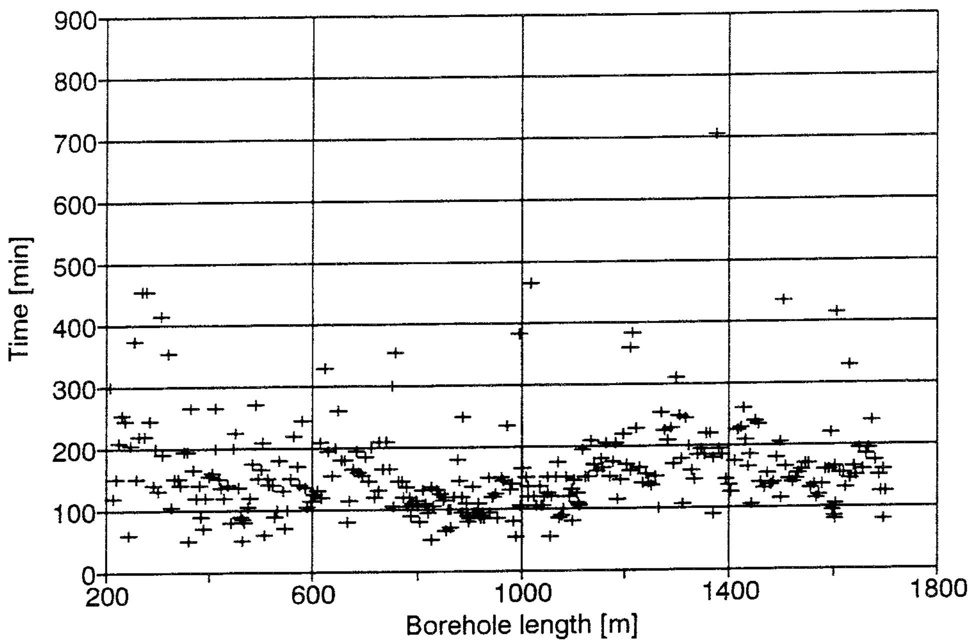

Figure 7.2 Time between core-trips as a function of depth

Still, the speed difference between shallow and deeper depths can be noticed on the minimum values showing some 40-50 minutes in the upper part of the hole and 80-90 minutes at the deepest positions.

The scattered dots above the approximately 250-minute level are round-trips with the drill rods to change the bit or to perform pumping tests.

7.2 Consumption of bits and shells

According to the daily reports a total of 22 diamond bits were used. Of these 18 were fully worn out. Further, three sets of reaming shells were used.

Except for one experimental bit, standard diamond impregnated ones for hard rock were used during the whole operation. To start with the average duration of each bit was low, 20-30 m but due to improvements in adjusting the drilling parameters to the existing rock property, the length of life was increased considerably to reach 80-100 m in the lower half of the hole.

The average length of life turned out to be approx. 70 m (see Figure 7.3).

The longest section drilled with one and the same bit was 135 m at the level 618-753 m.

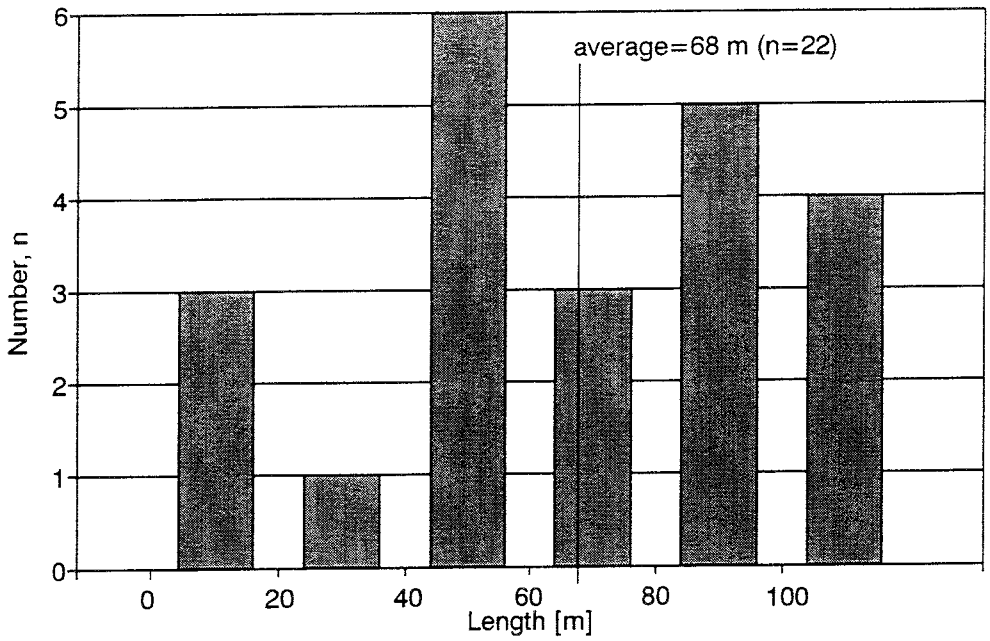

Figure 7.3 Frequency of bit duration (metres per bit)

7.3 Core length statistics

The deepening of KLX 01 by means of conventional coring as background information, the expected mean core length was set at approx. 4 m. It was also expected that during drilling several zones were bound to be penetrated with a 3 m core barrel.

The actual results were that the 6 m barrel was used all the way and the mean core length finished at 5 m.

The frequencies of core lengths for the whole section and split into 300 m intervals are shown in Figure 7.4.

The graphs show quite clearly that most of the core trips were performed with a full or nearly full barrel.

However, the section 800-1100 m is an exception showing a mean value of less than 4 m. By way of explanation this level represented the most fractured one.

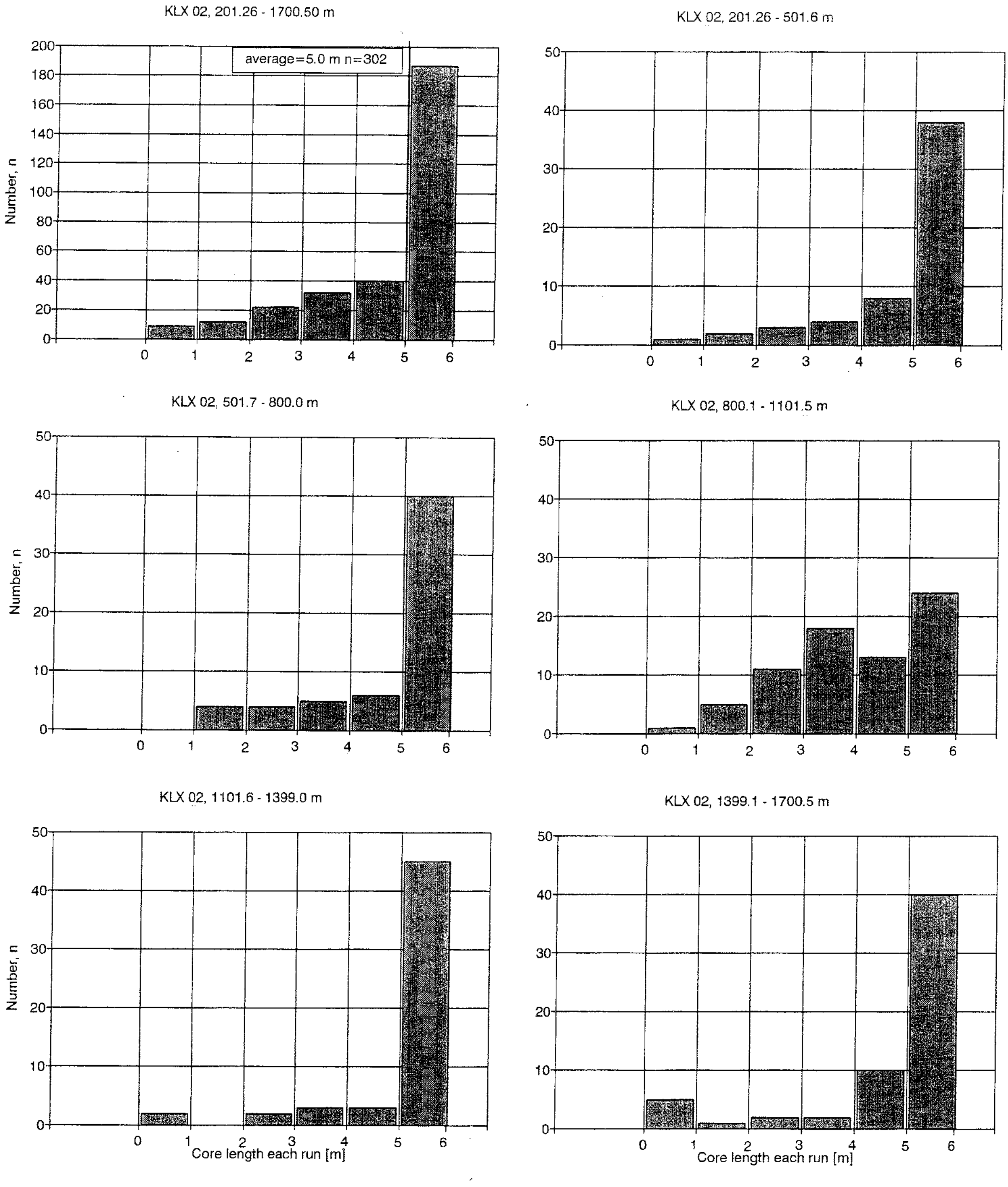

Figure 7.4 Frequencies of core lengths, total and split into 300 m intervals

7.4 Core recovery

The core recovery was close to optimal since only 0.6 m was classified as losses.

The piece of core was lost at the level 275 m. It was caused by a left core, three metres long, that was not fully recored and recovered.

The result was in general better than expected considering the depth and rock properties. The successful recovery (99.96%) might be explained by the WL method in combination with proper equipment and skilful drillers.

7.5 Air-lift efficiency

In theory the continuous air-lift in the predrilled hole would decrease the hydraulic pressure in the borehole and therefore reduce the entrance of drilling fluid and cuttings to the rock (see Figure 7.5).

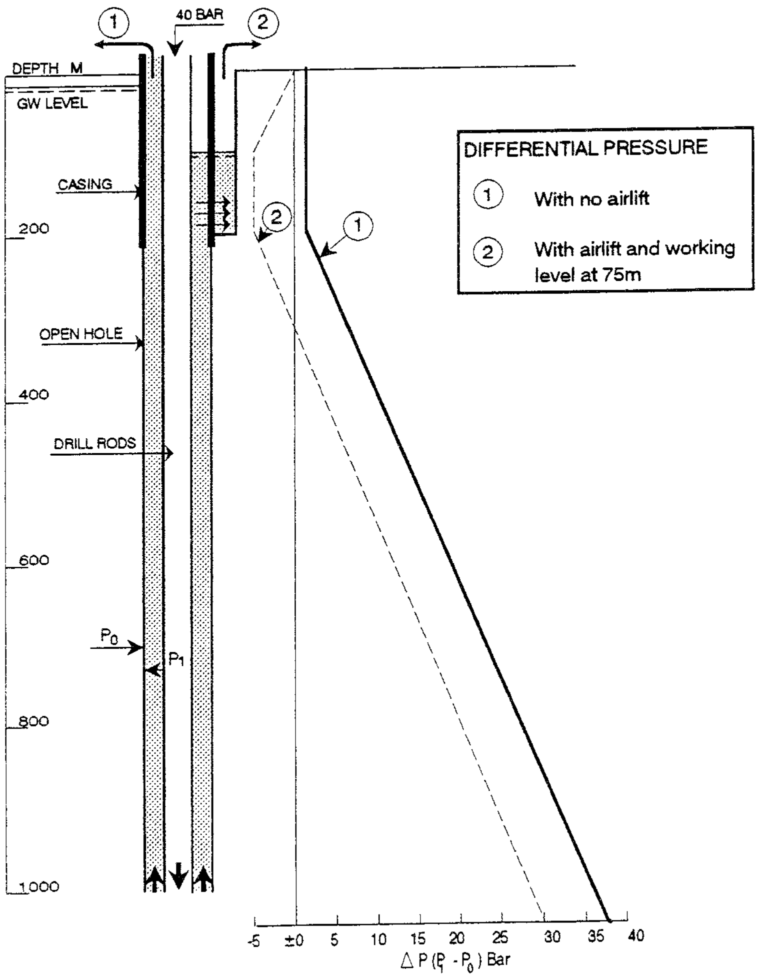

Figure 7.5 Schematic illustration of the hydraulic differential pressure in the borehole during drilling with and without air-lift pumping

The main reasons for using the air-lift system was to minimize the contamination of the formation water and to avoid clogging of fractures by entrance of fine grained cuttings.

From a technical drilling point of view the air-lift system would help to circulate the drilling fluid and in that sense also be of assistance in keeping the hole clean from cuttings. On the other hand the hazard would be a potential risk of having insufficient pressure during core-trips or other stand-by situations leading to unstable hole conditions.

To start with the system worked as planned. However, at the level of approx. 250 m there was a considerable loss of circulation that reduced the flow-rate from the air-lifting system. The low portion of return water then continued till the point a new compressor was used for pumping at a lower level (approx 1050 m). The intervening space was then cleared of sedimented cuttings and the flow rate recovered to the initial one. The beneficial effect of this event is clearly shown in Figure 7.6 where the concentration of the tracer in the return water is plotted (see also return flowrates in table 6.3).

The plot shows an initial dilution of the return water by a factor of 2-3. From approx. 350 m the dilution decreased and at 500 m it is practically none. This means that much less formation water from the upper part of the hole was produced indicating clogged conditions. This state continued until the new compressor was set in at 1050 m. The dilution then started to grow and ended at a factor of 3-4.

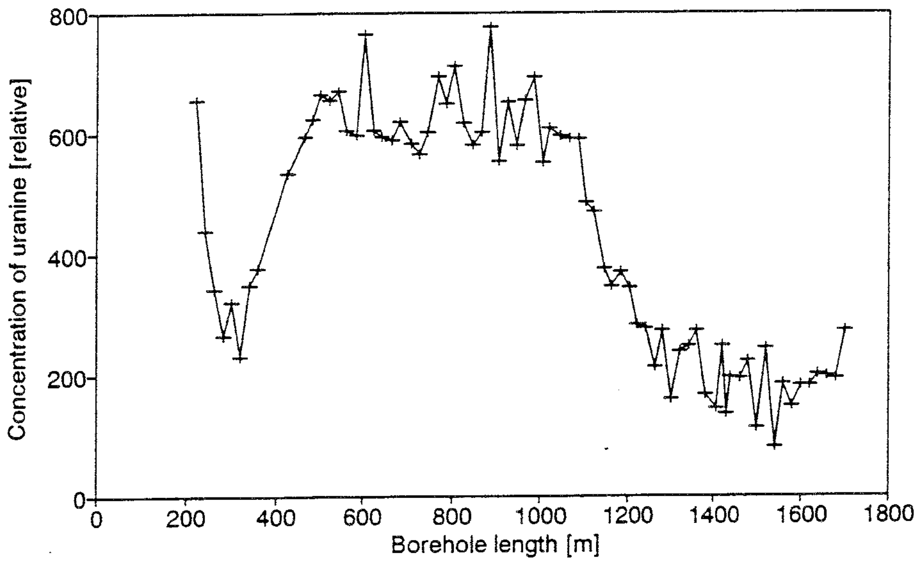

Figure 7.6 Concentration of the tracer in the return flushing water

The clogging process is not fully understood. However, it is obvious that due to a 45 times larger flow-area in the predrilled Ø 215 hole than in the Ø 76 hole the conditions for sedimentation of cuttings were provided. One way or another cuttings seem gradually to have settled in the space between the outer casing and the wall in the predrilled hole. Finally the filling reached the level for the outlet of air (160 m). Hence, the water-producing fractures at 167 and 185 m were cut off. At this stage all return fluid probably flowed in the space between the inner and outer casing and/or in the space between the rods and inner casing. Under such circumstances there was less pressure reduction from the air-lift system in the hole. As can be seen from flow measurements (see Table 6.3 above) only 15-25% of the fluid returned while the rest entered the rock.

It was proved that sedimentation of cuttings can be prevented by using more air. However, this means a big compressor at high rates. It is probably a better choice to reconstruct the casing-air-lift system. One way would be to use the interspace between the two casings as a channel for air and return fluid.

7.6 Flushing water monitoring

To keep a track on how much water is entering the hole and how much is coming back, the volumes of water in and out were monitored (see further Sections 4.4 and 6.6).

The aim was to continuously log the volumes and to have them accumulated as a function of time and hole depth. By using a tracer (see Section 4.4) added to the input flushing water, it is also possible to calculate the portion of flushing water coming out of hole again. The rest of the water out of hole would be formation water extracted by the air-lift pumping.

It was early recognized that the flow meter for logging the return volume did not function. Instead the flow was measured manually at the disposal containers.

The reasons for the failed monitored flow-rate were several:

- no pre-sedimentation of cuttings to start with, resulting in clogging in the flow-line

- down-the-hill conditions for the flow line resulting in unfilled tubing

- very low flow rates due to the bad support from the air-lift system (300-1050 m)

- mechanical fault within the flow meter (200-approx. 900 m).

To avoid problems with unfilled tubing and impact of cuttings the system needs improvements. Principally that can be achieved by a modification of the system according to Figure 7.7.

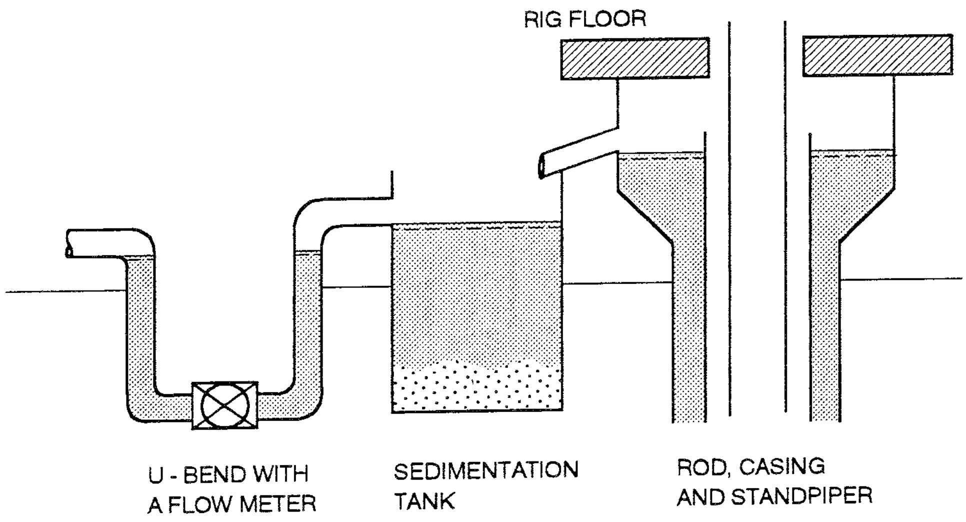

Figure 7.7 Suggested modification of system for monitoring of return flushing water

7.7 The usage of drilling parameters

Drilling parameters have in theory at least two practical applications:

\+ a continuous recording of different forces used in the process of drilling (technical optimization)

\+ a continuous mapping of certain mechanical properties of the rocks penetrated (geological documentation).

If not properly measured, the use of the data for evaluation of drilling technique or geological features are very limited.

In this case several monitoring disturbances occurred due to mechanical failures (see Section 6.6).

Furthermore, the torque was not measured at all and the lack of this parameter makes it impossible to understand and evaluate the detailed features of hole cleaning.

The rate of penetration is often used as a value of drillability. However, to create a drillability index corrections have to be made for thrust, speed of rotation, torque and some other less important parameters. In this case some data are missing and others are less reliable. Any attempt to use them for detailed indexation and evaluation does not therefore seem feasible.

For future applications most of the sensors or their attachment to the rig have to be changed or reconstructed. The most important issues are to consider

- a new place/system for rate measurements

- a way to measure torque on mechanically geared rigs

- a more accurate thrust measurement also taking into account mechanical holding forces

- a procedure for analysing the reliability of the parameters on-line.

8. HYDROTESTS

8.1 Method and performance

According to the programme air-lift pumping tests were planned for each approx. 300 m.

The objects of the tests were partly to have a rough estimate of the capacity of each section and partly to take 'undisturbed' samples of the formation water.

However, due to costs involved for the rig in the stand-by position, it was stated that the flow rate had to be reasonable to continue a test. The criterion used was a maximum of 20 pumping hours within which time the turnover of water in the section had to be doubled.

Prior to testing the drill rods were retrieved and the core barrel dismantled. The drill string was then used to sink an inflatable packer down to the top of the test section.

The packer was first hydraulically inflated and then mechanically opened for flow entrance.

The attachment to the wall was tested by gentle pulling and the hydraulic connection with the test section by means of a pressure drop when the valve was opened.

An air-tube, Ø 22 mm, was set inside the drill string at the level of 100 m together with a Ø 15 mm tube for monitoring the static head. The latter tube ended at 125 m.

For air-lifting an 11 bar compressor was used rated for an air-flow of more than 2 m^3/min.

Drawdown and recovery of the static head were recorded manually using an electric probe.

The flow rate was measured visually by letting the mixture of air and water into a closed tank. The tank size was 1 m^3 and it was graduated with 100-l markers.

In all four tests were carried out. A fifth one (section 200-500 m) failed due to problems with the packer.

Main data from the four tests are shown in Table 8.1.The drawdown and recovery plots are to be found in the file for technical documents (5). The pumping tests have recently been hydraulically evaluated (9).

Table 8.1 Main data from air-lift pump tests

Tested level (m.b.rl.)	Flow rate end of test (l/min)	Draw-down (m)	Remarks
508-800	<300	45	Interrupted after 1 h due to low flow rate
798-1101	950	50	Duration 9.5 h
1097-1429	<100	60	Pumped over weekend for 22 h
1427-1700	900	50	Duration 10 h

The reason for interrupting the pumping test 508-800 m was that the criteria for chemical sampling could not be fulfilled within a reasonable amount of pumping hours.

8.2 Method evaluation

The critical part in the performance of the test is to run an unprotected packer at the front of the drill string. In all, seven attempts were made to run the packers but only four were successful. In one of the failed attempts the size of the packer was incorrect. In the two other runs the packer was damaged on its way down.

The experiences stated above clearly indicates a need for a more reliable method. One such method would be to use a packer specially designed for wire line coring (NQWL Packer Type II). With this system the packer is run inside the rods and then expended just beneath. This ensures that no damage occurs when running the packer. The system is also faster since the speed of running and pulling the rods can be higher.

Despite the method, experience also shows that duplicates of certain key equipment should be kept at or close to the drilling site.

8.3 Flow meter logging

Also included in the drilling programme was a flow-log to be run at an early stage of the past-completion programme. The goal with this log was to have a first hint on where the major permeable zones are and to relate this information to events that occurred during drilling.

The log was run after approximately 2 weeks of test and clearance pumping and was performed by ABEM Geo Science. For further information concerning the test and clearance pumping, see reference (9).

The plot of the log (Figure 8.1) indicates three pronounced inflows of water accompanied by sharp changes in conductivity and temperature in the upper part of the hole. These levels are approx 215, 250 and 320 m.

The most dramatic one, at the level 250 m, corresponds quite well with observed fractures and with a sudden loss of circulation during drilling. This loss of fluid most certainly started the sedimentation of cuttings in the predrilled hole (see also Section 7.4).

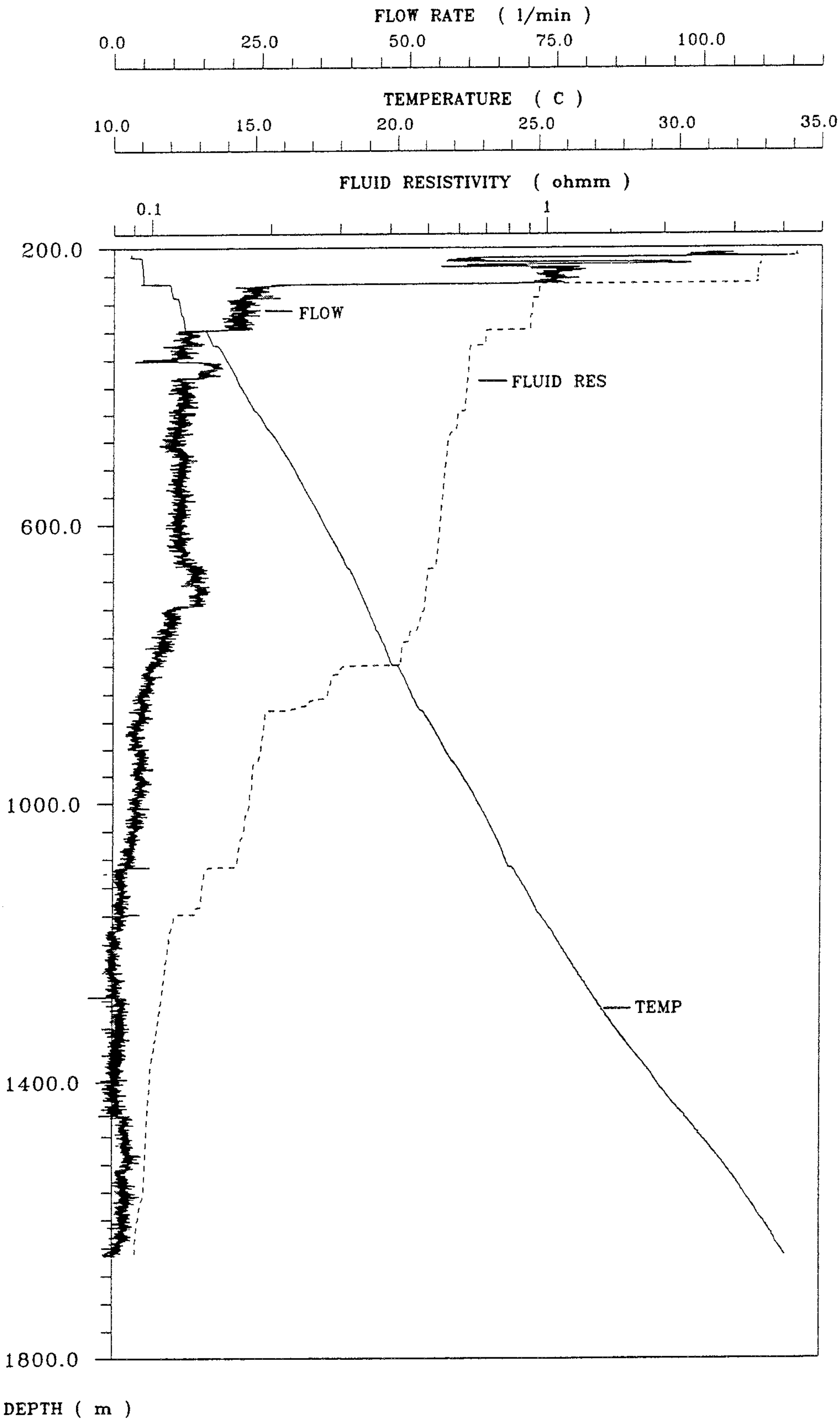

Figure 8.1 Result of the flow meter log

It is also obvious that the zone approx 700-1100 contains several water-producing fractures, but none of them very sharp. However, the conductivity and, to some extent, the temperature indicates single open water-producing fractures at several levels i.e. 800, 860, 1090 and 1110 m.

The probe reached the bottom at 1649 m. This indicates that the hole - at that time - had been filled with approx 50 m of sediments after completion.

9. HOLE COMPLETION

The completion of the operation was performed according to a special programme (8).

The objective was to ensure that a clean and safely accessible hole was left for further tests.

A special issue was to seal off the predrilled hole from the cored hole. This was finally done by running a 8" casing directly placed on the shoulder of the Ø 215 reamed hole (see Figure 9.1).

It must be noted that the final choice of casing, o.d. 193.7 mm, did not allow any sealing by clay pellets as planned. For this reason hydraulic leakage can occur beneath the casing. Such leakage can also be transmitted to the cored hole either through the space between the casing and the transmission cone or along the welded part of the cone entering at the level 202.95 m.

To avoid such possible leakage it is essential to cut off any potential passage of flow between the holes by a proper method.

Since the sealing system has also to be retrievable, cement cannot be used. Soft materials like rubber (also the swelling type) or leather might therefore be a better alternative.

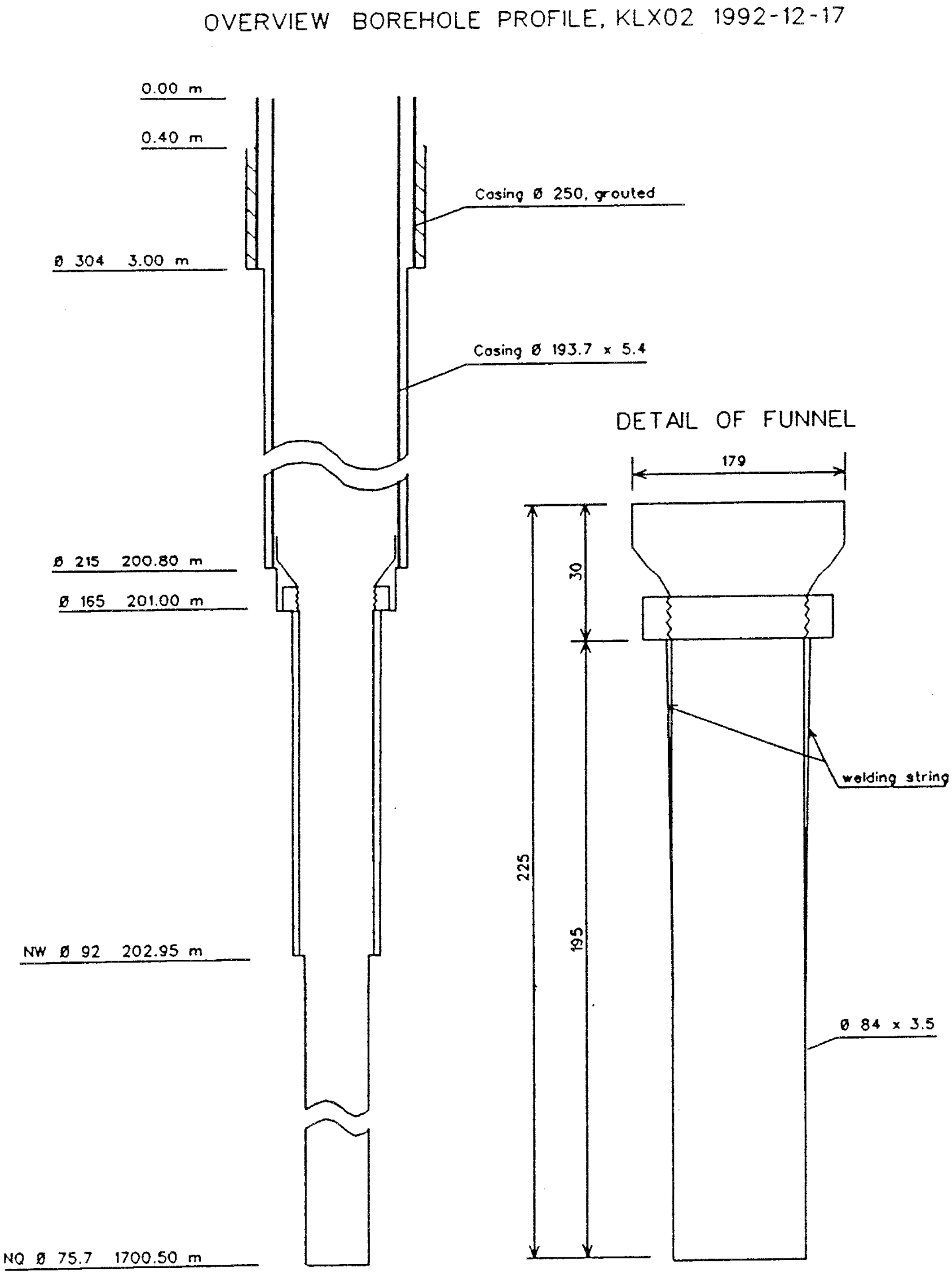

Figure 9.1 Hole configuration after completion

10. GENERAL CONCLUSIONS

The experience gained from the execution of planning, drilling and documentation can be stated in the following general conclusions.

* The procedure concerning planning the drilling proved to be efficient and the same or a similar model can be used for future deep drilling projects.

* The operative organization tested during drilling also proved to function, but was never faced by severe problems. However, for future projects it is essential that the number of site personal be slightly increased.

* The method used for predrilling proved to be successful and can be recommended in future projects.

* The wire-line method, performed by highly experienced drillers and a slightly oversized rig, proved to be effective, especially compared to the conventionally drilled KLX 01. The method is therefore recommended for future deep drilling projects.

* At certain levels critical torque was built up due to high friction between the wall and rods. This has been evaluated to be a function of bad cleaning and/or loss of circulating fluid.

* The air-lift method used to ease the circulation of fluid partly failed to function. It is recommended that the system be improved in order to avoid sedimentation of cuttings.

* The collection of drilling parameters did not function as expected. For future applications improvements are needed. Furthermore it is essential that the torque be measured.

* The measuring system for return water (flow out of the hole) did not function as expected and must be improved by development of a new flow system.

* Most of the hydraulic tests were carried out as planned. However, the method could be improved by using special packers adapted to wire-line rods.

* For a safer sealing between the casing of the predrilled and the core drilled holes at the stage of completion a new sealing-off method should be considered.

* Due to several circumstances, the time gap between predrilling and core drilling had to be kept short. In future projects it is essential to increase the time in order to permit accurate planning and performance of the test programme.

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DEEP DRILLING KLX 02

Results of core logging

A1.	Legend	
A2.	Interval	0-300 m
A3.	Interval	300-900 m
A4.	Interval	600-900 m
A5.	Interval	900-1200 m
A6.	Interval	1200-1500 m
A7.	Interval	1500-1700 m

HOLE : LGD

GEOLOGIST: STR

DATE : 940315

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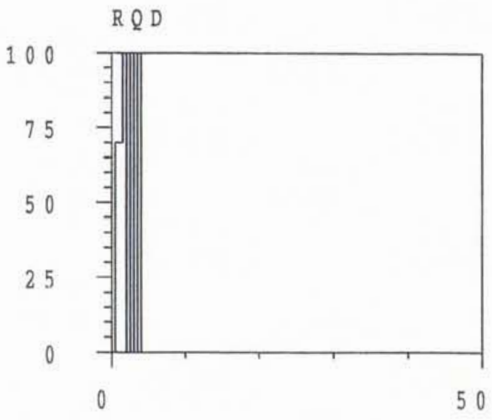

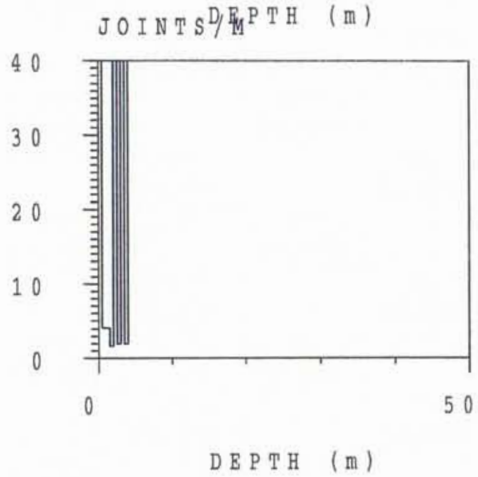

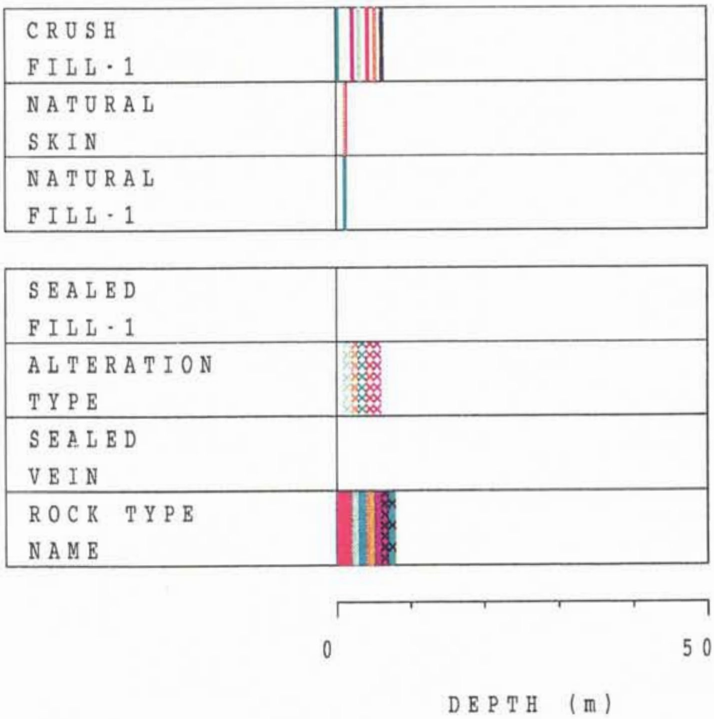

ROCK TYPE NAME

- FINE-GRAINED GRANITE
- ASPO DIORITE
- GEENSTONE
- SMALAND GRANITE
- MYLONITE
- BRECCIA
- GABBRO

ALTERATION TYPE

- EPIDOTISIZED
- WEATHERED
- CHLORITISIZED
- OXIDIZED
- TECTONIZED

NATURAL SKIN

- CAVITIES
- DULL
- WEATHERED
- OPEN

CRUSH FILL-1

- CHLORITE
- CALCITE
- EPIDOTE
- HEMATITE
- IRON OXIDES
- UNCERTAIN CRUSH ZONE

HOLE : LGD

GEOLOGIST: STR

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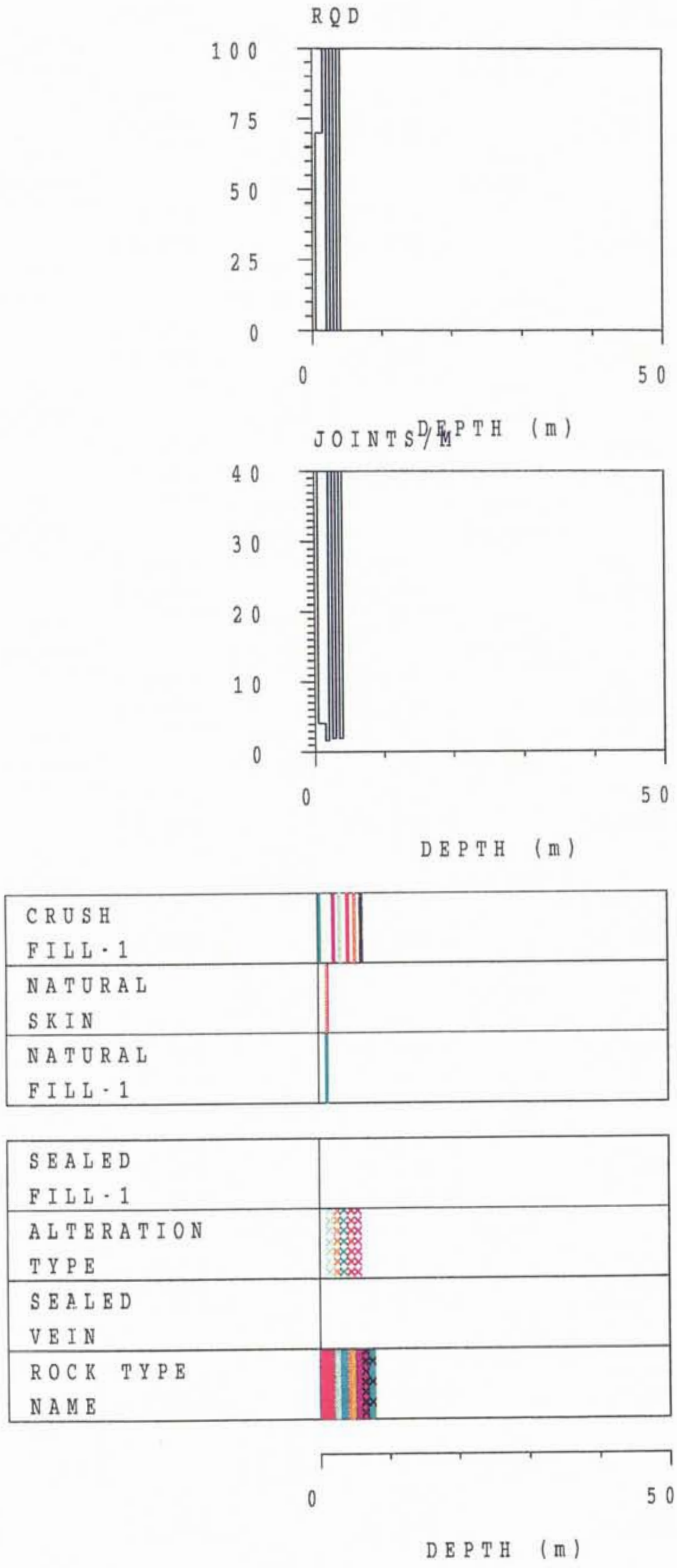

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- HEMATITE
- IRON OXIDES
- UNCERTAIN CRUSH ZONE

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N Platts, D J Blackwood, C C Naish
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Paul Wersin, Kastriot Spahiu, Jordi Bruno
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Reassessment of seismic reflection data from the Finnsjön study site and prospectives for future surveys
Calin Cosma[1], Christopher Juhlin[2], Olle Olsson[3]
[1] Vibrometric Oy, Helsinki, Finland
[2] Section for Solid Earth Physics, Department of Geophysics, Uppsala University, Sweden
[3] Conterra AB, Uppsala, Sweden
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Jan Cramer (ed.)[1], John Smellie (ed.)[2]
[1] AECL, Canada
[2] Conterra AB, Uppsala, Sweden
May 1994

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Sven Åke Larsson[1,2], Eva-Lena Tullborg[2]
[1] Department of Geology, Chalmers University of Technology/Göteborg University
[2] Terralogica AB
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Göran Bäckblom (ed.), Christer Svemar (ed.)
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INTRAVAL Working Group 2 summary report on Phase 2 analysis of the Finnsjön test case
Peter Andersson (ed.)[1], Anders Winberg (ed.)[2]
[1] GEOSIGMA, Uppsala, Sweden
[2] Conterra, Göteborg, Sweden
January 1994

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The structure of conceptual models with application to the Äspö HRL Project
Olle Olsson[1], Göran Bäckblom[2], Gunnar Gustafson[3], Ingvar Rhén[4], Roy Stanfors[5], Peter Wikberg[2]
1 Conterra AB
2 SKB
3 CTH
4 VBB/VIAK
5 RS Consulting
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Kjell O Wannäs, Tom Flodén
Institutionen för geologi och geokemi, Stockholms universitet
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Hans Wanner[1], Yngve Albinsson[2], Erich Wieland[1]
[1] MBT Umwelttechnik AG, Zürich, Switzerland
[2] Chalmers University of Technology, Gothenburg, Sweden
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D J Blackwood, A R Hoch, C C Naish, A Rance, S M Sharland
AEA Technology, Harwell Laboratory, UK
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A R Hoch, S M Sharland
Chemical Studies Department, Radwaste Disposal Division, AEA Decommissioning and Radwaste, Harwell Laboratory, UK
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Hans Widén[1], Patrik Sellin[2]
[1] Kemakta Konsult AB
[2] Svensk Kärnbränslehantering AB
September 1994

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J Henshaw
AEA Technology, Decommissioning & Waste Management/Reactor Services, Harwell, UK
January 1994

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Ignasi Casas[1], Jordi Bruno[2], Ester Cera[1], Robert J Finch[2], Rodney C Ewing[2]
[1] MBT Technología Ambiental, Cerdanyola, Spain
[2] Departement of Earth and Planetary Sciences, University of New Mexico, Albuquerque, NM, USA
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Antti Öhberg[1], Pauli Saksa[2], Henry Ahokas[1], Paula Ruotsalainen[2], Margit Snellman[3]
[1] Saanio & Riekkola Consulting Engineers, Helsinki Finland
[2] Fintact Ky, Helsinki, Finland
[3] Imatran Voima Oy, Helsinki, Finland
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S H Whitaker, A Brown, C C Davison, M Gascoyne, G S Lodha, D R Stevenson, G A Thorne, D Tomsons
AECL Research, Whiteshell Laboratories, Pinawa, Manitoba, Canada
May 1994

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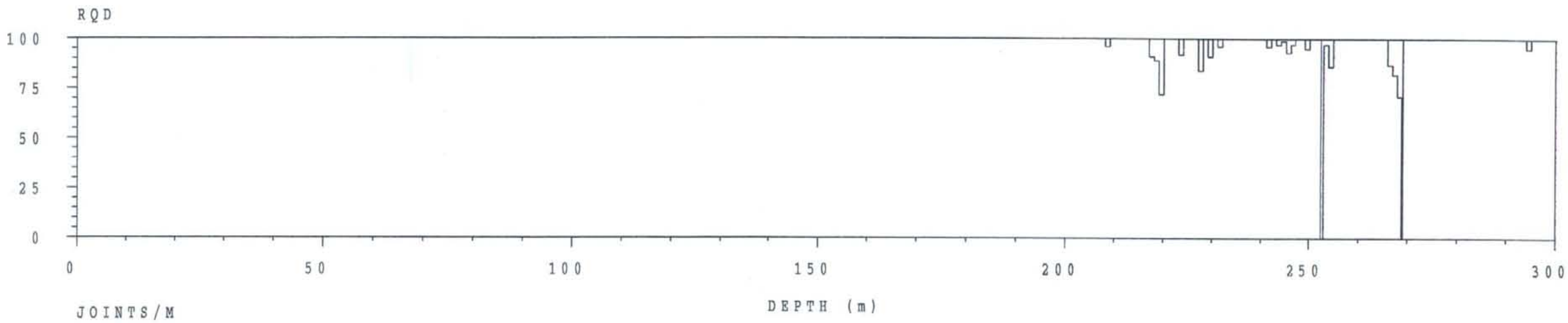

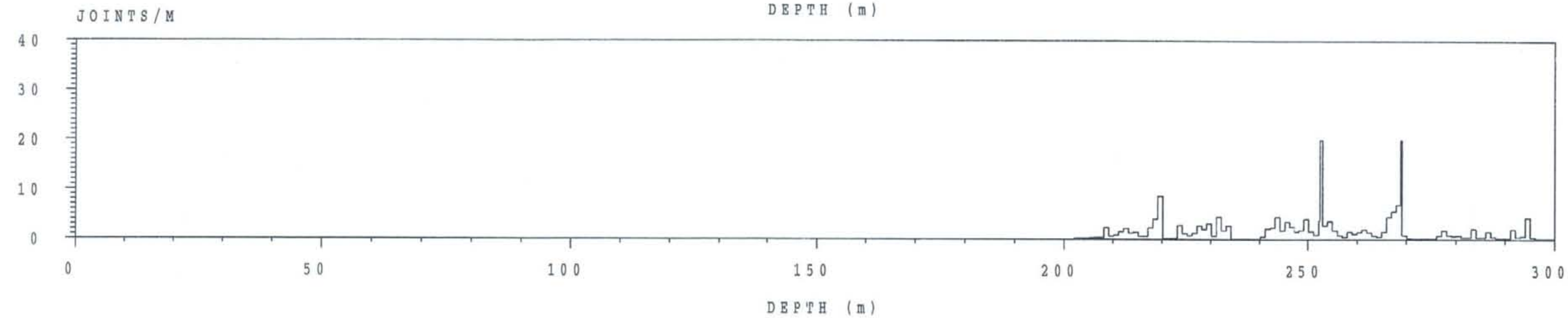

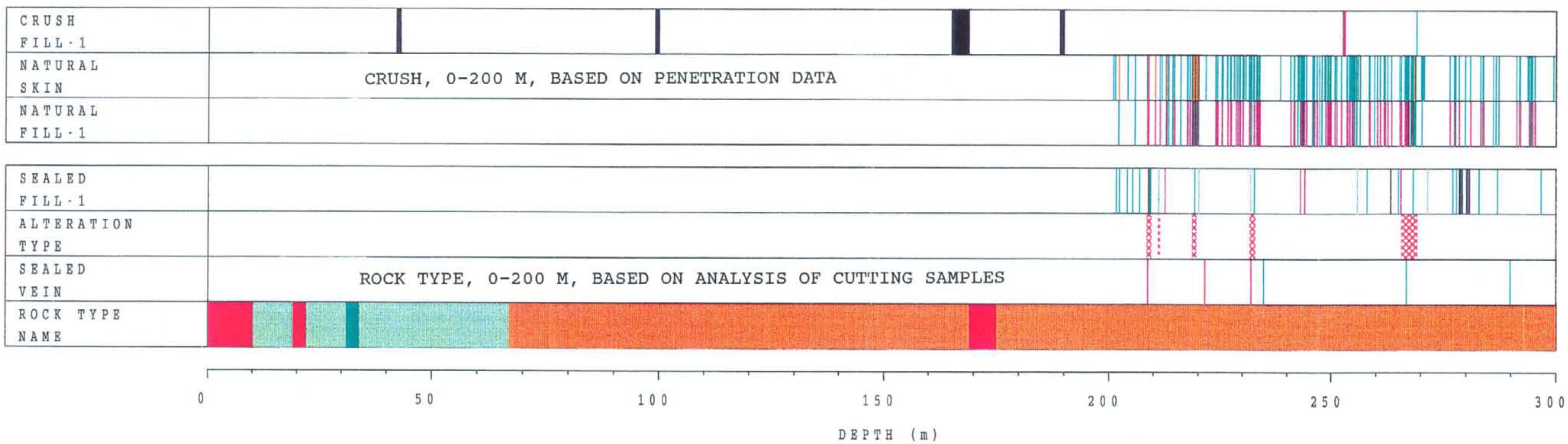

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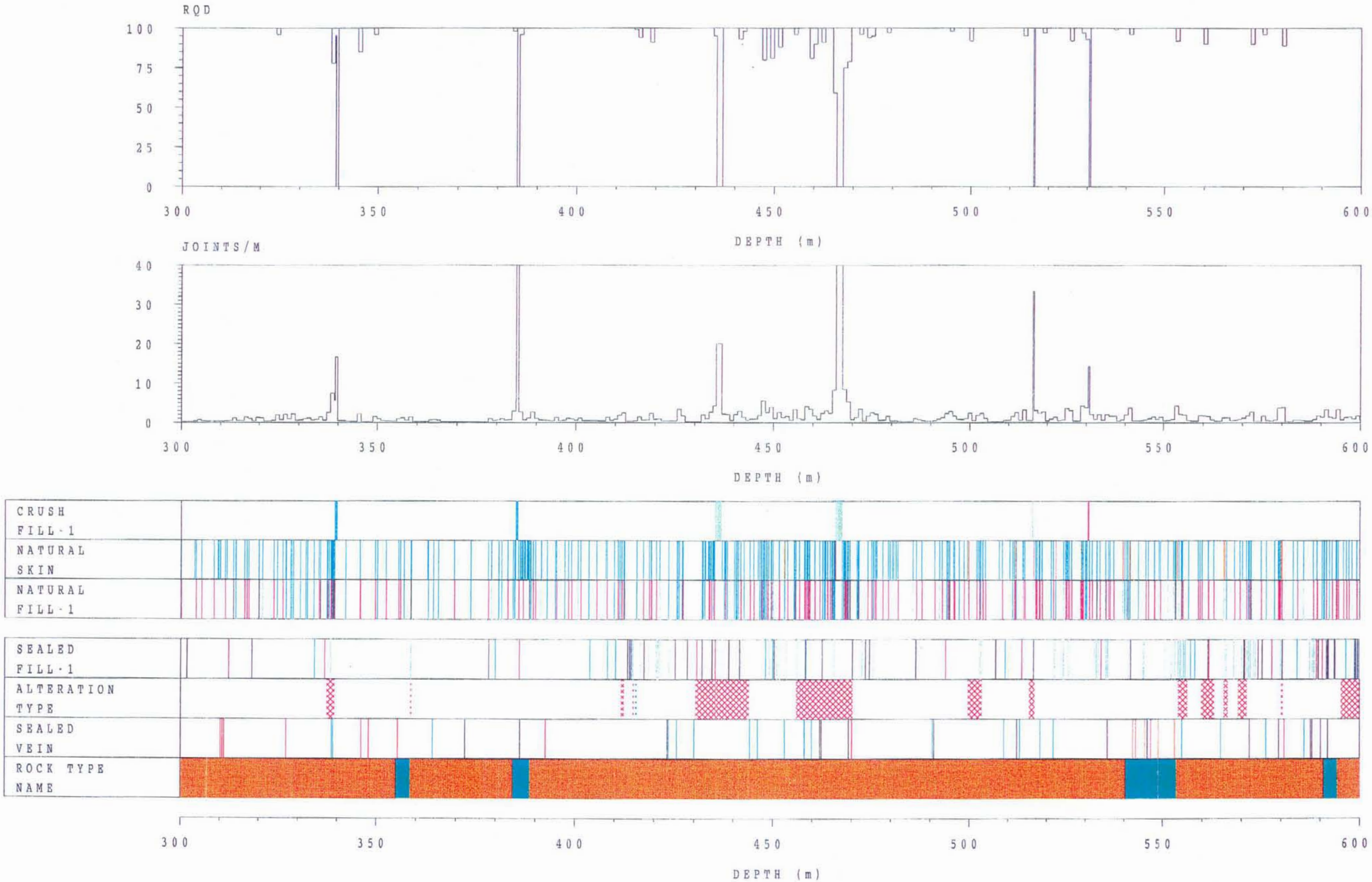

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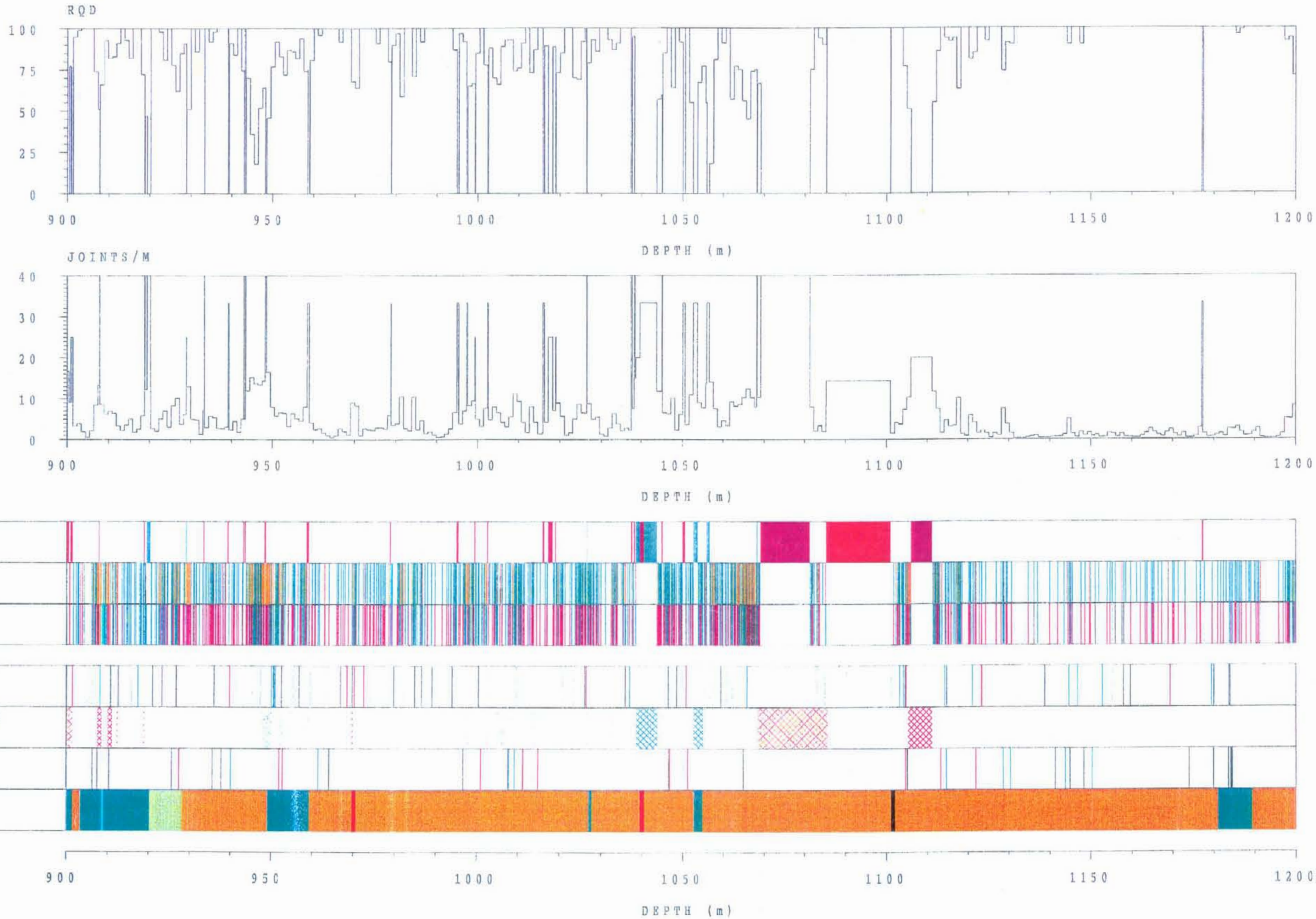

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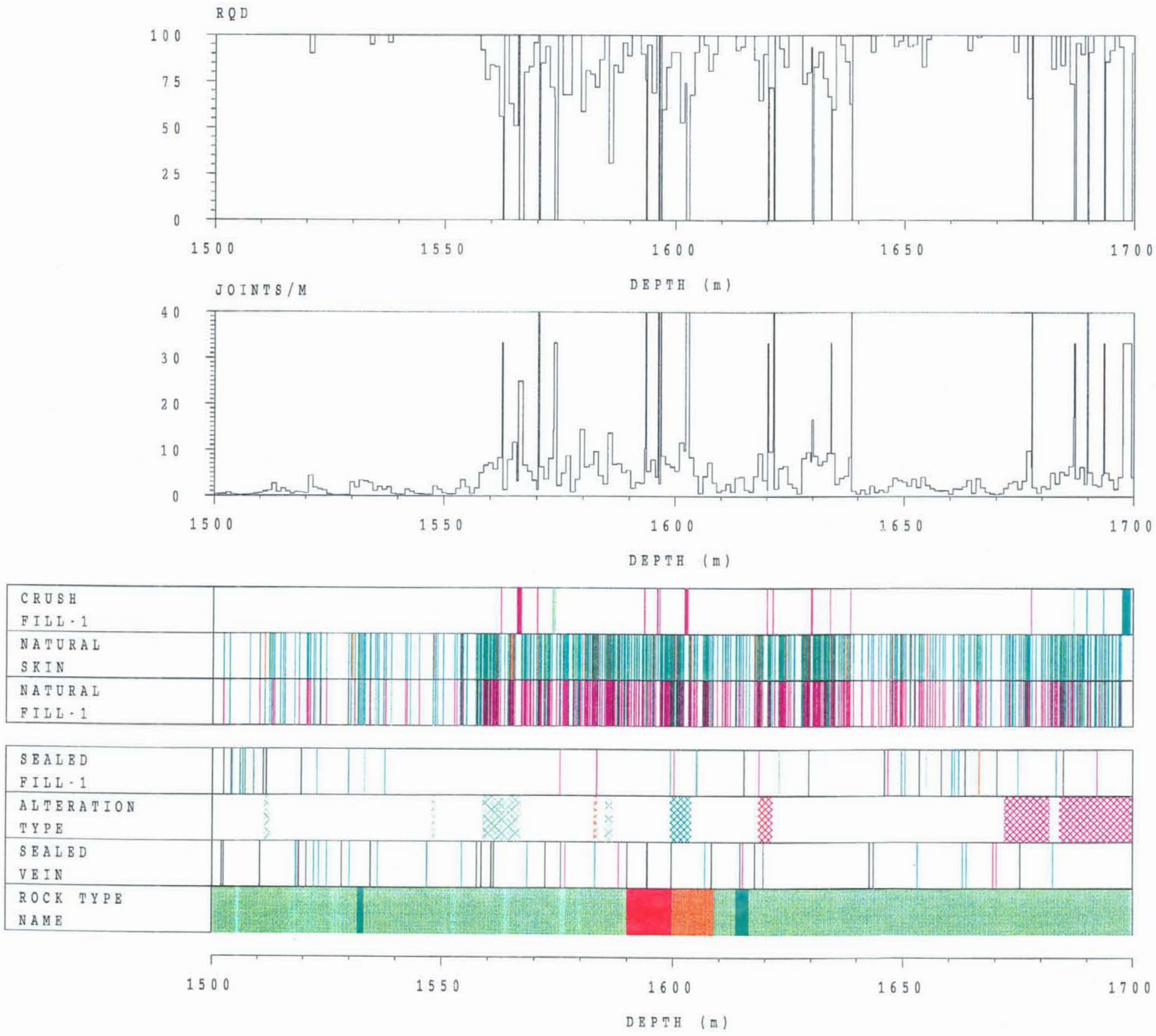

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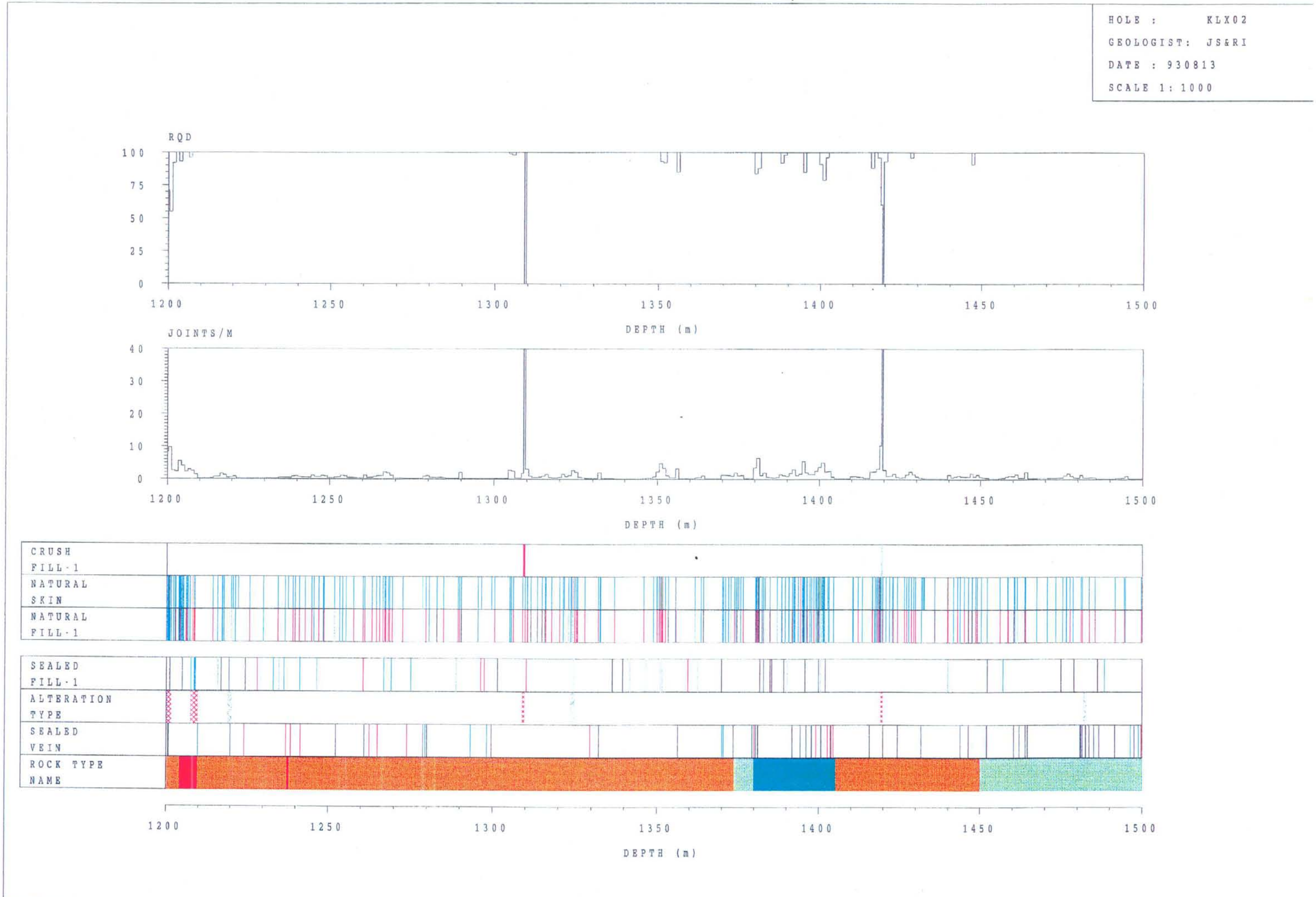

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1400
1450
1500
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JOINTS/M
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30
20
10
0
1200
1250
1300
1350
1400
1450
1500
DEPTH (m)
CRUSH
FILL-1
NATURAL
SKIN
NATURAL
FILL-1
SEALED
FILL-1
ALTERATION
TYPE
SEALED
VEIN
ROCK TYPE
NAME
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1250
1300
1350
1400
1450
1500
DEPTH (m)