URL Excavation Design, Construction and Performance

NWMO TR-2008-17

December 2008

G.W. Kuzyk and J.B. Martino

Atomic Energy of Canada Limited

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ABSTRACT

Title: **URL EXCAVATION DESIGN, CONSTRUCTION AND PERFORMANCE**
Report No.: **NWMO-TR-2008-17**
Author(s): G.W. Kuzyk and J.B. Martino
Company: Atomic Energy of Canada Limited
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Abstract

Atomic Energy of Canada Limited’s (AECL’s) Underground Research Laboratory (URL) is a well-characterized facility constructed in the Lac du Bonnet batholith in southeastern Manitoba. Characterization commenced prior to construction, providing a sound geotechnical understanding of in situ conditions for large-scale experiments in a granitic rock mass representative of intrusive igneous rock of the Canadian Shield.

Achieving a stable underground opening requires knowledge of the geological conditions, determining the design of the ground-support system, the design and implementation of a suitable excavation method and a suitable contractual arrangement with experienced excavation, inspection and quality control personnel.

Excavation is normally performed by qualified Contractors to the quality requirements set out in the agreed upon contracts. This requires a form of contract that efficiently and cost-effectively delivers the specified excavation quality and the needed characterization information within an inherently uncertain working environment. The onus is on the owner to identify an appropriate contract format and closely monitor the progress of the work to ensure the specified requirements are met. This can best be achieved by close cooperation between the owner and the Contractor.

The preferred form of contract developed at URL for shaft sinking was a cost plus fixed fee contract format, which facilitated quality excavation in a manner that encouraged cooperation between the owner and the Contractor. Horizontal tunnelling and construction work was carried out under time and material service contracts. Integration of the Contractors’ organizations facilitated close cooperation and was the key for success in these contracts.
The URL was carefully designed, constructed and monitored throughout its existence. The Observational Method developed by Peck for geotechnical projects was implemented. Detailed records comprising shift inspection, construction and geotechnical reports were kept of the construction activities and the quality of the excavations. The stability of the excavations, some being subjected to high in situ rock stresses, was demonstrated and documented over more than twenty years.

This report reviews the excavation contracts in the context of the knowledge that existed at the time of procurement, the knowledge gained from the effectiveness of those contracts in achieving the desired quality of the excavations, the specification and achievement of the excavation method standard.

TABLE OF CONTENTS

LIST OF TABLES

Page

LIST OF FIGURES

Page

LIST OF FIGURES (Appendices A and B)

Page

Glossary of Terms

Backfill – In a geological repository, the material used to fill remaining empty spaces after the waste packages and buffer have been emplaced, e.g., a mixture of glacial lake clay and crushed granite from the repository excavations.

Barrier – A feature of a disposal system, which delays or prevents radionulcides from escaping from the repository and migrating into the biosphere. A ‘natural barrier’ is a feature of the geoshpere in which the repository is located. An ‘engineered barrier’ is a feature made by or altered by man and is typically part of the waste package or part of the repository.

Batholith – A large mass of intrusive igneous rock, most of which consolidated at a considerable depth below the surface of the earth.

Blast hole - A drilled hole in rock or other medium used to emplace explosives in order to excavate or remove the rock or medium.

Blast round – An arrangement of blast holes in order to excavate a volume of rock or other medium with a predetermined cross sectional shape in order to advance an excavation.

Blasting set – A movable platform used during shaft sinking to support the installation of permanent shaft sets and equipment. The blasting set is armoured to prevent damage to the permanent sets from blasting operations on shaft bottom.

Bootleg – A remnant or portion of a blast hole on the face of an excavation remaining after detonation of a blast round, which may or may not contain un-detonated explosives.

Buffer – A barrier surrounding the waste package in a repository, consisting of highly impermeable material primarily intended to retard the movement of water.

Buffer material – A clay-based sealing material of specified properties that would comprise the buffer surrounding each used-fuel container in a repository.

Burn cut – A cluster of parallel blast holes drilled at a right angle to the face in which some of the holes are loaded with explosives and some are left empty. When the shock waves are reflected by the empty holes the rock is shattered and ejected into the empty spaces by the escaping gases.

Careful blasting – A blasting method intended to reduce vibration and overpressure – to protect instrumentation and furnishings installed close to the blasts.

Conceptual design – A technical description of a project that provides the information needed to prepare preliminary detailed designs and specifications.

Contract – A legally binding exchange or promise or agreement between parties that the can be enforced by law. Contracts have different forms, which specify payment method and how work is to be performed for what payment. In this report Stipulated Unit Price Contract refers to a contract where a pre-specified amount is paid for the work described in the contract. A service contract refers to supply of labour at determined levels (which may or may not include performance bonuses). In a service contract, equipment and supplies are purchased and billed on top of the sum negotiated for labour. See also cost (of work) plus fixed fee contract below.

Contractor – A person or organization retained to provide service and/or materials to the owner under a contract.

Controlled drill and blast – A specialized blasting method designed to minimize fracturing or damage to the final walls of an excavation.

Core logging – The analysis and recording of the characteristics of rock and overburden samples retrieved from the formations through which a borehole passes.

Cost (of work) plus fixed fee contract – A form of contract where the Contractor is award a certain fee for the performance of the work outlined in the contract plus allowances for additional costs incurred during performance of the work as specified in the contract. The fee portion covered items that were fixed and easy to estimate, such as the Contractor's overhead and specified equipment and services.

Crosshead – A framework that runs on the shaft guides used to secure the sinking bucket to prevent it from swinging in the shaft during hoisting.

Cryderman mucker – A clam-shell type loader activated by hydraulic cylinders operated from a shaft stage used to excavate broken rock after blasting operations at shaft bottom.

Decoupled charges – An explosive charge that is separated from the walls of the blast hole in which it is placed, thereby created an air-gap between the explosive charge and the rock surface of the blast hole.

Dependent charge – An explosive charge in a blast hole the performance of which is dependent on one or more preceding detonating charges in blast holes creating a void for breakage and displacement of the broken rock created by the dependent-charge blast hole.

Diamond drill core – A rock sample obtained from a drill hole by drilling with a hollow diamond bit and drill string that contains a core barrel to collect the sample.

Disposal containers – A durable receptacle for enclosing and isolating radioactive wastes for disposal.

Drill jumbo – A drill platform used in tunnelling on which one or more drills are mounted.

Excavation damage zone – A zone in the perimeter wall around an excavation characterized by permanent and measurable changes in both the pre-excavation stress field and the material behaviour of the rock mass. The extent of the excavation damage zone is influenced by the excavation method and stress re-distribution from the presence of the excavation. It is surrounded by an excavation disturbed zone where changes to rock mass properties are not permanent.

Excavation method – A means of creating an opening in rock (e.g., drilling and blast excavation, tunnel boring).

Galloway stage – A multi-decked work platform that is suspended below the permanent shaft sets near the shaft bottom during sinking. It carries part of the equipment used for sinking operations and can be raised or lowered by hoists as required during drilling, blasting and mucking, the installation of shaft wall ground support, shaft wall lining, and shaft sets and conveyance guides.

Gear truck – A platform on a rail mounted chassis used to facilitate the drilling and loading of the upper portion of a tunnel blast round.

Gneiss – A coarse-grained metamorphic rock in which bands, rich in granular minerals, alternate with bands in which flaky minerals predominate.

Granite – A coarse-grained igneous rock consisting mostly of quartz (20-40%), alkali feldspar and mica. A number of accessory minerals may be present.

Ground control – The process of installing of rock bolts, screen mesh, strapping and/or shotcrete to provide for the protection of personnel and equipment from loose and/or spalling rock in underground excavations.

Ground support system – Mechanical devices designed to reinforce, support and/or strengthen the rock mass surrounding excavations to make it more stable. Ground support systems can include rock bolts, screen, shotcrete, cable bolts, and tunnel sets or combinations of these.

Grout – A fluid mixture of cement and water, or a mixture of cement, sand and water used to seal boreholes and fractures in the rock mass or to seal surfaces and structures.

Half-barrel – Terminology used to identify the remnant or trace of a blast hole left on the walls of the excavation after blasting. The percentage of half-barrels is a measure of the total length of blast holes traces remaining compared to the total length drilled. This is a visual measure of effectiveness of the controlled drilling and blasting results, as drilling alignment and blast hole charges have to be precisely controlled to achieve breakage while still leaving a blast hole remnant or trace. Ideally, the remnant or trace is in the form of a “half-barrel” or “half a cylinder”. Actual distinction between a half-barrel and blast hole trace is often left to the discretion of the individuals carrying out the evaluation.

Heading – An excavated tunnel, ramp, shaft or raise.

Hydraulic fracturing – A process in which fluid is injected under pressure into a geological stratum from a borehole to induce artificial cracking. It can be used to measure ambient stresses in a rock body in a plane perpendicular to the borehole axis. Also known as ‘hydro-fracturing’.

Hydrogeology – The study of geological factors relating to the earth’s water.

Instrument arrays – Instruments mounted in boreholes drilled radially in arrays or rings around a tunnel or excavation to measure changes in rock mass characteristics at the instrument locations as the excavation proceeds.

Joint venture – An agreement by two or more individuals or organizations to undertake a specified body of work.

Level – A main heading or passage excavated horizontally from a shaft or access ramp to provide access and services from the shaft or access ramp to underground excavations connected to the main heading or passage.

Muck – Broken rock from excavation operations.

Nuclide – A species of atom characterized by its mass number, atomic number and energy status.

Observational method – A methodology that emphasizes adaptation of plans to meet an anticipated range of conditions encountered in the development of geotechnical projects, with regards to excavation design, excavation method and rock support.

Owner – The individual or organization that issues a contract and has legal and financial possession and/or responsibility for a site of the facility being constructed.

Permeability – The capacity of a porous rock, sediment or soil to transmit a fluid.

Pilot and slash – A method of excavating a heading whereby a small pilot opening is first excavated then followed by one or more slash excavations to reach the final perimeter profile.

Powder factor –The mass of explosives per unit volume of solid rock broken in a blast. It is a measure of the blast efficiency for a given explosive and rock mass.

Progress monitoring – Observations by the owner or representatives of the owner to ensure the contract requirements are being met as specified in the contract.

Qualified contractors – Those Contractors considered to have the necessary experience, abilities, and financial qualifications to successfully bid and perform the awarded work on a given project.

Quality assurance – Planned and systematic actions necessary to provide adequate confidence that an item, process, facility or person will perform satisfactorily in service.

Quality control – Actions that provide a means to fix and measure the characteristics of an item, process, facility or person in accordance with quality assurance requirements.

Raise – A vertical or sub-vertical heading driven up from an excavation or opening below. It can also be applied to “raise bore” excavations where a cutting bit is drawn from below by a rig at the top location of the raise through a pilot hole.

Ramp – An inclined heading driven down from surface, or up or down from another excavated opening.

Repository – An underground structure excavated in rock where containers of used nuclear fuel would be sealed to permanently isolate them from the natural environment.

Rope anchors – Anchoring devices constructed of steel plates, rock bolts and concrete used to anchor tail ropes to the shaft wall.

Scooptram – A low-profile load-haul-dump vehicle used in underground tunnelling and mining operations to move broken rock or muck.

Shaft – A vertical heading driven down from surface and used for transferring personnel, equipment and materials, for ventilation, and for transporting broken rock.

Sinking bucket – A device suspended from a hoisting device and used to transport men or materials during excavation (sinking) of a shaft.

Sub-contractor – A person or organization hired or contracted by a Contractor.

Sublevel – A secondary passage or access tunnel excavated horizontally from a raise or ramp to provide intermediate access to the rock mass between main levels.

Suspension ropes (ropes or cable) – Steel hoist cables used to suspend and move a conveyance in a shaft (e.g., a sinking stage, Galloway stage, shaft bucket or cage) during shaft sinking operations.

Tail ropes – The section of steel hoisting rope that passes round the return sheave on a sinking stage or Galloway stage suspended in the shaft.

Tender for/of Contract – The call for bids (i.e., tenders) to be submitted by Contractors on the type of contract developed by the owner for a defined scope of work. The owner's requirements for quality assurance are specified in the tender package.

Tugger – A compressed air driven utility hoist used underground for lifting or sliding heavy loads.

Tunnel – A horizontal or slightly inclined heading driven from surface or from another excavated opening.

Waste package – The waste form in an appropriate container(s) suitable for handling, transporting, interim storage and placement in a repository.

1. INTRODUCTION

Atomic Energy of Canada Limited (AECL) has gained extensive experience in the design, construction and performance of stable underground excavations in the construction and operation of the Underground Research Laboratory (URL). This experience is outlined in terms of the types of contracts developed for use in excavation, specifications required to meet the desired excavation schedule and performance, and finally the performance of the excavations in terms of stability. The experience from conducting and operating the URL shows that underground openings that will remain stable for decades can be designed, and constructed, even in high-stress rock conditions.

AECL led the design and construction of the URL in southeastern Manitoba, as part of Canada's Deep Geological Repository Research and Development (R&D) Program (Simmons and Soonawala 1982). The URL was constructed in several phases and different contracting approaches were taken in each phase. These phases include the:

- excavation of the upper shaft;
- excavation of the lower shaft or shaft extension;
- 240 Level excavation (240 m depth below surface);
- 420 Level excavation (420 m depth below surface);
- boring of the upper ventilation raise; and
- boring of the lower ventilation raise.

The purpose of this report is to describe the:

- objectives of each excavation phase;
- state of knowledge of the geosphere at the time of each excavation phase;
- design, construction method selection and inspection processes;
- contractual and risk-sharing arrangements; and
- short- and long-term post-excavation stability of the openings.

1.1 BACKGROUND

The Government of Canada accepted (NRCan 2007) the Nuclear Waste Management Organization's (NWMO) recommendation of Adaptive Phased Management (NWMO 2005) as the long-term management approach for Canada's used nuclear fuel. Adaptive Phased Management includes: centralized containment and isolation of the used fuel in a deep geological repository in a suitable rock formation, such as crystalline rock or sedimentary rock; monitoring of the used fuel repository to support data collection and confirmation of the safety and performance of the repository; and the potential for retrieval of the used fuel for an extended period, until such time as a future society makes a decision on the final closure and the appropriate form and duration of postclosure monitoring.

Research and development (R&D) has been underway in Canada since 1978 to support the assessment of the most appropriate approach for the management of Canada's used nuclear fuel. One aspect of this R&D is the development of the required technologies for implementation of a deep geological repository, which is a component of the NWMO's Adaptive Phased Management approach. The R&D program for the deep geological repository for used nuclear fuel has been sequentially managed by Atomic Energy of Canada Limited (AECL), Ontario Hydro (OH) and Ontario Power Generation Inc. (OPG) and now the NWMO.

1.1.1 Functional Life and Characterization Needs

OPG prepared Preliminary Requirements for a Deep Geologic Repository for Used Nuclear Fuel (Simmons 2006) that includes the following performance requirement. The structures, systems and equipment at the deep geologic repository shall have a functional life based on normal maintenance, replacement and refurbishment as follows.

- All structures, systems and equipment shall have a functional life that extends from the time of their construction or installation until the emplacement of all used fuel and other radioactive wastes have been completed. The preliminary requirement is 40 years, based on allowing 10 years for deep geologic waste management facility construction and 30 years for waste emplacement operations. If a structure, system or equipment is used for activities prior to the construction of the deep geologic repository, then the time of this service shall be added to the functional life specified in this preliminary requirement.
- All structures, systems and equipment necessary for post-operational access and maintenance shall have a functional life that extends to the completion of these activities. The preliminary requirement is an additional 10 years beyond the operational (emplacement) period.
- All structures, systems and equipment that are necessary for retrieval of used fuel prior to repository closure, and for the decommissioning and closure of the deep geologic repository, shall have a functional life that extends from the start to the completion of these activities. The preliminary requirement is an additional 25 years beyond the operational (emplacement) period.

This performance requirement establishes a long period of access to the deep geological repository and implies that the excavations and installations must be stable and safe for the duration of these functional lifetimes. This requires both controlled excavation and the application and on-going maintenance of ground-support systems. A ground support system comprises mechanical devices designed to reinforce, support and/or strengthen the rock masses surrounding excavations to make them more stable. These devices can include mechanical bolts and cables grouted in place, pre-cast concrete or shotcrete linings and pre-formed mechanical sets with interstitial lagging to increase strength and competence. Grouted anchor bolts and cables can be tensioned to provide immediate reactive force. It may also be necessary to install ground control systems, such as mechanical anchor bolts, wire mesh screen, strapping and/or shotcrete to provide protection for personnel and equipment from loose or spalling rock. Ground control systems provide limited reactive force or support to the rock mass and are generally used in areas where the rock mass is self supporting and competent.

One important requirement for siting and construction of a deep geologic repository is the detailed physical and chemical characterization of the host rock prior to, concurrent with and following the excavation process. These characterization activities must be integrated with the construction activities and are an important component in their planning, contracting and execution. This methodology was applied during the URL construction.

1.1.2 Use of Underground Laboratories

Many national waste management programs perform aspects of their R&D activities in an underground laboratory (UL) to maximize their understanding of operative processes and to ensure that their plans are as safe and efficient as practicable for implementing their deep geological repository. Pusch and Svemar (2004) identified the purposes of underground laboratories as contributing to:

a. development of repository system concepts;
b. confirmation of a preferred repository system concept;
c. qualification of repository system design; and
d. qualification of a repository site.

Purposes (a) and (b) can be achieved in either a generic underground laboratory (i.e., away from the potential site of a repository but in similar physical or chemical conditions) or in an on-site laboratory (i.e., on a potential repository site). Purpose (c) would likely require access to an on-site UL and purpose (d) would definitely require access to an on-site UL.

The construction of an UL provides an opportunity to develop and refine the excavation methods, including ground support, the characterization activities and the integration of these into a suitable relationship with a construction Contractor. The access and underground openings in an UL may require very careful excavation and characterization processes to provide a safe working environment and to clearly establish the appropriate boundary conditions for future scientific and engineering studies. The functional life of a UL will depend on whether it is generic or on-site. The functional life of a generic UL will be based on the need for the facility and the functional life of an on-site UL might be related to the functional life of the repository.

The UL also provides a demonstration of the stability of underground openings over long time periods (e.g., for several years longer than a repository if a repository is later developed at the site being investigated). Records should be kept on the condition of underground openings, such as scaling records, ground control and support maintenance. These provide evidence of the stability of openings to show that long life is achievable. Preservation of such records from excavations in a UL or repository will be critical in providing an evidence base for the long-term stability of deep geologic repository excavations.

The generic ULs, such as AECL's Underground Research Laboratory and SKB's Äspö Hard Rock Laboratory, provided the dedicated facilities to perform integrated in situ scientific and engineering research investigations into the a large number of properties and processes including rock properties, evolution of groundwater chemistry with time, geochemical interactions between groundwater and engineered barriers, transport of groundwater through sparsely, moderately and highly fractured rock, geochemical and transport behaviour of dissolved and colloidal radioisotopes in geological environments and diffusion into the interconnected pore spaces of the rock matrix and the excavation damage zone.

The design, performance and safety assessment of the deep geologic repository will require in-depth understanding of the in-situ environment, the temperature and time-dependent deformation characteristics and failure behaviour of the host rock, techniques to minimize the excavation damage, minimize the cost and ensure safe working conditions during construction and the adsorption properties of the rock fracture and fracture in-filling material around the excavations. The generic ULs have been key tools in the development and demonstration of the sciences and technologies for long-term used-fuel management including the testing and confirmation of site characterization, design, construction, operation, decommissioning and closure tools and methods.

1.1.3 Activity Integration

Contractors experienced in underground civil and mine excavation will construct the generic or on-site ULs and repositories once environmental review, regulatory approval and site characterization are completed. The challenge for the facility owner will be to establish a contractual environment that will safely and efficiently provide the required excavation quality and project flexibility and the needed characterization information in a naturally variable geological environment. A mutually agreeable contract wording must be developed for dealing with the expected range of natural variations and any unexpected situations that may arise.

The cost and schedule commitments must be consistent with the end-product requirements. The onus is on the facility owner or his agent to clearly define the requirements and end products in the tender documents, which includes the required excavation quality and the need to gather characterization information, especially that to be gathered concurrent with the excavation process and before the installation of permanent structures (e.g., excavation liners, utilities, ground support consistent with safety requirements). Unless contracts are prepared with these issues in mind and with an appropriate sharing of risk between the Contractors and the facility owner, either the desired excavation quality may not be achieved, characterization information may be lost or the project resources may be at risk. Contractual arrangements between the owner and each Contractor must clearly define the sharing of risk, offer incentives for good performance and provide the flexibility to react appropriately to changing conditions and circumstances while achieving the project objectives. Development of a good working relationship between the principal Contractor and owner’s repository staff is a key factor in the smooth operation of the contract.

Following construction, more information must be gathered on the response and performance of the underground excavation structures and environment throughout the facility’s lifecycle. The owner has an ongoing commitment to ensure the needs are defined and that this information is gathered and maintained over long time periods.

1.2 LOCATION AND SETTING OF THE URL

The URL is located near the town of Lac du Bonnet, Manitoba, about 110 km northeast of Winnipeg. The shaft was excavated to a depth of 443 m in the Lac du Bonnet granitic batholith, which is considered representative of many igneous rock intrusions throughout the Precambrian Canadian Shield. Areas of research, development and demonstration at the URL have included:

- surface and underground characterization,
- solute transport studies, groundwater geochemistry and microbiology,
- analysis of temperature- and time-dependent deformation and failure characteristics of rock,

- excavation damage assessment and excavation stability, including the development of controlled drilling and blasting, and
- development and performance assessment of clay- and concrete-based sealing material.

Twenty nine major experiments or experimental programs have been carried out at the URL (Chandler 2003). These are listed under four broad experimental categories in Table 1. Numerous smaller experiments were also conducted.

Table 1: URL Program of Experiments

Solute Transport (ST) • ST in Highly Fractured Rock (HFR) • ST in Moderately Fracture Rock (MFR) • Quarried Block Migration Experiment (QMBE) • In Situ Diffusion Experiment • Excavation Damage Zone Connected Permeability Test • Japan Atomic Energy Research Institute Rockmass Experiment • Recharge Infilitration Experiment (RIEX) Materials and Sealing Studies • Buffer/Container Experiment • Isothermal Buffer-Rock-Concrete Plug Interaction Test (ITT) • Fracture Zone Grouting Experiment • High Pressure Grouting Simulator • Large Concrete Blocks • Seal and Interface Evaluation/Effect of Salinity (SEAS) • Light Backfill Placement Trials • Buffer Coupon Long-Term Test (BCLT) • Concrete Rock Interface Study (CRIS) • Dedicated microbial borehole and microbial studies	Excavation Damage/Excavation Stability • Stress Characterization Program • Room 209 Excavation Response Test • Engineered Blast Feasibility Study • Mine by Excavation Response Test • Heated Failure Tests (HFT) • Blast Damage Assessment Study (BDA) • Mine-by Connected Permeability Test • Excavation Stability Study (ESS) • Thermal-Hydraulic Experiment (THE) Multi Disciplinary • Characterization program • Tunnel Sealing Experiment (TSX) • Composite Sealing Experiment (CSE)

Most of the underground openings at the URL were excavated using controlled drill and blast methodology. The method is flexible and cost effective in hard, igneous rock and could be used to excavate a deep geological repository. Controlled drilling and blasting methods that minimize damage to the final excavation perimeter would facilitate the construction and performance of the seals and bulkheads within a repository. During the construction of the URL controlled drilling and blasting methods were integrated into the blast designs to demonstrate that shafts and tunnels can be excavated with minimal damage in competent highly stressed rock.

1.2.1 Geologic Setting of the URL

At the URL site, there are several fracture zones around thrust faults located as shown in Figure 1. According to the geological characterization carried out before and during construction, the number of fractures decreases with depth (Everitt et al. 1990). The in situ stress distribution is strongly influenced by the fracture zones. Before the excavation of the URL shaft, in situ stress was determined from 80 hydraulic fracturing tests (Doe 1987, Thompson and Chandler 2001 and Thompson et al. 2002). More than 1,000 overcore tests were also performed to check the in situ stress distribution during and after the excavation of the URL (Martino et al. 1998). As shown in Figure 2, it was found that the horizontal stress magnitude increases almost linearly down to about 270 m in the near-surface region of moderately fractured rock, and the URL in situ stress magnitudes in the moderately fractured rock are similar to those measured elsewhere in the Canadian Shield. There is a sudden large increase in horizontal stress as excavation depth increases below the deepest fracture zone intercepted by URL excavations (FZ2 in Figure 2) and into the deeper region of massive sparsely-fractured rock.

The maximum horizontal stress at deeper locations in the URL is approximately 60 MPa, which is almost six times higher than the vertical stress at the 420 Level. Under these high stress magnitude and ratio conditions, both hydraulic fracturing and overcore stress determination methods were problematic and excavation-scale measurements, such as convergence and under-excavation, were required to define the 60 MPa in situ stress magnitude (Martino et al. 1998).

The strength of Lac du Bonnet granite has been characterized as follows.

- The short-term, laboratory-scale unconfined uniaxial strength (σc) of intact granite is about 180 to 200 MPa (Jackson 1989).
- The estimated long-term in-situ unconfined uniaxial strength in massive, unfractured granite is about 150 MPa (Martin 1993(a)).
- The in-situ microcrack-initiation strength in massive, unfractured granite is approximately 100 MPa (i.e., ~50% of σ_c) (Martin 1993(a)) but is influenced by local geologic fabric.

Highly saline pore water with salinity about three times higher than seawater is found in the rock matrix at the 420 Level (Gascoyne 2004). Several characterization studies that examined the geological, geochemical and geomechanical characteristics of the URL site were developed and performed in the following five regions, all these studies involved numerical modeling components.

- <u>Five km^2 of exposed granite outcrop on the surface</u>: A hydrogeologic network of over 130 shallow and deep boreholes was established to monitor the variations in hydraulic head caused by construction of the URL.
- <u>Highly fractured rock:</u> Extensive characterization was performed to determine the hydraulic and solute transport properties of the fracture zones that cross the URL site,

and to test the ability to predict the transport of solutes through the geosphere using available numerical tools. The work included studies of the mechanical-hydraulic coupling that was apparent in the measured responses.

- Moderately fractured rock: A comprehensive study of solute transport through rock having between one and five fractures per linear metre was performed at the URL. There was also a study of the effect of excavation on the hydraulic and mechanical response in the moderately fractured rock near the surface.
- Low stressed, sparsely fractured rock: Experiments in this region studied the performance of engineered sealing materials in the absence of rock damage caused by in situ stress. Studies were also directed at improving excavation methods such as controlled drilling and blasting. The existence of a single water-bearing fracture in this region allowed the study of the hydro-mechanical response of the fracture to excavation and the study of radionuclide transport in an isolated block taken from the fracture.
- Highly stressed, sparsely fractured rock: The rock in this region is stressed to the point of fracturing in the roof and floor of the excavations. In some cases, the fracturing can result in small, but continuous regions of highly damaged rock. Experiments in this region studied rock response to excavation, including designing excavations to minimize stress-induced damage. Experiments were conducted to study technologies for constructing seals in excavations with zones of damaged rock. The high pore water salinity in this region also provided an environment for testing the effect of salinity on engineered barrier components. Diffusion tests were also conducted in the region.

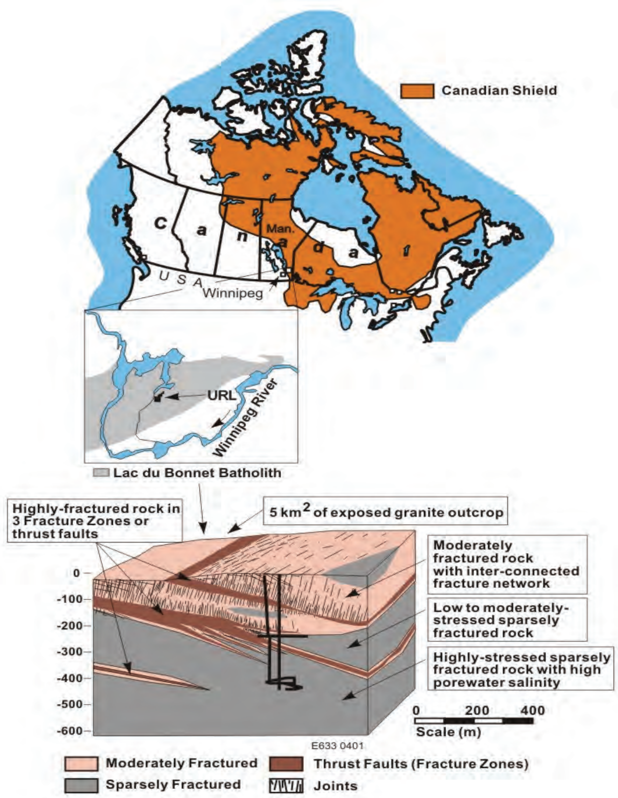


Figure 1: The Location of the URL Within the Canadian Shield (upper image) and the Geologic Setting of the URL (lower image)

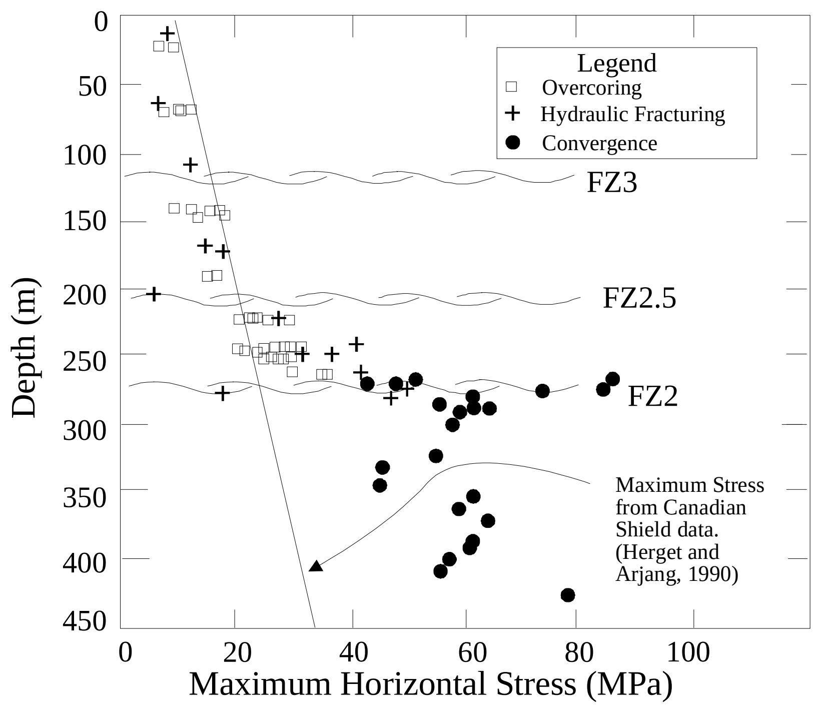


Figure 2: Maximum Horizontal In Situ Stress with Depth at the URL

2. URL EXCAVATION

The main underground excavations at the URL include a 443-m-deep shaft, major developments at the 240 and 420 Levels, shaft stations at the 130 and 300 Levels and a 1.8-m-diameter bored ventilation raise. Figure 3 shows an isometric view of the underground excavations at the URL.

The URL program can be divided into the following four stages:

1. Site Evaluation Stage
   - Installation of monitoring system: 1980 - 1984
   - Monitoring: 1981 - present
2. Construction Stage
   - Surface building: 1982 -1987
   - Underground access: 1983 -1990
3. Operating Stage: 1989- 2002
4. Decommissioning Stage: 2003 - 2010

This report describes the contracts used for development in Stage 1 and 2.

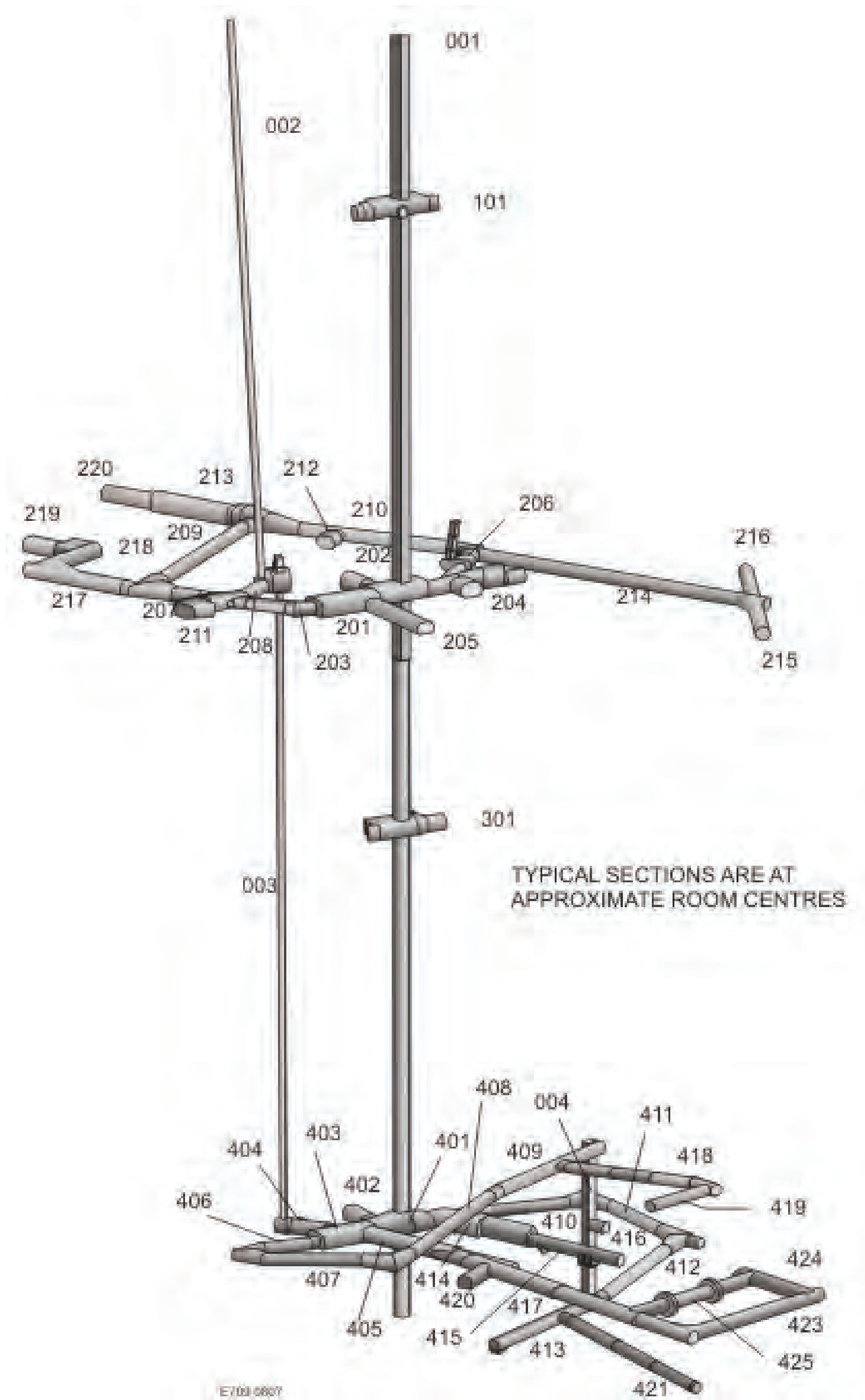


Figure 3: Isometric View of URL Showing Excavations and Room Numbers

Shaft collar excavation and construction of the surface facilities took place between 1983 and 1984. Excavation of the URL shaft to a depth of 255 m began on 1984 May 12 and continued for the remainder of the year. The loop of horizontal excavations on the 240 Level and the raise-bored ventilation shaft from the 240 Level to surface were completed by 1987. The main shaft was extended to a depth of 443 m in 1988, followed by the excavation of the 420 Level and the raise-bored ventilation shaft to the 240 Level over the following three years.

Considerable time and effort was put into the preparation of relevant design specifications, development of appropriate contract formats and ongoing project management (Peters et al. 1990). Innovative equipment was utilized for construction activities and quality control procedures were implemented to ensure that data were reliable and made available to the research program during every step of construction. Research and construction activities were fully integrated to allow simultaneous collection of geotechnical information during excavation (Kuzyk et al. 1986a).

The shaft was sunk with a drill and blast method in two phases utilizing a benching method of advance in the upper rectangular section to 255-m depth and a full-face method in the lower circular section (Kuzyk et al. 1990). During the first phase of sinking, it was found that the benching method did not adapt well to the controlled drilling and blasting method. The rectangular section proved to be a poor geometry because it resulted in stress concentrations and unacceptable damage at the corners.

From a geotechnical characterization perspective, the orientation of the rectangular cross-section was not aligned with the in situ stress field making interpretation of rock displacement measurements difficult. This was exacerbated by the benching method itself as the shaft floor was not level at the shaft instrument arrays, which complicated the interpretation of data from instrument arrays installed to monitor rock mass response to excavation.

To improve on the quality of blasting and facilitate the analysis of rock displacement measurements, the shaft extension was excavated using a circular cross-section and a full-face method of advance. Controlled drilling and blasting principles were applied to the full-face blast designs. This change resulted in much less damage to the walls of the shaft and improved stability (Hagan et al. 1989).

In this report greater emphasis is given to the lower shaft contract due to its greater suitability for the development of a UL or repository. Important lessons were learned during the excavation of the upper shaft, with respect to concurrent excavation and geotechnical characterization activities during the upper shaft sinking. These lessons allowed the more successful contract used for the lower shaft excavation to be developed. To put those lessons in context it must be remembered that the integration of excavation and R&D activities was a new construction approach at the time.

The ventilation raises (upper – surface to 240 Level and lower – 240 Level to 420 Level, Rooms 002 and 003 in Figure 3) provided ventilation exhaust and a second means of egress from both levels. The raises were bored with a raise bore machine that represented a different method for excavation (i.e., mechanical excavation with a fixed circular cross section and cutter head configuration designed for a specific rock characteristics) that provided inherently smoother surfaces (i.e., without discontinuities to concentrate stress). The raise bore method improved stability and reduced the depth of the excavation damage zone (EDZ). Remote operation of the raise boring enhanced safety but prevented the installation of ground support until after each raise had been reamed to final dimensions. This delay in the opportunity to install ground-support might be detrimental in the case of weak or highly stressed yielding rock, which can lead to progressive failure without the ability to immediately respond to the situation.

A two to three metre section of the 420 Level to 240 Level ventilation raise located at FZ2 about 30 m below the collar of the raise in Room 208 (240 Level), developed a breakout notch from stress re-distribution. This notch remained stable until the location was washed for photography. The spalling then reactivated and shotcrete was applied to maintain safety. Basically, the ventilation raises remained stable requiring very little maintenance and no ground control or support, with exception to that mentioned above, throughout the operating stage of the URL.

Because no characterization was planned during the raise boring, a stipulated price contract was suitable for this work. However because the presence of Fracture Zone 2 was known, alternative clauses were built into the contract for the 420 Level ventilation raise to allow work to proceed using a cost plus format if difficulties were encountered with water inflows raise-boring through FZ2. In situations where rock quality may be lower (poor ground or jointed rock mass) or have potentially higher inflow rates, contract flexibility is important to ensure work is done with a proper sharing of risk between the owner and the Contractor. While this was ultimately not required in the raise boring work at the URL, the flexibility to accommodate such uncertainties is important.

2.1 UPPER SHAFT (Surface to 255 m)

The rectangular section of the shaft was excavated from the surface to the depth of 255 m and has dimensions of 2.8 m x 4.9 m. The bottom few metres were later altered in developing the transition to the circular shaft cross section. Figure 4 shows both the bench and full-face blasting methods used for shaft sinking at the URL. The full-face blasting method used in the lower shaft extension (255 m to 443 m) is discussed in detail in Section 2.2.

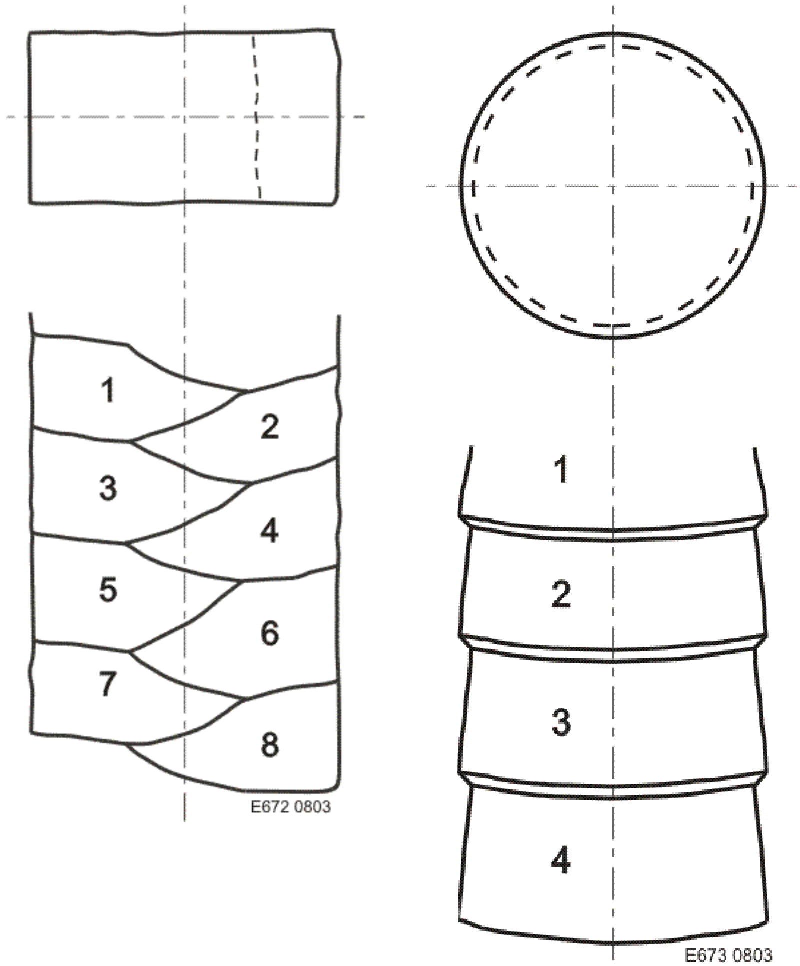
2.1.1 Shaft Collar Construction Contract

The shaft-collar construction contract (Contract No. 2 – Shaft Collar Excavation and Foundation Work) was tendered separately from the rest of the upper shaft. The shaft-collar refers to the top 15-m portion of a shaft, where access is gained from the surface. This permitted the preparation of the shaft collar for the shaft-sinking phase, including installation of foundations for the head frame, and the assessment of the trial drill-and-blast rounds as an input to the tendering documents.

The shaft collar excavation and head frame foundation was included in one contract, which involved:

- excavation of the shaft collar to a depth of 8 m using a pre-shear blast design;
- temporarily halting shaft-collar excavation after the initial 8 m for a three-week period, which permitted unrestricted shaft-collar access by experimental staff and continuance of foundation installation;
- completion of shaft-collar excavation to a depth of 15 m;
- provision of temporary ladder access to 8 m, removal for excavation and reinstallation of ladder access to 15 m;

- installation of services (compressed air, water, electrical) to 8 m, removal for excavation and reinstallation of services to 15 m;
- temporary material-handling facilities from surface to shaft-collar bottom to 8 m and then to 15 m if required for an additional price; and
- provision and installation of temporary ground support.



(a) Benching Method (b) Full-Face Method

Figure 4: Shaft Excavation Methods Used at the URL

This excavation contract defined the type of excavation method required to start sinking the shaft. It was considered and bid as a straightforward construction contract for a fixed (or stipulated) amount for the work, based on information provided by the owner and the conditions set out in the contract. While this contract form proved suitable for the scope of work defined for shaft collar excavation, it lacked the flexibility to accommodate characterization activities and owner-imposed delays when applied to advancing the upper shaft excavation (Section 2.1.2)

To help assess the Contractor's capabilities the following information was required as part of their tender.

- Construction data relating to the percentage of work performed by joint venture partners and subcontractors.
- The names of subcontractors working under the Contractor's supervision.
- Organization chart for the staff working on the contract.
- Estimated hours of work.
- Manpower tabulation with classification of personnel, including any subcontractor's workforce.
- Labour conditions, collective agreements and workforce accommodations,
- Insurance summary.
- Safety program and arrangements for first aid.
- Construction schedule.
- Plant and equipment schedule.
- Construction methods.
- Schedule of payments.
- Schedule of unit prices for labour, equipment and materials.
- Alternates to schedule of unit prices.
- Breakdown of tender, e.g., general requirements, site work, shaft collar installations, and indirect field cost.

The shaft collar contract specifications included recommendations from an external blasting consultant for controlled drilling and blasting of the shaft collar excavations. These recommendations included:

- excavation sequence;
- pre-shearing requirements for the shaft concrete excavations, bottom of collar pad to 8 m depth and shaft excavations from 8 m depth to 15 m depth; and
- drawings of a pre-shear blast plan, typical pre-shear blast hole, typical bench round plan and section.

The pre-shearing, essentially a blast done ahead of the main excavation blast round to make the rock in the selected area easier to remove, did not work well due to confining stresses in the shaft corners.

2.1.2 Upper Shaft Construction Contract

While shaft collar excavation was in progress, AECL carried out a pre-qualification process for Contractors interested in bidding on the underground construction work at the URL. The pre-qualification process was used to short list qualified Contractors to bid on the underground work at the URL. This was fair to all Contractors responding to the pre-qualification because an extensive and costly effort is required in preparing tenders for specialized jobs such as this and pre-qualification limited the number of bidders to those deemed capable. A short list of bidders also served to simplify the tender evaluation process.

Thirty Contractors requested pre-qualification documents and 18 submitted applications. Of these, five were pre-qualified in the shaft-sinking category. Preliminary discussions were held with the pre-qualified Contractors to exchange ideas on appropriate contract format and how best to handle many of the unusual requirements of the project.

The upper shaft was constructed under Contract No. 4 – Hoist Installation and Shaft Sinking, and was of a similar form (stipulated unit price) as the shaft collar contract. The decision to use this contract form was based on its common use at the time for underground mining development. Mine owners generally preferred this contract format as it placed all the risk on the contactor, who was expected to provide a fully serviced operating shaft that can be turned over as a "turnkey" project to the mine operators. The arrangement generally requires the mine operator take a hands-off position, leaving it up to the Contractor to complete the work. Since the Contractor tenders with his lowest competitive unit price per metre, the owner is in a good position to obtain the best price possible. However, to be successful, the contract must focus on completing the shaft as stipulated and as quickly as possible without delays or extra work. Risk of changing geological conditions is built into the unit price as is the Contractor's profit margin. It is not in the Contractor's interest to have owner imposed delays or additional work, as this will cut into the profit expected, even if the owner offers generous compensation for additional work. This is especially true if the Contractor's resources are committed to a subsequent project after the tendered scheduled completion date. The stipulated unit price contact format was less successful at the URL due to the owner's requirement for extensive geotechnical characterization in conjunction with excavation advance, something that is not typically done in mining developments.

AECL provided designs and specifications for the upper shaft contract. These were prepared using a separate design contract awarded to a joint venture non-resident engineering team comprised of Canadian Mine Services (CMS) and Wardrop Engineering Ltd. The joint venture team assisted AECL with the preparation of eight contract documents for the construction of the URL as well as with tendering arrangements and tender evaluations. CMS engineers also provided information concerning state-of-the-art blasting theory and recommendations for controlled blast design. An explosives manufacturer and distributor, CIL International, supervised trial bench blasts on surface prior to the sinking of the shaft collar and provided blast designs as part of their blasting product supply service.

The Contractor was responsible for cooperating with AECL's blasting consultant and also responsible for demonstrating performance of the blast designs. However, in practice these blast designs did not work as hoped. The specified design involved an attempt to provide controlled drilling and blasting on perimeter walls but it was not effective because of the need to fan the bench blast holes. The contract did not accommodate alternatives but as the blast designs did not yield the expected results, the Contractor had to resort to their own established blasting method for benching in order to maintain progress. The objective of having three levels of wall quality in the upper shaft excavations was not achieved (discussed further in Section 2.1.4 below). Performance objectives were not identified in the contract, therefore, the project manager had no baseline to evaluate the controlled drilling and blasting performance. This illustrated the need for a contract format that ensures the requirements of the owner are defined and understood by the Contractor with provision for geological uncertainty. In order to have a cooperative arrangement between the owner and the Contractor, it is essential that both parties have a common understanding of the objectives and requirements of the contract.

For the upper shaft, a stipulated unit price contract format had unit prices for delays requested by AECL for characterization. As discussed above, the Contractor assumed most of the risk under this contract format. Additional costs associated with unforeseen delays or extra time needed for geotechnical characterization work had to be negotiated and agreed upon, usually after the fact. This is not a desirable situation for either the owner or the Contractor and does not lead to a good working relationship.

In addition to those items mentioned for Contract No. 2 above, the Contractor was required to provide the following in their tender for Contract No. 4.

- Historical information to prove the Contractor's ability to complete similar projects by contract type and brief work description.
- Acknowledgement of any bulletins issued after issuance of the tender specifications.
- Company financial statements covering changes and additions since the submission of pre-qualification information for Contract No. 4.
- Construction schedule in bar chart format, supplemented by narrative description indicating on a calendar week by week basis the performance of work tendered.
- Construction methodology for:
    - Hoist installation, and
    - Shaft sinking, including: mobilization, equipping, services, shaft plant, surface operation, Contractor's work cycle, drilling and blasting, shaft dewatering, experimental access, core drilling, station and pump station excavation, shaft furnishings, skip/cage installation and commissioning, electrical substations and demobilization.
- Schedule of payments.
- Labour and equipment unit rates.
- Tendered unit and lump sum prices with estimated quantity provided.
- Alternatives to the schedule of prices.

Contract management for shaft sinking was the responsibility of an external engineering company, experienced in mine development. This company had a project manager and shift inspection specialists on site to administer Contract No. 4 for AECL and oversee the construction and commissioning work carried out by the shaft sinking Contractor. However this put the external engineering company in a difficult "go-between" position when it came to resolving contract related issues. Due to the need for AECL staff to create change orders to the contract in order to permit characterization delays, a complicated and inefficient communications protocol existed that resulted in difficult working relationships.

2.1.3 Estimated and Actual Construction Schedule and Cost

This section describes the estimated and actual project schedule and cost for Contract No. 4, the sinking of the upper shaft. It should be noted that the various components of the work are not absolute values as certain judgements were used in allocating various lump sums, unit rates and estimated values. Thus the cost summary should be used as a relative guide and should be should be assessed by experienced individuals when being applied to other applications.

Sinking of the upper shaft (covered by Contract No. 4) commenced in 1984 May and was completed to a depth of 255 m by 1985 May.

Table 2 provides a summary of the work carried out during the Upper Shaft Sinking Project.

Three instrument arrays were installed during sinking, which interrupted sinking activities for the duration of time required to diamond drill boreholes and install instruments for monitoring rock mass response. Shaft sinking crews, with the exception of staff to continue with shaft operation and maintenance, were demobilized from the project while characterization work was being carried out at the instrument arrays. Also, shaft-sinking advance was halted for the excavation of shaft stations at 130 m and 240 m depths and again to install furnishings at the 130 and 240 Level shaft stations.

The 130 Level shaft station (Figure 3) comprised only Room 101, which included an open area for receiving equipment and materials as well as an excavation for a loading pocket. To accommodate possible future lateral development from the 130 Level, it was deemed advisable to excavate the loading pocket during shaft sinking, as it would have been very expensive to do this after shaft furnishings were installed. The loading pocket furnishings were not installed as lateral development was never carried out.

The 240 Level shaft station included an open area for receiving equipment and materials as well as a loading pocket excavation (Room 201). Level development work, comprising Room 202, electrical sub-station; Room 203, access tunnel; Room 204, mine water pump station and settling sump; Room 205, refuge station; and Room 206, access tunnel and diamond drill station; Room 207 north access tunnel; Room 208, ventilation raise base; and a portion of Room 209 access tunnel, was excavated as part of the upper shaft sinking contract (Figure 3).

The total costs incurred in the construction of the Upper Shaft were $5,634,000 in 1984 Canadian dollars.

Table 3 provides a breakdown of the total costs and Table 4 provides the unit costs.

The general contract costs of $115,000 (1984 Canadian dollars) represent 2.0% of the total cost and include the shaft sinking Contractor’s mobilization, demobilization, additional insurance required under the contract, delays for contract start up and safety instruction for personnel working on the project.

Shaft sinking costs of $1,054,000 (1984 Canadian dollars) represent 18.7% of the total costs and include:

- contract work items including shaft excavation, installation of shaft timber sets, installation of shaft timber bearing sets, final scaling and cleanup, Contractor’s indirect costs, and miscellaneous extra work such as wooden wedges for timber blocking, timber inspection, modifications to one bearing set and the use of AECL’s loader during the sinking; and
- AECL Owner-supplied materials, which included the shaft timber sets and detonators and explosives.

Table 2: Actual Progress of Work for the Upper Shaft Sinking in Calendar Days

Item	Description	Start Date: 1984/5	Finish Date: 1984/5	No. of Days	Accumulated Days
1.	Sink Shaft to Instrument Array No..1 (62-m-depth)	May 13	June 11	30	30
2.	Install Instrument Array No. 1	June 12	June 30	19	49
3.	Sink Shaft to 130 Level Shaft Station	July 01	August 06	37	86
4.	Excavate 130 Level Shaft Station	August 07	August 25	19	105
5.	Sink Shaft to Instrument Array No. 2 (185-m-depth)	August 26	September 29	35	140
6.	Install Instrument Array No. 2 - Note (1)	September 30	October 21	22	162
7.	Sink Shaft to Instrument Array No. 3 (218-m-depth)	October 22	November 15	25	187
8.	Install Instrument Array No. 3	November 16	December 01	16	203
9.	Sink Shaft to 240 Level Shaft Station.	December 02	December 15	14	217
10.	Excavate 240 Level Shaft Station - Note (2).	December 16	February 12	59	276
11.	Sink Shaft to shaft bottom.	February 13	March 03	19	295
12.	Final scaling, shaft bottom furnishings, miscellaneous	March 04	March 18	15	310
13.	Install furnishings at 130 and 240 Level stations and hoist changeover commissioning	March 19	May 19	62	372
14.	Deficiencies and additional work.	May 20	June 12	24	396

Note (1) Includes two days for hoist repairs.

Note (2) Includes Christmas shutdown period – 13 days.

Table 3: Breakdown of Total Costs for the Upper Shaft Sinking Project

Item	Description	Cost ($ of the Day)	% of Total
1.	General Contract Costs	115,000	2.0
2.	Shaft Sinking	1,054,000	18.7
3.	Characterization Requirements	727,000	12.9
4.	Level Development	521,000	9.3
5.	Hoist Equipment	1,157,000	20.5
6.	Furnishings and Construction	1,209,000	21.5
7.	Resident Engineering	851,000	15.1
	Total Cost	**$5,634,000**	**100.0**

Table 4: Unit Costs of Excavations for the Upper Shaft Sinking Project

Item	Description	Shaft Sinking	Level Development
1.	Depth of Shaft Excavation	235.5 m	---
2.	Theoretical Volume of Excavation	3,229.0 m^3 (4.9 m x 2.8 m)	---
3.	Actual (Estimated) Volume of Excavation	3,991.5 m^3	2,276.8 m^3
4.	Contractor’s Total Cost	$ 1,054,000	$ 521,000
5.	Estimated Engineering Cost	$ 349,000	$ 226,000
6.	Total (Estimated) Capital Cost	$ 1,403,000	$ 757,000
7.	Contractor’s Unit Cost per Metre (Cubic Metre)	$ 4,479 / m	$ 229 / m^3
8.	Total (Estimated) Unit Cost per Metre Cubic Metre)	$ 5,961 / m	$ 328 / m^3
9.	Total Unit Cost per Cubic Metre based on Theoretical Volume	$ 434 / m^3	---
10.	Total Unit Cost per Cubic Metre based on Actual Volume	$ 351 / m^3	---

Costs associated with the characterization work carried out in the shaft during sinking operations totalled $727,000 (1984 Canadian dollars) and represented 12.9% of the total cost. These costs included costs associated with flattening the shaft bottom, diamond drilling and installing cover plates to protect instrumentation from blast damage at the three instrument arrays; additional work related to the usage of a blasting set and modifications to the blasting set, removal of manway ladders at the shaft collar prior to start up, installation of a shaft watering sprinkler system, installation of a water collection system at instrument arrays for hydrogeological monitoring, excavation of access tunnels on the 240 Level and the installation of a instrument array in Room 209 on the 240 Level.

Costs associated with level development carried out in conjunction with shaft sinking and immediately after reaching the final shaft depth of 255 m totalled $521,000 and represented 9.3% of the total costs. This cost included Room 101 shaft station on the 130 Level; Room 201 shaft station on the 240 Level; owner supplied explosives; ground control and support; contract indirect costs; staging for installing ground control and support; excavations required for pump station and ventilation services; and 240 Level trackwork.

Costs associated with the URL hoist installation totalled $1,157,000 (1984 Canadian dollars) and represented 20.5 % of the total costs. These costs included:

- Contractor costs of $321,000 for hoist installation, miscellaneous extra work, delays for hoist repairs, installation of ropes and counterweights and Contractor indirect costs; and
- Owner-supplied equipment costs of $836,000, including the hoist ($680,000), hoist commissioning ($30,000) skip/cage and counterweight ($82,000) head sheaves ($33,000), and hoist ropes ($11,000).

Costs associated with furnishings and construction totalled $1,209,000 (1984 Canadian dollars) representing 21.5% of the total cost. These costs included:

- Contractor costs for shaft electrical/mechanical furnishings ($432,000), dewatering system ($265,000) shaft station steelwork ($86,000) and concrete work ($52,000), Contractor's indirect costs ($90,000), and miscellaneous equipment such as mine phones, cap lamps, shaft power cables, track concrete work, temporary ventilation ducting, shaft signal system, additional brattice timber, potable water lines, shaft station doors, mine dry facilities, Contractor's site office, shaft loading pocket dump doors, pipe hanger brackets, etc; and
- Owner-supplied equipment cost of $203,000, including mine power centres ($67,000), armoured electrical shaft cables ($36,000), dewatering pumps ($32,000) motor control centre ($26,000), downcast fan and propane heater ($23,000), portable power panels ($12,000) shaft brattice timber and shaft bottom pump ($7,000).

Costs associated with non-resident and resident engineering support totalled $851,000 (1984 Canadian dollars) and represented 15.1% of the total cost. Resident engineering included shift inspection and contract administration by an on-site engineering firm ($751,000). Non-resident engineering support was provided by two off-site engineering firms, who assisted with conceptual designs and contract preparation ($100,000).

2.1.4 Construction Requirements

The Contractor was required to excavate a rectangular shaft 2.8 m by 4.9 m at the specified location. The shaft was excavated from a collar depth of 15 m to a finished depth of 255 m. Shaft stations were designated at approximately 130 m and 240 m depths.

Excavation work was carried out over a 16-hour period per day, from 4 PM (1600 hours) to 8 AM (0800 hours). The eight-hour period from 0800 hours to 1600 hours was reserved for AECL's experimental and characterization work. On occasions other than the 0800 to 1600 hours shift the Contractor was required to provide AECL experimenters access to the shaft bottom upon request, except when blasting activities were taking place. Such additional access was billed to AECL at a pre-determined standby rate tendered under the contract.

Related work carried out by the Contractor under Contract No. 4 included:

- shaft operation and maintenance,
- hoisting plant operation,
- supervision and support services,
- geotechnical diamond drilling during shaft sinking,
- shaft ground support,
- dewatering,
- cast-in-place concrete work, and
- heavy timber construction.

The characterization requirements place severe constraints on the shaft sinking process, and required meticulous coordination of the characterization and shaft sinking activities. The requirements to have the shaft available to the AECL's staff at 0800 hours each morning severely affected the Contractor's sinking cycles. The Contractor could not start to load a bench if it was not possible to blast it, complete the mucking and have the shaft cleaned and ready for 0800 hours. The time lost in this manner was used to install services or carry out equipment maintenance whenever possible. However, if no further work relating to shaft construction could be carried out, the Contractor argued for delay compensation as part of the contract.

For the 0800 to 1600 dayshift period, the Contractor was required to provide the following services for geotechnical characterization activities:

1. Provide by 08:00 hours each morning:

    - compressed air and water lines were set up for washing and cleaning,
    - the shaft bottom above the sump area, mucked out in preparation for geological mapping,
    - shaft walls scaled and the shaft bottom and shaft walls to a height of 3 m above the last bench washed clean to the structural concrete placement standard CSA A23.1 M77,
    - 110 volt AC power supply for instrumentation and the setup of floodlights for geological mapping, and
    - two step ladders of approximately 3 m height for shaft wall mapping;

2. Set up a special shaft sinking stage and level it at the specified elevation +/- 10 mm, and locate it horizontally with an accuracy of +/- 10 mm as required by AECL's staff (The sinking stage was designed to accommodate both the Contractor's construction-related requirements and AECL's requirements for geological mapping and for installing and monitoring instruments);
3. Install approximately 24 reference points on the shaft walls and shaft bottom each day for control of stereophotography and geological mapping, surveyed with an accuracy of +/- 10 mm; and
4. Drill holes in the shaft wall for installation of convergence pins, and hydrogeological testing.

In addition the Contractor was required to:

1. provide AECL's staff additional time in the shaft to complete geological mapping or access instrumentation located throughout the shaft;
2. provide AECL's staff access to the shaft during the excavation shifts to read instruments, or collect data relating to the excavation; and
3. install water rings at intervals as required down the shaft to catch water flowing down the shaft walls for chemical analysis and hydrogeological monitoring of water inflow rates.

An electric hydraulic diamond drill was designed and built by Longyear Canada Inc. to meet the drilling requirements at instrument arrays within the shaft. Because of the space constraints at the shaft bottom, the drill had to be compact, while still meeting the specifications of being electric hydraulic, capable of drilling HQ-3 size (96-mm-diameter) holes, and capable of drilling within confined locations with difficult orientations.

AECL specified that the Contractor was required to prepare and submit blast designs for conventional (uncontrolled) and controlled drilling and blasting with 50% and 90% half barrel remnants specified to be left uniformly distributed around the perimeter of the shaft or shaft station excavations after scaling down all loose or shattered rock. A half barrel is the remnant of the drill hole loaded with explosive at the perimeter of the excavated area. The Contractor was required to maintain detailed drawings of the shaft and shaft stations as excavated showing the blasting results. They were then required to make comparisons with the blast designs and submit reasons for any deviations. It was envisioned that these three levels of blasting would be achieved in the excavations as directed by AECL, but in practice only the controlled drilling and blasting was used. Blasting procedures and methods used were as recommended in either E.I. du Pont de Nemours and Co. Blasters' Handbook or CIL Blasters' Handbook unless otherwise directed by AECL's Resident Engineer. The lack of clarity of this excavation quality specification (i.e., lack of a clearly defined method of measuring blast quality) became a source of contention during upper shaft sinking.

The Contractor was responsible for the safe handling, deliverance and use of explosives from the AECL explosives magazines located on surface. The Contractor was required to ensure that blast vibrations did not exceed peak particle velocities of 50 mm/s at concrete structures when blasting adjacent to concrete structures (Kuzyk et al. 1993, Kuzyk et al. 1991).

A summary of the bench blast cycle time for all benches in the upper shaft (18.4 m depth to 255 m depth) is provided in Table 5. The average shaft advance per bench was 0.69 m and the average cycle time per bench was 6.45 hours/bench. This translated to 9.31 hours per metre of advance.

Table 5: Average Bench Cycle Time Summary for the Upper Shaft

Item	Description	
1.	Number of benches (18.3 m depth to 255 m depth)	343
2.	Shaft advance / bench	0.69 metres
3.	Measure and mark face	0.15 hours
4.	Drilling time with hand-held plugger drills	1.11 hours
5.	Inspection, cleaning and loading of explosives	0.75 hours
6.	Blasting and ventilating blasting gases	0.46 hours
7.	Mucking	1.98 hours
8.	Installation of ground control and support	0.09 hours
9.	Setup and tear down of the blasting set	0.10 hours
10.	Support characterization activities	0.30 hours
11.	Delays	0.80 hours
12.	Cycle time per bench	6.45 hours per bench
13.	Cycle time per metre of advance	9.31 hours per metre

2.1.5 Excavation Maintenance and Monitoring

The Contractor was solely responsible for maintaining the stability of all underground excavations and excavated surfaces and for the maintenance and support where necessary of all excavated surfaces for the duration of the contract. As soon as possible after blasting each round, the Contractor was to scale and remove all loose, unstable or potentially unstable material from newly-exposed surfaces and to maintain all exposed surfaces free of hazardous materials.

In order to facilitate AECL's geological mapping, the Contractor painted a horizontal chainage mark and elevations at 2.5 m above the bench and around the shaft walls after the walls were washed down. When excavating the shaft stations, the Contractor was required to paint chainage marks and numbers at 5.0 m intervals on each side of the excavated headings after the headings were washed down.

2.1.6 Post-construction Stability of Excavation

The Shaft Inspection Record Books are mandated by the Province of Manitoba and describe the condition of and activities in shafts and raises in operating mine sites within the Province. The Shaft Inspection Record Books for the URL do not indicate any instability. Some notations are made regarding small pieces of rock removed from timber sets during shaft inspections. Upon discussing these notations with the Contractor on site, it was determined that this material was related to mucking operations, e.g., spillage from the skip during loading and/or dumping.

2.1.7 Lessons Learned

The URL involved excavating a dedicated facility in an undisturbed site for the development and in situ demonstrations of geotechnical technologies required to construct a repository for high-level radioactive waste and used nuclear fuel. This provided an opportunity to characterize changes in the surrounding geosphere as excavation advanced. The management and execution of stipulated unit rate contracts, e.g., Contracts No. 2 and No. 4, presented several challenges and some benefits to the early stages of the development of the URL. Carrying out geotechnical characterization activities during construction of the URL identified issues relating to appropriate contract format that would be applicable to the construction of a repository.

In the early stages of Contracts No. 2 and No. 4 it became apparent that conventional mining and construction methods were not entirely satisfactory for meeting the needs of the scientifically controlled project envisioned when the tender documents were prepared. The contract documents attempted to utilize conventional mining techniques for construction of a facility for which the owner’s objectives also included controlled drilling and blasting and detailed characterization during construction. It is now recognized that the conventional approach to shaft sinking and normal contract format at the time favoured objectives that were time and cost sensitive whereas the URL project required an approach that favoured the acquisition of knowledge relating to the in situ characteristics of the rock and the quality of work, in addition to cost- and schedule-control. Contract documents similar to those favoured by mine developers proved to be inadequate, as the Contractor’s performance expectations were compromised by owner interference and the owner’s desired excavation quality was not achieved. Although the mining technology used to construct the URL was appropriate, the excavations needed to be designed like an underground civil engineering project where stability and longevity is a prime objective. While this can be seen as a negative outcome for the type of contract used in the upper shaft, the experience gained allowed a more suitable form of contract to be developed for subsequent work.

One clear outcome from the upper shaft development was the recognized need for a “design-as-you-go” approach to accommodate geological uncertainty and evolving site characterization information. This approach is known formally as the “Observational Method” (Peck 1969). The Observational Method is a geotechnical project design, construction and management method that requires a project to develop alternative approaches to work that can be applied should conditions change as the project progresses. This allows a project to be modified as it progresses, using information on the geotechnical conditions encountered and the rock mass responses measured during previous excavation and construction. The method consists of a series of necessary and interrelated steps that embody good scientific and engineering practice. The embodiment of these steps in a project structure limits the inevitable uncertainties in predicting the performance of the underground works and hence the suitability of the designs.

The need for the integration of geotechnical characterization and construction activities was an important realization during the upper shaft construction. Geological characterization was carried out to determine baseline conditions and to gain a comprehensive understanding of the excavation-induced changes in rock mass and groundwater conditions that could affect the long-term stability and performance of the underground facilities. Since excavation advance exposed changing conditions, it was recognized that, in future construction projects, geotechnical characterization and construction activities should be integrated so that knowledge of current in situ conditions was readily available for planning purposes. The form of Contract No. 4 that resulted in a Contractor developing the excavation with no or limited financial compensation for the owner's interference for characterization program activities was clearly not suitable.

Contract No. 4 administration carried out by the on-site project manager was not effective because the contract format did not effectively accommodate geotechnical characterization activities that the owner carried out while excavation and construction work was in progress. As these contracts progressed, it became clear that the third-party contract manager was ineffective at resolving issues relating to delays and access by AECL's characterization staff during shaft sinking activities.

During the execution of the Contract No. 4, a process of increasing coordination of shaft sinking activities was developed. Each morning the Construction Manager, AECL's Characterization Coordinator and the Contractor's superintendent met to schedule work for the following 24 hours and plan upcoming activities. Weekly management review meetings were held to discuss problems that had arisen since the previous meeting and to plan the work for the following week. Monthly meetings involved AECL and Contractor senior management in discussions and resolutions of priorities, budget and schedule-related decisions. This set the philosophy for cooperative efforts in subsequent contracts.

The upper shaft clearly showed that a standard stipulated unit price contract was not a suitable form to advance the access and do initial development when characterization of the rock mass was important. However the lessons learned led to improvements in the contract form that were implemented in the lower shaft development.

2.2 LOWER SHAFT (255 m to 443 m) (Contract No. 8)

The approach to construction of the shaft extension (lower shaft) was based on lessons learned from the upper shaft development. In 1986 July, a cooperative research program began with the US Department of Energy (US DOE), and the parties agreed to extend the existing URL shaft from 255 m to 465 m. Later changes in the USDOE program resulted in the depth of the URL shaft extension being modified to 443 m. To help meet the needs of the owner's geotechnical characterization program, in particular simplifying numerical analyses of shaft excavation, the shaft extension was excavated with a circular cross section (see Figure 4), using a full-face blasting method that included a burn cut. This resulted in excavation of cylindrical volumes of rock, which provided a geometry that was more easily analyzed.

Full-face blasting with a burn cut was not common practice in Canada at the time. Once it was decided to use this excavation method, steps were taken to ensure that the desired blast performance was realized and safe working conditions were maintained. These steps included:

- procuring expertise in blast design and performance monitoring;
- preparing full-face blast designs for inclusion with the contract;
- allocating time for testing and refinement of the blast designs in the initial few blast rounds;
- monitoring the performance of the blast hole layout and detonation sequence with vibration monitoring equipment; and
- using accurate electronic detonators in a trial blast to provide a baseline to compare with the pyrotechnic detonator system planned for the rest of the blasts.

A Galloway stage was designed and built to accommodate the geotechnical activities and the sinking Contractor's requirements for full-face blasting in a 4.6-m-diameter shaft. Blast designs incorporated controlled drilling and blasting techniques to minimize blast-induced damage on the shaft walls. Geotechnical activities were carried out on the day shift, and sinking activities were carried out between 4:00 PM (16:00 hours) and 8:00 AM (08:00 hours).

To better accommodate the geotechnical activities and full-face sinking method, a cost-plus fixed-fee contract format was used (Kuzyk and Versluis 1989). The format reduced the risk to the Contractor and allowed geotechnical activities to be integrated with excavation and construction. Basically, the requirements for geotechnical characterization activities carried out by AECL's staff were the same as those identified in the upper shaft sinking contract. However, the requirements were more clearly defined and explained in the tendering process. The risks were more evenly shared between the Contractor and the owner (e.g., AECL took responsibility for blast design and performance while the Contractor was responsible for blast implementation according to AECL's plan), and the miners' bonus was modified from that used in the upper shaft excavation contract to include meeting the owner's requirements rather the maximizing construction advance and schedule.

AECL's construction management organization for the lower shaft was based on experience gained during the sinking of the upper shaft. It was recognized that AECL needed to increase direct control of the project if geotechnical activities were to be accommodated effectively and the risk of blast-induced damage was to be minimized during sinking operations. Therefore, AECL decided to assume direct responsibility for construction management during the shaft extension.

2.2.1 Construction Management Organization

The construction management organization for the shaft extension, shown in Figure 5, provided the flexibility necessary to effectively accommodate diverse in situ conditions and optimize the plans for the technical program and construction integration. The technical team was responsible for managing the characterization program, including identifying and planning the research and development activities during sinking operations; coordinating characterization activities during dayshift and at instrument arrays; core logging; and supervising the diamond drilling Contractor. The geotechnical information gathered during characterization necessitated design changes as the sinking progressed.

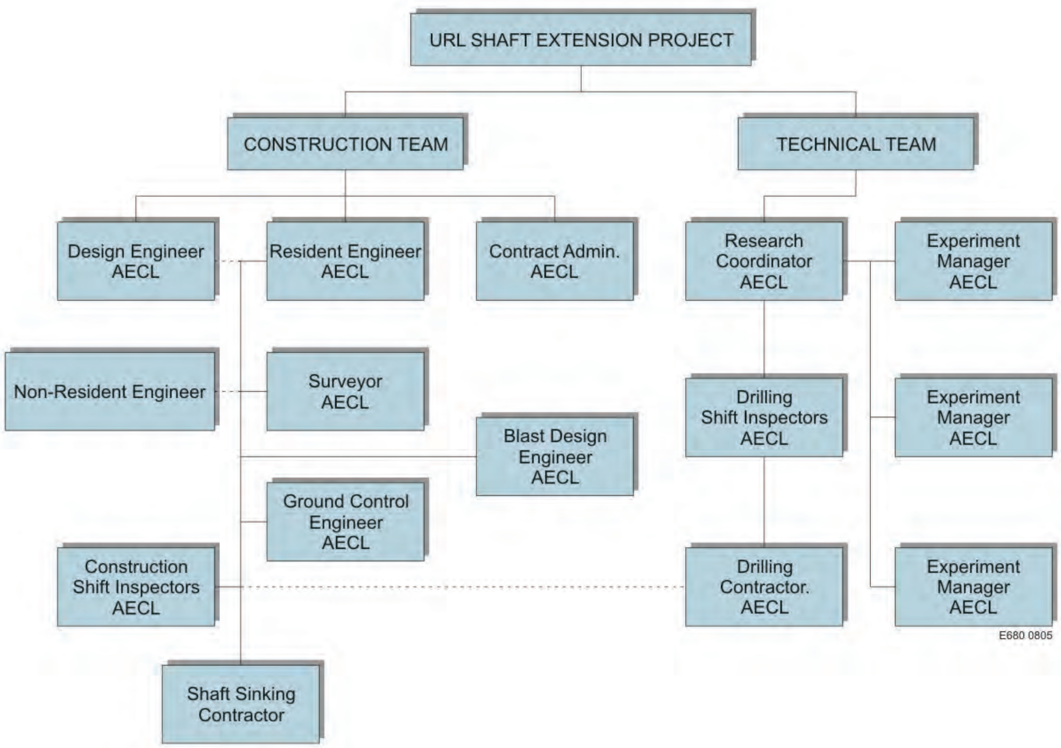


Figure 5: Project Management Organization for the URL Shaft Extension

The construction team was responsible for construction, including managing the design, excavation, construction and operation of facilities required to carry out the sinking work and to support the technical program (Peters et al. 1988). They also assumed a lead role in integrating the technical activities with construction.

AECL employees filled most of the positions in the construction management organization for the lower shaft-sinking project. However, the need for specialized technical expertise in some areas and short-term increase in staffing in others necessitated the attachment of consultants in five key areas. These areas were: non-resident engineering; contract administration, blast design; ground control; and quality control inspection.

A non-resident engineering team comprised of V.B. Cook Co. Ltd and Wardrop Engineering Ltd. were contracted to design the shaft extension and produce the construction drawings and were retained as the shaft extension design consultants to provide non-resident engineering support. The non-resident engineering firms provided support in updating construction drawings, producing as-built drawings, providing consulting support, and providing designs to accommodate modification and changes, such as 300 Level safety doors, 300 Level loading pocket, and a shaft dewatering system.

2.2.2 Design Process

The design process for the shaft extension involved the preparation of both conceptual and detailed designs. Designs were needed to describe the scope of the project, and the equipment, services, facility and technical program requirements for contract tendering. Detailed designs were provided where the information existed to prepare them; otherwise conceptual designs were prepared (Kuzyk et al. 1990, Peters et al. 1988, 1990).

The conceptual designs primarily consisted of drawings and descriptions of the unique equipment, methods and procedures needed for the geotechnical program. Experience gained from the upper shaft sinking contract, and requirements for technical activities identified by the technical team, formed the basis for these designs. It was necessary to prepare the designs for the contract tendering process, so the sinking Contractors had an understanding of the project requirements. However, these designs could not be completed in detail until the sinking contract was tendered because the Contractor's input was also needed.

The non-resident engineer assisted AECL in the preparation of the conceptual and detailed designs. The designs were then subject to an extensive iterative review to ensure that the characterization program objectives were being met in the most practical and cost-effective way. Specific emphasis was place on assessing and minimizing the impact of characterization activities during excavation and incorporating hydrogeological and geomechanical information into the planning process. It was recognized that the conceptual designs would be subject to change once the shaft-sinking contract was awarded. It was expected that the Contractor would have additional requirements and suggestions for improvements based on their specific experience.

Conceptual designs were provided for:

- a Galloway stage configuration commonly used in the shaft sinking industry with innovative modifications to accommodate the geotechnical activities;
- full-face blast rounds employing controlled drilling and blasting techniques and a burn-cut to provide a flat geometry at shaft bottom for geotechnical modelling purposes;
- Instrument array layouts with protective cover plates to protect instrumentation from shaft blasting operations, installation of shaft sets and services, as well as other sinking activities;
- shaft stations, electrical sub-stations and level layouts to minimized rock mass disturbance until characterization could be completed; and
- shaft bottom facilities, including a water handling system complete with retention sumps to facilitate hydrogeological monitoring.

The non-resident engineer also assisted AECL in the preparation of detailed designs and specifications for:

- a second 259-kW standby diesel generating unit to provide standby emergency power for the 420 Level;
- additional transformers and switching gear in a substation on the 420 level;
- an air cooled rotary screw compressor, which increased the compressed air capacity at the URL to 54 m^3/min from 34 m^3/min at 690 kPa;

- a loading pocket equipped with a measuring hopper and storage bin to better accommodate the hoisting of development muck from the 420 Level (the 204 Level loading pocket was not equipped with a measuring hopper);
- hoist modifications, including a new rope, sinking cross-head and bucket, upgraded brakes and motor ventilation system, since the hoist had to operate in single rope configuration;
- a return air system comprised of a ventilation raise bored between the existing 240 Level and the new 420 Level, with a booster fan installed on the new level to maintain the required ventilation air volume of 11 m^3/sec; and
- shaft bearing sets comprising cast concrete, which incorporated water collection rings and catch basins, to provide support for the new steel set configuration.

The shaft extension had a 4.6-m-diameter. Steel sets were installed at 3 m intervals, instead of fir timber sets at 2.5 m intervals in the upper shaft. The new spacing reduced the number of sets required and improved shaft wall visibility and access, which was beneficial for characterization. All components of the steel sets were galvanized for corrosion protection.

A probe hole was drilled from a diamond drill station excavated on the 240 Level near the shaft. Relevant information regarding the rock mass characteristics below the 240 Level, e.g., geological, in situ stress and hydrogeologic conditions in Fracture Zone #2, were included. This information of ground conditions reduced the uncertainty and the risk associated with shaft sinking.

The non-resident engineer also assisted AECL with the preparation of a description of the scope of work for the shaft extension project, a cost estimate and schedule and an evaluation of the cost reduction alternatives in preparation for tendering the contract for constructing the lower shaft. As much detailed design information as possible was assembled for tendering. This information mainly comprised drawings and specifications for upgrading the electrical and mechanical services needed for the shaft extension and other levels. It also comprised detailed core logs and other geotechnical information available from the characterization program.

2.2.3 Construction Contract and Tendering Process

The lower shaft was constructed under Contract No. 8, URL Shaft Extension. The form of tender for this contract was cost of work plus fixed fee format. The scope of work covered under this contract generally comprised the provision of all labour, some of the required materials and the sinking plant necessary to extend the existing 255 m shaft to 465 m (changed later to 443 m), excavate two shaft stations and install the required equipment and furnishings for these. The owner, AECL, would carry out an interactive characterization program in association with the construction work. Although the Contractor’s prime responsibilities were to provide a safe, efficient and cost effective shaft sinking operation, his experience and expertise in mining were also valued. As part of the work, the Contractor provided expert advice and opinion on various mining topics at AECL’s request. It was emphasised from the outset in the contract tender, that for the program to be a success, a superior cooperative effort would be required from both AECL’s and the Contractor’s staff in all aspects of the work.

The form of tender specified that the bonus paid to the Contractor’s crews would meet with AECL’s approval and would be based upon a combination of production, wall quality, drill alignment, safety, housekeeping and cooperation. This requirement was based upon AECL’s experience during the upper shaft sinking contract where the bonus was based on the Contractor's need to attract experienced shaft sinking personnel and a desire to maximize their profit margin by increasing productivity or the rate of advance. Excavation quality and support for the geotechnical characterization activities that had to be carried out in conjunction with shaft sinking were not adequately supported in the upper shaft excavation.

The cost plus fixed fee contract format provided the owner with a high degree of control and project flexibility and placed acceptable limits on the Contractor's risks. The fixed fee covered items that were fixed and easy to estimate, such as the Contractor's overhead and specified equipment and services. For the shaft extension project, work that was considered as fixed (fixed-fee) or "lump-sum" were:

- profit;
- head office support and overhead;
- site supervisory and administration staff; and
- provision of specified sinking plant and equipment, e.g., sinking stage, drill jumbos, mucking equipment, cross-head and sinking bucket.

These components are based on a specified project scope and were activities considered to be under the Contractor's control. Permanent shaft furnishing and equipment were included here if supplied by the Contractor, however as an alternative, these could be procured directly by the owner. Some examples of furnishing purchased by AECL are shaft sets, hoist ropes, compressors and electrical power centres. There was an opportunity for cost saving if AECL procured these items.

The cost component of the contract covered the direct cost of Contractor supplied labour, specified materials and specified consumables required to carry out the work and included:

- direct wages, salaries and expenses, including bonus payments to the Contractor's crew;
- approved subcontracts;
- materials incorporated in the work;
- expendables, such as drill steel and explosives;
- equipment rentals;
- on-site maintenance; and
- statutory labour burdens, fees and taxes.

These expenses were controlled by the owner through an invoice procedure. The resident contract administrator, see Figure 5, was part of the construction team and was responsible for reviewing and recommending payment of the Contractor's invoices on a schedule established in the contract.

The scope of the work was identified in the contract documents. Specific reference to the nature of the geotechnical characterization requirements and the scientific studies was made in the tendering documents. To ensure the appropriate topics were identified, there was input from the owner's geotechnical group. The non-resident engineering firms, who also helped with the tendering and award process, provided assistance with technical issues on integration of characterization and scientific studies with excavation.

The scope of the work described in the contract included items such as:

- modifications to the head frame and hoisting system, commissioning the hoisting system for sinking operations and provisions of all sinking equipment;
- operation and maintenance of the hoisting system during sinking operations;
- operations support for the scientific studies, including time required for the shaft turnover, access time required by scientific groups, electrical and mechanical services needed at instrument arrays, data systems and devices to protect instruments from sinking activities;
- conceptual and detailed designs for the Galloway stage to accommodate scientific studies and sinking activities;
- excavation size and configuration, Level station configurations, loading pockets;
- shaft furnishings and services, water handling; and
- removal of all sinking equipment upon completion, installation of permanent operating systems and commissioning of the hoisting system for operating service.

Some of the contract requirements had an impact on the speed at which the shaft could be advanced. The need for the Contractor to turn over the shaft at 08:00 hours every day limited the advance rate as it limited the Contractor progress to what could be completed in 16 hours. A sliding schedule may have accelerated the advance but it would have made staffing for characterization activities more difficult. The full face blasting methodology had the potential of increasing the advance rate when scientific studies were not planned, as blast round lengths could be increased to take advantage of the additional time available. This shortened the time required for shaft sinking and allowed a method to reduce the overall cost of the excavation.

Each Contractor was required to meet a number of requirements in their tender including providing tables showing:

- duration, estimated cost of work, fixed fee and total cost divided into the;
    - shaft sinking (prepare and install shaft equipment, active sinking, station excavation 300 Level, station excavation 440 Level, four instrument arrays, and other items),
    - services and furnishing (shaft sets, shaft services, station landings, loading pocket, hoist changeover and other items), and
    - fixed fee per diem rates for changes in the work which effect the duration of the contract (deletions or additions);
- task breakdown for estimated cost of work (labour, materials and expendables & expenses); and
- crew make-up (crew type, personnel, quantity, hourly rate including fringe benefits, and bonus).

The tender also had to include:

- detailed description of the Contractor’s organization with names and curriculum vitae for key personnel;
- detailed description of the Contractor’s planned approached to the work;
- detailed description of the facilities and services provide by the Contractor as included in his fee;
- description of the Contractor’s experience with particular emphasis regarding work accomplished of a similar nature;
- subcontractors and material suppliers proposed for work on this project;

- alternate materials or equipment proposed, if any;
- list of addenda/bulletins issued by AECL to the tender documents and identified by addendum or modification number, date issued and date received that were considered in the preparation of the tender;
- agreement to recognize and abide by the contract documents;
- name and address of the bidding company; and
- signature of duly authorized person and corporate seal.

Major equipment, such as shaft sets, power centres and armoured shaft cables, mechanical services, hoist ropes, etc., required for permanent installations, was purchased by AECL to ensure competitive pricing. Major consumables, such as explosives, rock bolts and wire mesh for ground control, etc., were also ordered directly by AECL. Purchases by the Contractor were approved by the resident engineer before invoicing, and purchases over $10,000 required AECL approval before the Contractor could place the order.

The request for proposals for bidding on the shaft extension contract indicated that the Contractor would be required to:

- assume responsibility for the care, operation, maintenance and control of the existing headframe and hoist room;
- change over the existing hoist operation for sinking including installation of bucket, crosshead, hoist (sinking) ropes, dump, chute, collar doors, hoist transformer, hoist motor fan and adjustment of hoist, brakes, position indicator and controls;
- modify existing headframe steel, and installation of stage ropes, sheaves and winches;
- provide hoist operators and shaft men for all AECL and Contractor requirements;
- deepen the shaft from 252.650 m to 464.150 m (later modified to 443.0 m) in a circular cross section with an A-line diameter of 4.600 m;
- excavate shaft stations at nominal depths of 300 m and 440 m (later modified to 300 m and 420 m);
- install all shaft sets, pipes, cables, water rings, platforms, brattice, and miscellaneous ground support;
- assist AECL staff with the installation of four instrument arrays, with necessary piping, conduits and cover plates to protect the instruments and connecting cables;
- install shaft bottom sump and dewatering pumps;
- drill and install rock bolts, convergence pins and elevation markers;
- remove sinking components, reinstall skip/cage, hoist ropes, counterweight and conveyance and readjustment of power, brakes and controls; and
- return the facility to the owner control once construction was completed.

Contractors that were previously pre-qualified to tender work at the URL for Contract No. 4 were contacted to determine if they were interested in tendering the shaft extension project. Four of the five pre-qualified Contractors submitted expressions of interest for Contract No. 8. Bids were received and AECL selected a short list of three Contractors for further consideration.

Evaluation methodology of the bids received during the tendering process was predetermined and based on point ratings under the following categories by a team consisting of four evaluators.

- Site personnel and organization – qualifications, appropriate experience, depth and range of expertise and management system.
- Approach and methodology – knowledge of AECL's project requirements, effective and cooperative approach, cost and schedule control, and overall cost and technical benefit.
- Sinking plant – appropriate equipment and descriptive information.
- Adequacy of tender – completeness and level of detail, and logic of tender preparation.

After the bids were received and before the evaluations were finalized, visits to the Contractors' head office operations were arranged for the evaluation team as part to the review process. This was done to get a better understanding of the Contractor's capabilities, obtain any supplementary information subsequently identified, and to have preliminary discussions to exchange ideas on contract formats and how to best handle many of the unusual requirements of the shaft extension project.

The final selection was based upon criteria for technical ability and cost. The cost component of the selection criteria was a direct comparison of the normalized cost.

2.2.4 Actual Construction Costs

Table 6 is a detailed cost breakdown and summary of the lower shaft sinking costs incurred under Contract 8, in 1987 dollars of the day. The original committed contract cost was $3,320,700. Over the duration of the contract, AECL approved changes to the work, which accounted for an additional $722,600. Final expenditures upon completing of the contract totalled $4,266,400, which was $223,100 or 5.5% over the committed or budgeted cost. In the most part, this overrun was attributed to work at the loading pocket and electrical substation excavation at the 420 level, 300 Level station excavation and hoist system modifications.

Table 6: URL Lower Shaft Contract 8 Construction Cost

Description	Cost Type	Original Commitment	Approved Changes	Approved Commitment	Final Cost	Variance	Design Quantity (m^3)	Cost per (m^3)
Material - Shaft Sets(?)		$180,000		$180,000	$153,500	$26,500		
Hoist System Modifications	Labour	$32,700	$9,600	$42,300	$32,600	$9,700		
	Materials	$22,400	$11,300	$33,700	$43,100	-$9,400		
	Miscellaneous	$14,600	$3,400	$18,000	$29,800	-$11,800		
	Sub-Total	$69,700	$24,300	$94,000	$105,500	-$11,500		
Shaft Sinking	Labour	$340,200	$42,400	$382,600	$578,900	-$196,300	3,124	$185.31
	Consumables	$71,700	$7,500	$79,200	$68,100	$11,100	3,124	$21.80
	Miscellaneous	$14,200	$700	$14,900	$23,000	-$8,100	3,124	$7.36
	Ground Control	$0	$46,400	$46,400	$46,400	$0	2,600	$17.85
	Sub-Total	$426,100	$97,000	$523,100	$716,400	-$193,300	3,124	$229.32
300 Level Station Excavation	Labour	$52,600	$26,300	$78,900	$64,400	$14,500	540	$119.26
	Consumables	$4,900	$2,300	$7,200	$28,200	-$21,000	540	$52.22
	Miscellaneous	$1,300	$300	$1,600	$2,800	-$1,200	540	$5.19
	Ground Control	$0	$12,300	$12,300	$12,300	$0	540	$22.78
	Sub-Total	$58,800	$41,200	$100,000	$107,700	-$7,700	540	$199.44
420 Level Station Excavation	Labour	$59,800	-$2,600	$57,200	$86,800	-$29,600	608	$142.76
	Consumables	$5,600	-$800	$4,800	$6,800	-$2,000	608	$11.18
	Miscellaneous	$1,500	-$100	$1,400	$4,200	-$2,800	608	$6.91
	Ground Control	$0	$21,300	$21,300	$21,300	$0	608	$35.03
	Sub-Total	$66,900	$17,800	$84,700	$119,100	-$34,400	608	$195.89
Shaft Furnishings	Labour	$140,200	$0	$140,200	$149,700	-$9,500	3,124	$47.92
	Materials	$374,400	$158,200	$532,600	$592,100	-$59,500	3,124	$189.53
	Miscellaneous	$8,200	$0	$8,200	$12,800	-$4,600	3,124	$4.10
	Sub-Total	$522,800	$158,200	$681,000	$754,600	-$73,600	3,124	$241.55
300 Level Station Furnishings	Labour	$26,000	$17,500	$43,500	$34,800	$8,700	540	$64.44
	Materials	$16,800	$51,300	$68,100	$100,600	-$32,500	540	$186.30
	Miscellaneous	$3,000	$0	$3,000	$0	$3,000	540	$0.00
	Sub-Total	$45,800	$68,800	$114,600	$135,400	-$20,800	540	$250.74
420 Level Station (Including Loading Pocket)	Labour	$44,700	$0	$44,700	$107,100	-$62,400	608	$176.15
	Materials	$20,100	$0	$20,100	$72,200	-$52,100	608	$118.75
	Miscellaneous	$4,700	$0	$4,700	$400	$4,300	608	$0.66
	Sub-Total	$69,500	$0	$69,500	$179,700	-$110,200	608	$295.56
Other Miscellaneous Costs	Labour	$5,800	$0	$5,800	$5,500	$300	4,272	$17.95
	Other	$72,400	$15,100	$87,500	$76,700	$10,800	4,272	$19.24
	Sub-Total	$78,200	$15,100	$93,300	$82,200	$11,100	4,272	$19.24
Shaft Operations	Labour	$502,000	$113,700	$615,700	$444,800	$170,900	4,272	$104.12
	Consumables	$27,900	$7,300	$35,200	$15,300	$19,900	4,272	$3.58
	Sub-Total	$529,900	$121,000	$650,900	$460,100	$190,800	4,272	$107.70
Fee	General	$67,500	$0	$67,500	$65,100	$2,400	4,272	$15.24
	Hoist System	$22,400	$17,300	$39,700	$39,700	$0	4,272	$9.29
	Excavations	$730,300	$61,700	$792,000	$789,200	$2,800	4,272	$184.74
	Furnishings	$248,100	$20,900	$269,000	$277,200	-$8,200	4,272	$64.89
	Operations	$204,700	$79,300	$284,000	$281,000	$3,000	4,272	$65.78
	Sub-Total	$1,273,000	$179,200	$1,452,200	$1,452,200	$0	4,272	$339.93
Total Costs		**$3,320,700**	**$722,600**	**$4,043,300**	**$4,266,400**	**-$223,100**	**4,272**	**$998.69**

2.2.5 Contract Requirements

Based on experience and lessons leaned from the upper shaft sinking contract discussed in Sections 2.1.2 and 2.1.7, AECL requested that the Contractors meet contract requirements in their tender for Contract No. 8, URL Shaft Extension, that facilitated:

- following AECL's overall project control, including, overall project direction,

- meeting quality standards identified by AECL, which included procedures for:
  - coordination and integration of efforts between the Contractor and AECL's construction and technical teams.
  - implementation of design changes resulting from the geotechnical characterization as the shaft was extended,
- maintaining cost control and accounting procedures,
- monitoring and reporting on cost and 18-month schedule projections,
- carrying out a changeover of the URL hoist and shaft from normal balance operation to sinking configuration to extended depth; including bucket, crosshead, hoist ropes, surface dump, chute, shaft collar and safety doors, hoist modifications, brakes and commissioning, and then return to normal balanced operation after completion of the lower shaft extension,
- constructing a transition zone in the shaft from rectangular (2.8 m x 4.9 m) to circular (4.6 m dia.) configuration,
- preparing shaft bottom at four instrumentation arrays and installation of protective steel cover plates over connecting cables and conduits,
- excavating a loading pockets at the 300 and 420 Level stations and installing loading pocket equipment and furnishings at the 420 Level station,
- installing a sump and mine pumping and water handling equipment at the bottom of the shaft, and
- accept on-site delivery and responsibility of all items required for the shaft extension, including items purchased by AECL.

AECL recognized the benefits of utilizing the Contractor's experience and required the Contractors to make recommendations for improvements to conceptual designs and prepared for shaft extension contract that would enhance the technical program and improve upon schedule and cost objectives.

2.2.6 Lower Shaft Construction

Several changes in the construction methodology resulted in improvements during the lower shaft extension contract. These included:

- a Galloway Stage was utilized to facilitate shaft sinking;
- controlled drilling and blasting designs were specified (Significant effort went into ensuring that controlled drilling and blasting designs were followed and the results were recorded. Designs were revised to improve upon performance);
- construction and technical team meetings with the Contractor on a regular basis to improve communications between the groups, plan activities, monitor performance, resolve issues and coordinate construction and geotechnical activities;
- project management controls, including underground survey control, inspection and quality requirements and contract change control were formalized;
- a bonus incentive system that recognized project requirements and the efforts of the shaft sinking crews was implemented; and

- an enhanced safety program that provided site training, incident and accident reporting, an occupational health and safety committee and formalize mine rescue training and support was implemented.

2.2.6.1 Galloway Stage

A Galloway stage (Figure 6) was designed to accommodate the Contractor's construction activities and AECL's geotechnical characterization activities as efficiently as possible. It was necessary to design the stage specifically for the 4.6-m-diameter lower portion of the URL shaft. The stage consisted of five decks, each with a centrally located bucket well that allowed the sinking bucket to pass through the decks.

The stage was suspended on three 29-mm-diameter steel ropes in double fall and could be moved at about 1.5 m/sec. Three sheaves were mounted on the top deck (Deck No.1) to provide three-point suspension for better stability. Tail ropes were dead-ended at suspension rope anchors located in the upper rectangular section of the shaft. A pneumatic tugger was also mounted on Deck No. 1 to raise and lower the Cryderman mucker and to accommodate take-up spools for electrical power and a blasting cable. The blasting cable allowed blasts to be fired electrically from surface after all personnel were cleared from underground.

The top deck (Deck No. 1) provided overhead protection for personnel on lower decks and was used to install steel sets in the shaft. Deck No. 1 had provisions to chair the crosshead, but this system turned out to be unnecessary because the stretch of the hoist rope was not significant enough to require chairing.

Both Decks No. 1 and No. 2 were designed so that electrical and mechanical maintenance could be carried out on the two drill jumbos and Cryderman Mucker positioned on these decks. The Cryderman mucker and drill jumbos were mounted on rails so they could be lowered into operating position and retracted to provide the clearance required for research activities on the two decks below.

Working decks 4 and 5 were equipped with aluminium cover plates for the bucket well and other openings required for the Cryderman and drill jumbo setups. The working decks were suspended from Deck No. 3 on five 4-m-long telescoping bars and could be retracted during drilling, blasting, and mucking activities on the shaft bottom. The decks were lowered and retracted with the sinking bucket. When the decks were retracted, they were fastened to the underside of Deck No. 3 with three safety hooks or catches.

An unobstructed view of the shaft walls was required to accommodate stereo-photography of the shaft walls. Stereo-photography was carried out on Deck No. 5 while mapping was carried out simultaneously from Deck No. 4. Since the photographs were used as the base for mapping, the distance between the two decks was set to accommodate the turnaround of photographic prints in two days so that mapping work was sequenced with shaft excavation advance. (Note: Since high-quality digital cameras are now available, it would now be possible to use digitized photographs to assist with geological mapping and synchronize stereophotography, mapping and shaft excavation advance). These decks could also be used for sinking activities, such as scaling, rock bolting and screening. The decks were equipped with a railing to protect personnel from falling as well to serve as a grid for stereophotography.

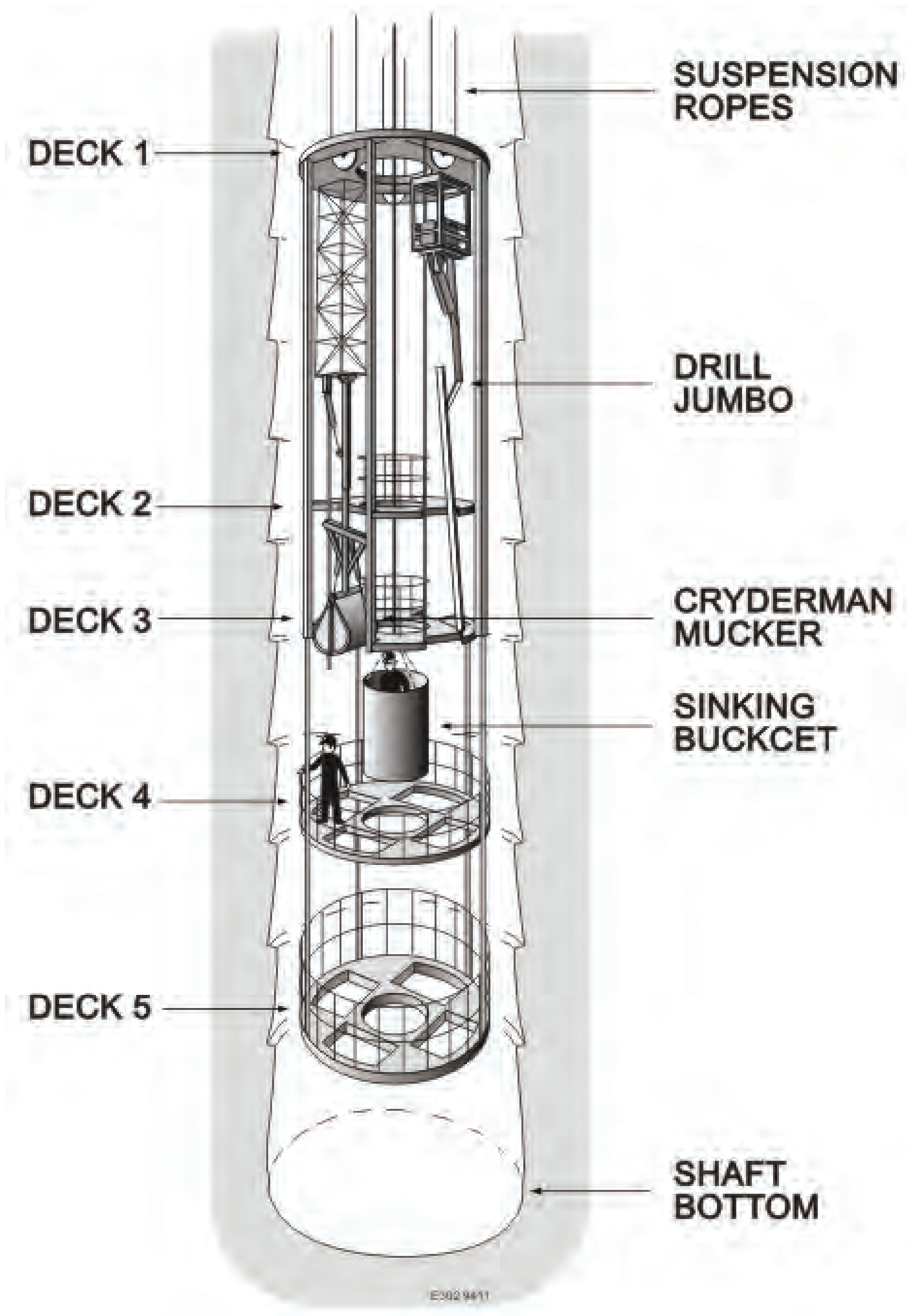


Figure 6: Galloway Stage Used in URL Lower Shaft Extension Excavation

2.2.6.1.1 Galloway Stage Operation

The two working Decks (No. 4 and No.5) were equipped with five horizontal screw jacks to align and stabilize the Galloway stage when it was set up for geotechnical activities (Figure 7). The Galloway stage could be positioned accurately and quickly during setup, using elevation markers installed at 2-m intervals on the shaft wall and two laser beams aimed down the shaft.

Pneumatic jacks, located on the upper three decks, facilitated rapid but less accurate setup during the shaft sinking cycle.

Figure 7: Screw Jack Stabilizer for Galloway Stage

The underside of the bottom Deck No. 5 was lined with about 15 cm of timber to protect it and the other decks from any fly-rock resulting from blasting operations.

Shaft sets were always maintained to within 60 m of the shaft bottom and usually were kept within 40 m. The Manitoba Mining Regulations required the sets to be maintained within 15 m of the shaft bottom; however, an exemption was granted to accommodate the Galloway stage. Shaft sets were advanced frequently to minimize the slow-speed travel zone required below the crosshead chairs usually located at the last set[1]. The chairs were advanced with the set installation. The Galloway stage could be positioned at any location below the last set. This allowed frequent inspection of the shaft walls, which enhanced the geotechnical activities and improved safety in the shaft below the permanent sets.

The stage winches were connected to standby diesel power so the Galloway stage could be moved during a power failure. A ladder-way was permanently installed between Deck No.1 and Deck No. 3 and temporary ladders were placed between Deck No. 3 and shaft bottom (also

[1] Note of clarification: Since the crosshead runs on guides fixed to the shaft sets, it has to be “chaired” when it reaches the last set. Access to the decks of the Galloway stage and shaft bottom is by the sinking bucket, which is configured to continue down the shaft at restricted speed below the chaired crosshead. The crosshead could also be chaired at the top deck of the Galloway stage at permanent setups, such as instrument arrays, providing the Galloway stage was positioned immediately below the bottom set.

between the working Decks No. 4 and No. 5 when extended) to provide access between decks for personnel.

Temporary and permanent services, such as power and data cables, compressed air and water lines, etc., were mounted on the shaft wall. The top three decks were built to provide sufficient clearance for the services and mounting brackets. Openings were provided to allow the ventilation ducting to pass through each deck and continue to the shaft bottom.

2.2.6.1.2 Galloway Stage Vacuum System

To assist with the cleaning of the shaft bottom to concrete placement standards, as required by the shaft bottom mapping in the geotechnical characterization program, a shaft vacuum system was built and commissioned specifically for testing on the shaft extension project. Although the technology was new, it was envisioned that a vacuum unit would speed up the shaft bottom cleanup and provide better access for geological mapping of the shaft floor.

The unit, called a Trans-Vac Suction system, was in the process of being introduced in the mining industry to assist mine operators in sweeping old and current mining areas, cleaning up backfill spills, cleaning shaft bottom and recovering precious metal participates from the bottom of mined out stopes. Figure 8 is a picture of the unit.

Figure 8: Vacuum Unit for Shaft Bottom Cleaning

The vacuum unit comprised a cyclone fan driven by a 75 kW electric motor on top on of a two-cubic metre hopper (the shaft sinking bucket comprised the hopper). The cyclone delivered

50 m^3/min of free air producing a suction capacity of 60 kPa. A 200-mm-diameter hose and nozzle with a compressed air venturi was effective at picking up dry or wet muck (up to 80% moisture content) and rock fragments up to 150 mm in diameter. The cyclone was powerful enough to lift the broken material up to 70 m vertically.

The shaft vacuum was tested on several shaft blast rounds and found to be very effective at completing the final cleaning of the shaft bottom after the broken rock from the blast had been removed with the Cryderman mucker. The final cleaning could be completed in about 30 minutes, which was faster and more effective then the conventional method with a blowpipe, shovels and water suction pump. However, there was not sufficient room for mounting the unit and permanent hopper on the Galloway stage. The unit had to be brought down to the shaft bottom from surface and returned after each use. This added an additional 45 minutes to the final cleaning. Reverting to the conventional method of shaft bottom cleaning saved time.

It was concluded that the shaft bottom vacuum technology would be more appropriate for use in a larger diameter shaft and with a modified Galloway stage configuration.

2.2.6.1.3 Galloway Stage Drill Jumbos

Drill jumbo boom assemblies were designed and built specifically for the shaft extension project. The shaft area was relatively small for a Galloway stage and the drill jumbos had to be designed to use a minimum amount of the area available. Since the bucket traveled in the centre of the shaft, it was not practical to mount a multi-boom jumbo centrally on the Galloway stage. Therefore, two self-retracting single-boom jumbo assemblies (Figure 9) were mounted on opposing sides of the bucket well, each capable of drilling half of the round.

Figure 9: One of Two Drill Jumbos Used in Shaft Excavation

The jumbo boom cylinders were hydraulic, and each boom was equipped with an electric-hydraulic power pack located at the top of the tower assembly. Tamrock L550 pneumatic drills were mounted on 3.5-m-long chain feed shells. The booms were equipped with a system that provided parallel holding in one plane. A clinometer was installed on each boom to locate the correct lookout angle on the perimeter holes.

2.2.6.2 Controlled Drilling and Blasting

The size selected for the production blast holes was 38-mm-diameter, in order to provide the best explosive energy distribution throughout the blast round. The 38-mm-diameter was the smallest diameter possible for a threaded drill string. Changing to better quality drill steel solved initial drill rod failures. Generally, the drilling accuracy was good, due primarily to the design of the drill jumbos and the diligence of the drillers and shift inspectors.

The blast design for the full-face round used to excavate a circular shaft configuration is shown in Figure 10, Figure 11 and Figure 12. The full-face method used a burn cut located within the first of four rings of blast holes in the round, see Figures 10 and 11. The cut consisted of three blast holes and three 90-mm-diameter relief holes drilled to a depth of 600 mm below grade (Figure 11). This sub-grade drilling improved the ability of the blast of each cut-hole charge to break cleanly to the relief holes, and it also provided a sump at the shaft bottom, which facilitated cleanup and water handling during drilling of the next round.

Round lengths could be varied between 1 m and 3.5 m by adjusting the feed shells on the drill jumbos. Although round lengths up to 3.5 m were successfully blasted, most rounds were drilled 2.3 m, which was the optimum length for the 16 hours available to the Contractor each day (Figure 11). When additional information was required for geotechnical characterization, e.g., while excavating through an instrument array, round length could be shortened to provide more data points and geological information on rock mass response as the shaft bottom was advanced. Round lengths as low as 1 metre were excavated for this purpose.

To avoid drilling near bootlegs, each round pattern was rotated slightly so that the new cut was offset relative to the previous cut. Water was blown out of the relief holes by a small charge place on the bottom of each relief hole and initiated on the first delay period, No. 0.

The cut was designed to reduce the possibility of sympathetic detonation. When sympathetic detonation occurs, as is common with gelatine dynamites, the intended blast hole initiation sequence is overridden. This causes increased vibration, and greater potential for overbreak and fly rock. Increased fly rock increases the potential for damage to sinking equipment. Selecting the correct firing order and locating the later firing charges behind the relief holes helped to reduce or eliminate the effect of sympathetic detonation in the cut.

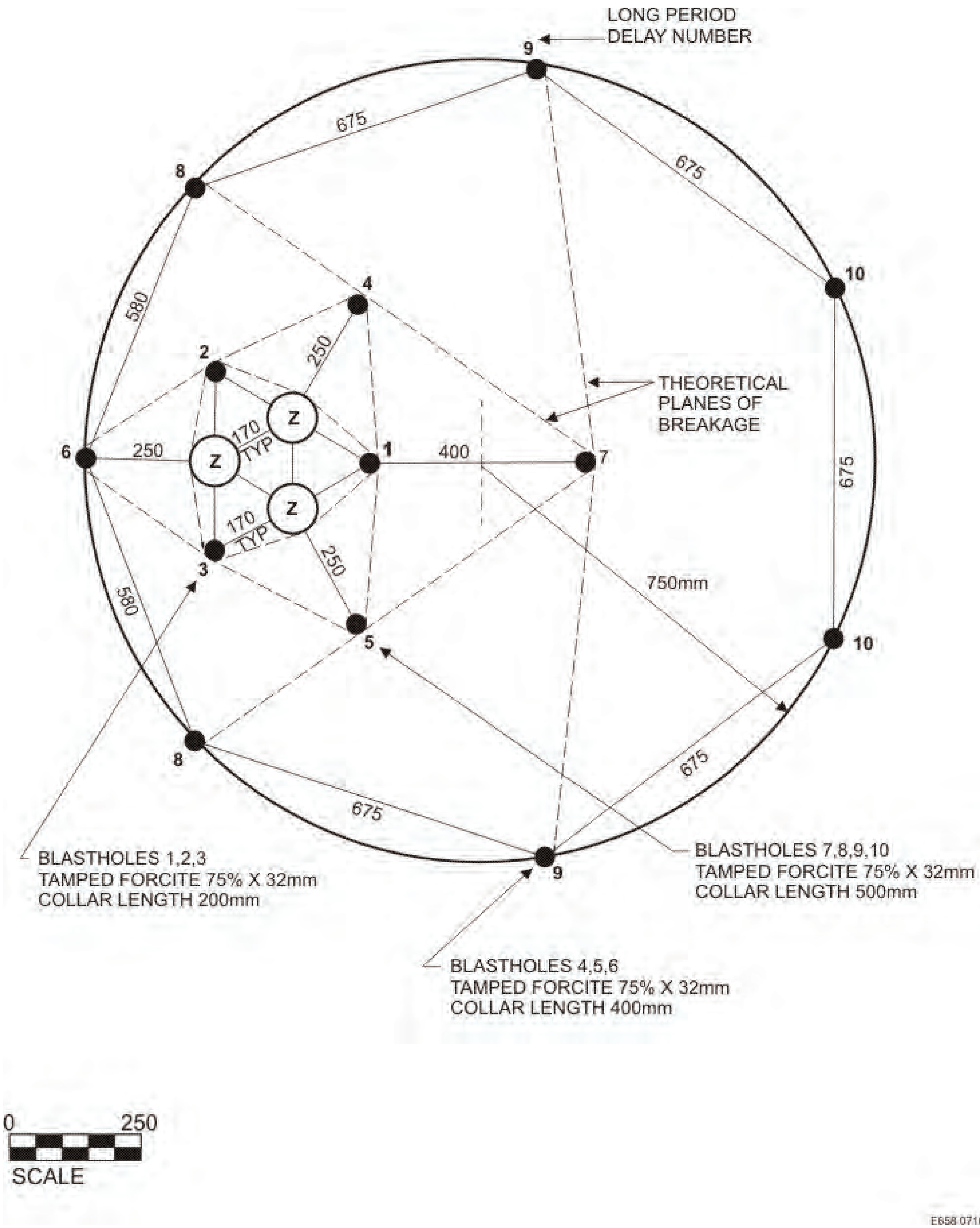


Figure 10: Cut Layout for the Full-Face Blast Design

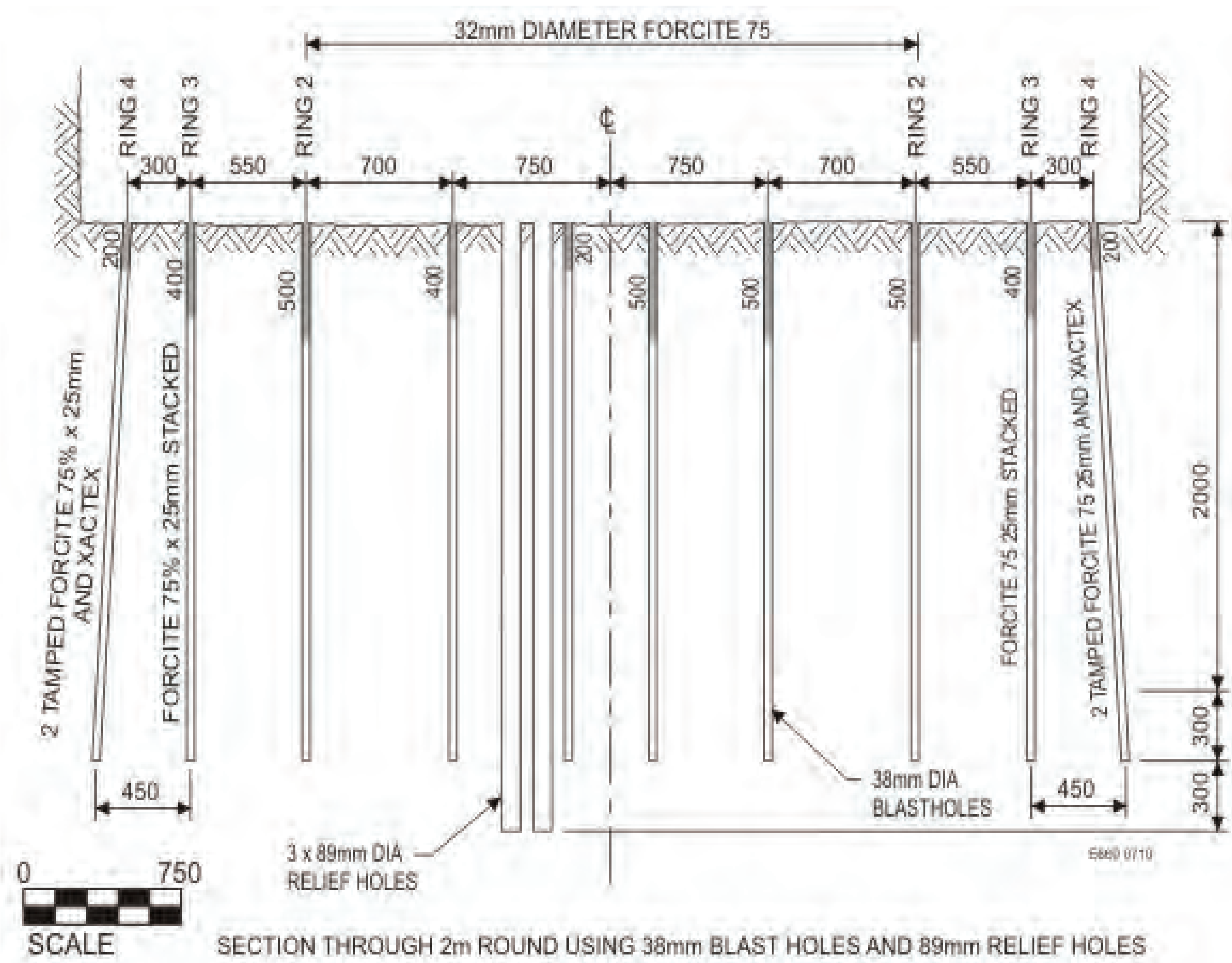


Figure 11: Profile of the Circular Shaft Blast Round Configuration

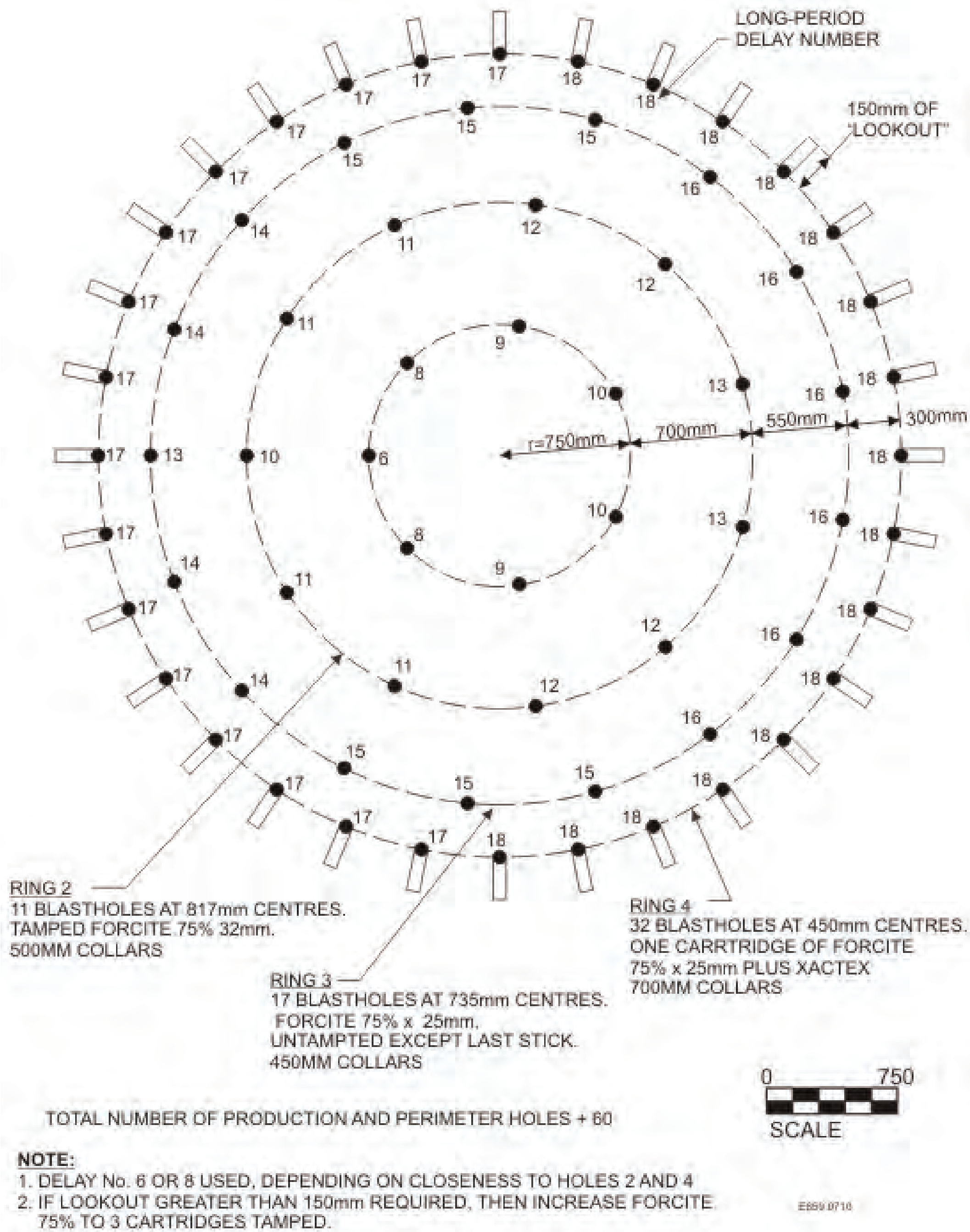


Figure 12: Full Face Blast Design Used in the Lower Shaft Sinking Contractor Showing Delay Allocation for Rings 1 to 4

The potential for flyrock damage to the Galloway stage was great because the sets were generally maintained as close as possible (40 m) to the shaft bottom to improve the mucking time. This placed the bottom deck of the Galloway stage as close as 15 m from the shaft bottom. To minimize damage caused by flyrock, the blasts were designed with 400- to 500-mm collars in the production holes located within rings 1 and 2. The cartridges were tamped for this reason and to optimize powder distribution within the round. Holes were stemmed with duct seal to help keep the explosives in place during the blast.

For rings 3 and 4, the charge density was varied by tamping, so more explosive energy was available at the toe of the holes where it is difficult to break out the rock. This loading procedure helped to minimize overall blast damage to the final walls of the shaft.

The blast design was based on blast holes being drilled in four concentric rings (Figure 12). All blast holes and relief holes were drilled as close as possible to vertical, with the exception of the perimeter holes where a lookout was required to maintain the shaft diameter.

The 73 blast holes were distributed as follows.

- 13 blast holes in Ring 1 and the cut.
- 11 blast holes at 817-mm centres in Ring 2.
- 17 blast holes at 735-mm centres in Ring 3 (the cushion ring).
- 32 blast holes at 459-mm centres in Ring 4 (the perimeter ring).

Blast holes in Rings 1 and 2 were charged with Forcite 75% in 32 mm x 400 mm cartridges. Blast holes in Ring 3 were charged with staked Forcite 75% in 25 mm x 200 mm cartridges, every third cartridge being tamped. Blast holes in the perimeter ring were charged with three cartridges of tamped Forcite 75% in 25 mm x 200 mm cartridges while and the remaining length was charged with 19 mm x 600 mm Xactex. The blasting products were supplied by Canadian Industries Limited (C-I-L). Forcite is a high-strength ammonia gelatine dynamite commonly used in production blasting at the time. Xactex, which was also an ICI explosive product, was formulated for perimeter blasting application. Both products were nitro-glycerine-based explosives.

Long-period Magnadet detonators with 4-m-long leads initiated each charge. The delay allocation for the full-face shaft rounds had the following beneficial features.

- One hole per delay in the cut for greatest assured relief.
- The minimum practical number of dependent charges on consecutive delay numbers.
- A charge mass per delay, which increases with increasing delay number and with decreasing tightness (confinement) of charges.
- As the perimeter is approached along any radial line, the dependent charges are detonating on every second delay numbers.

Because pre-splitting was unsuccessful in moderately to highly stressed confined rock, in was necessary to apply a smooth wall blast technique in which the lightly charged perimeter blast holes detonate after all of the production blast holes. Ideally, all such perimeter blast holes should detonate simultaneously, but this was not possible with the detonators available at the time, e.g., the Magnadet system. The Magnadet detonators exhibited appreciable scatter, i.e., delay time variability, especially for the longer delay periods. The manufacturer increased the delay interval to minimize the possibility of overlap. Therefore, the total duration of a blast was much longer than would have been needed for detonators having zero scatter. Also, since the blast designers were aware of the deleterious effect of possible overlap and crowding, they often placed charges on every second delay rather than consecutive delays.

It was expected that electronic detonators, which were an emerging technology at the time, could provide more accurate timing, thus eliminating the problem of overlap and crowding. During the latter part of the shaft extension project, electronic detonators were provided by C-I-L Inc. for testing. Two blasts fired with electronic detonators indicated that very accurate delay time could contribute appreciably to the degree of success of advancing long full-face shaft rounds.

The blast rounds for the lower shaft are summarised in Table 7.

Table 7: Comparison of Full-face Blast Rounds in the Lower Shaft

Item	Description	Units	Advance (Metres)				
			<1.5	1.5 – < 2.0	2.5 – < 2.5	2.5 – < 3.0	> 3.0
1.	No. of Blasts in Range		15	22	37	2	10
2.	Average Length Drilled	m	1.36	2.18	2.33	3.01	3.49
3.	Surveyed Advance	m	1.2	1.88	2.11	2.71	3.28
4.	Surveyed Volume	m^3	21.01	33.26	37.61	48.39	62.02
5.	Average Explosive Mass	kg	66.70	109.78	113.90	153.23	192.64
6.	Powder Factor	kg/m^3	3.16	3.14	3.0	3.17	3.32
7.	Percent Pull	%	89.84	86.12	90.70	90.04	93.97
8.	Percent Half-barrels	%	20.16	33.98	36.10	25.86	29.12
9.	Percent Overbreak	%	5.36	6.68	7.28	7.25	13.81
10.	Overbreak	mm	123	154	167	167	318

Note: Reblasts, Flattenings and Sump blasts not considered in this analysis.

2.2.6.2.1 Controlled Drilling and Blasting Performance

It was decided early in the shaft extension project to monitor five of the full-face blasts in order to evaluate the blast design and the effectiveness of the full-face sinking method. Blast vibration monitoring allowed the active detonation sequence to be evaluated, thereby enabling determination of the time at which each delay period actually fired, detection of any out-of-sequence firing order, measurement of actual vibration levels, and blast design optimization by identifying the number of explosive charges that actually contributed to the effective rock breakage. The equipment used at the URL included geophones connected to a Racal FM tape recorder and a Norland waveform analyzer. The geophones were located in four short drill holes, 100-mm long and 63-mm diameter, positioned about 20 m from the face.

The mean yield of explosive energy per metre of charge length should be low in the perimeter blast holes to limit damage to the surrounding shaft wall rock. Since the presence of water precluded the use of ANFO/polystyrene-type mixtures, decoupled charges of water-resistant cartridge-explosives were used. Overbreak and damage to the walls of the shaft decreased as the charges became increasingly decoupled. In practice, however, the blast designer had to select a charge concentration that could be realised from a limited number of perimeter explosives available at the time, e.g., Xactex or Primaflex. The shaft wall quality was dictated not only by geometry, charge concentration and detonation sequence of the perimeter blast holes, but also by the design of the earlier firing production blast holes, irrespective of the design efforts and care taken with the perimeter blast holes. A poorly designed cut and/or production ring could start a train of events that could result in an increased overbreak (Favreau et al. 1987).

Analysis of the data obtained from the monitoring program revealed that seven or eight charges out of 70 charges in each of the five blasts monitored did not detonate with full power, suggesting that about 10% of the explosive energy was wasted. This observation resulted in minor modifications to the number of blast holes, charge density in certain blast holes and the burden and spacing between blast holes and blast-hole rings. However, in general, blast monitoring indicated that out-of-sequence detonations, sympathetic detonations, misfires and high vibration were not creating problems for the performance of the blasts. Results of the blast monitoring are discussed by Hagan et al. (1989) and Mohanty et al. 1990.

Generally, the cut configuration worked effectively with each of the three cut charges detonated and contributed to the breakage process. Helper charges adjacent to the cut (blast holes No. 4, No. 5 and No.7 shown in Figure 10) were damaged in two blasts, indicating that the design spacing between the cut and the helper charges was too close. The most consistent observed blast malfunction was associated with misfired production charges in the Ring No. 1 and No. 2. The evidence indicated that either the burden distance or blast hole spacing was smaller than optimum. However, these first rings consistently broke to grade in all five blasts, suggesting that the number of blast holes in Ring No. 2 could possibly be reduced by one or two.

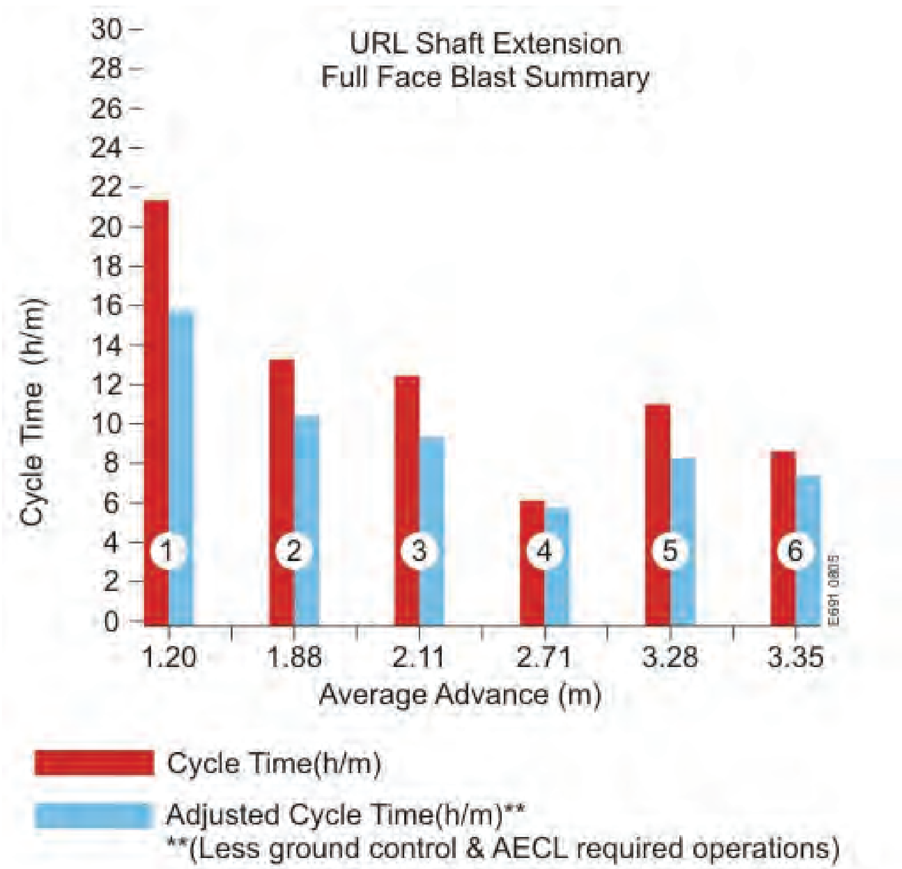
The statistics from one of the better trial blasts (Blast No. 63) were as follows.

- Average blast hole length – 2.31 m.
- Surveyed average advance – 2.20 m.
- Percentage pull – 95%.
- Volume overbreak – 6.5 %.
- Percentage half-barrels – 49%.
- Powder factor – 2.94 kg/m^3.

The key to the success of the blast designs was the triangular cut geometry that had been designed to minimize the effect of sympathetic detonation and dynamic pressure desensitization (Hagan et al. 1989). The high delay numbers of the long-period detonators exhibited delay time scatter and therefore ultimately led to the use of every second, rather than consecutive, delays for dependent charges.

The total number of full face excavation rounds is summarized in Figure 13. The number of rounds for each excavation length is shown.
Blast performance, particularly for the cut and around the perimeter of the blast round, depended upon drilling accuracy. The specifications laid out for the Contractor's crew included:

- drill hole collars within 100 mm of the design location;
- drill hole alignment within 100 mm of the intended design location when measured from the drill hole collar;
- lookout on the perimeter holes not to exceed 150 mm; and
- cut and production holes kept parallel and as close to vertical as possible.



Legend

Item	Round Length	Number of Blasts
1.	<1.5m	15
2.	1.5 to 2.0m	22
3.	2.0 to 2.5m	37
4.	2.5 to 3.0m	2
5.	>3.0m	10
6.	Best Week	4

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Figure 13: Summary of Lower Shaft Full Face Excavation Rounds

As controlled drilling and blasting was not effective in the upper shaft where the benching method was used, the full-face blasting method was used in the shaft extension:

- so controlled drilling and blasting could be used effectively to limit blast damage and maximize the quality of the rock walls;
- so an optimum round length could be cycled each day within the two shifts available to the sinking crews; and
- because the slightly concave face produced by the full-face blasting provided a better geometry for stability at the shaft bottom. This geometry was more desirable for the geotechnical program that included applying numerical models to simulate the behaviour of the rock mass around the shaft.

The burn cut was considered to be superior to the alternative angled-hole cut, or V-cut, for shaft excavation because V-cuts tend to result in poorly fragmented rock and can eject the rock violently for a considerable distance, with a large chance of damaging the shaft installation and sinking equipment. Thus the burn cut was used in the shaft extension.

2.2.6.3 Project Control and Inspection

It was anticipated that the cost plus fixed fee contract selected for the project would require more administrative effort than the conventional unit price contract used for the previous shaft-sinking contract. A Contract Administrator was retained to help AECL interpret the contract, control the project and monitor trend reports and change orders, and to help with scheduling.

Project control comprised the preparation and maintenance of logical work schedules and detailed cost estimates prior to commencement of and during the work. Project management software was used to schedule and track work. AECL retained prime responsibility for overall cost and schedule monitoring. These work schedules and cost estimates were updated during the project to reflect changes in schedule or cost.

All changes to the schedule and cost estimate were justified and recorded. Overhead costs, such as those incurred from support services, inspectors, researchers assigned to the project, and URL operating expenditures were the most significant portion of the shaft extension project costs. Thus ensuring the excavation and geotechnical characterization program proceeded smoothly resulted in cost savings. An Unscheduled Delay Log was used to record the duration and reason for unexpected delays to the Contractor’s work and the technical activities on a shift-by-shift basis. The log was critical for resolving contractual delay issues and was reviewed for agreement between the Contractor’s on-site supervision and AECL on a weekly basis.

Quality control and inspection procedures for the shaft extension project were developed to ensure the Contractor carried out the work as intended. To achieve quality control and inspection objectives for the shaft extension project, a project management team was organized to ensure that support services and non-resident engineering were available as required until the project was completed (Figure 5). The resident engineer controlled daily shaft extension activities, giving particular attention to coordinating and integrating the construction activities with the geotechnical program. Thus, all the requirements of AECL’s geotechnical research and development program were met, and the project was maintained on schedule and budget. The integration of the research and construction activities is described by Kuzyk et al. (1986a).

As noted earlier, AECL recognized that Canadian mining Contractors had limited experience with full-face shaft sinking using a burn cut. Therefore, AECL's project management organization assumed responsibility for blast round design, performance and inspection in the shaft extension. The Contractor assumed responsibility for implementing the blast design according to the AECL design. A blast design consultant was brought on board during the initial stages to support AECL staff in preparing and implementing the blast designs. The consultant remained on the project long enough to ensure that AECL blast design engineer and inspectors were trained in the full-face method. The AECL staff then assumed responsibility for blast design and inspection.

The requirement for a ground control consultant was initially questioned because of the minimal degree of ground support required during upper shaft sinking. However, for the lower shaft, the high stresses experienced below FZ2 (273 m depth) necessitated more attention to ground control, and a ground control consultant was retained on a service contract (Onagi et al. 1988).

With good inspection procedures and documentation, the AECL project management team was able to achieve an acceptable level of quality control on the shaft extension project, as demonstrated by the controlled drilling and blasting work shown in Figure 14. When necessary, deviations from design were brought to the attention of the project management team, who directed the sinking Contractor to make the necessary corrections through the change control process. AECL's project management team developed the activity record forms (see Appendix A) and quality control procedures. Commissioning and as-built inspection forms developed for the upper shaft sinking contract were adapted for the lower shaft sinking project.

Four shift inspectors were necessary to provide shift inspection coverage seven days per week for the two shifts between 16:00 hours and 8:00 hours that the Contractor's crews carried out sinking activities. Contracted engineering firms provided two inspectors who were experienced in underground mining activities and in dealing with shaft sinking Contractors. The remaining two inspectors were AECL staff experienced in the geotechnical programs and who had previously worked during upper shaft construction. Once shaft furnishing commenced, two contracted inspectors carried out electrical and mechanical inspections.

The shift inspectors did not have direct authority over the Contractor's crews. When the inspector identified a deviation in the design, the procedure was to advise the Contractor's shift supervisor. If the shift supervisor disagreed with the inspector, the resident engineer and the sinking Contractor's manager resolved the issue. This procedure avoided any conflict between the inspectors and Contractor's crew at the work face. In the course of the project, the Contractor's shift supervisor rectified most deviations without referring the issue to the resident engineer and the sinking Contractor's manager.

2.2.6.4 Coordination Meetings

Coordination and control of the contract and construction activities was achieved through the application of a variety of project management methods. The most important of these was the establishment of regularly scheduled meetings, which enhanced communication within the project organization.

- Daily morning meetings were held shortly after 8:00 hours, immediately after the shaft was turned over to the geotechnical research group. These meetings, generally lasting from 20 to 30 minutes, involved the resident engineer, shift inspector, sinking Contractor's project manager, and the geotechnical characterization group coordinator. The meetings reviewed the progress from the previous day, identify any problems or delays, and plan the activities for the following day with knowledge of other activities being planned, support arrangements for the following shifts, delays to the geotechnical program and/or sinking crews, revisions to procedures and/or design, equipment problems, servicing geotechnical staff working on the 240 Level, shaft turnovers, safety issues, watering of the shaft timbers, and optimization of round length in the 16-hour sinking cycle. Senior project management staff worked on day shift and were available for consultation on critical issues, if necessary.

- Weekly project meetings were to discuss issues or problems that persisted during the preceding week. The meetings were generally held on Thursdays so that work plans could be prepared for the weekend. They were 1 to 2 hours long and involved Contractors, specialized consultants when required, and staff from AECL's characterization and construction teams. Weekly meetings generally focused on the finalization of plans made during the monthly meetings, follow-up to ensure the availability of materials and supplies and the commitment of resources. The project schedule and objectives were reviewed and plans were laid out for the coming week. Frequently, changing in situ conditions necessitated "last minute" revisions to the plans laid out at monthly meetings. The weekly meetings provided enough advance notice to accommodate these changes. Minutes of the meetings were distributed and served as a record of decisions and a reminder of the plans made.

Figure 14: Controlled Drilling and Blasting Use to Provide Smooth Rock Surface for Geological Mapping

- Monthly site construction meetings involved senior management from both the Contractor and AECL's project staff. The project status was reviewed at these meetings, and major issues were discussed. Problems relating to contract administration and change in scope of the project were resolved. Reports on schedules, characterization program plans, project expenditures, the long-range variance against the project schedule and budget, safety, pre-purchased equipment status, etc., were made by key project management personnel. These monthly site construction meetings were successful in providing co-ordination and control of the overall project and served to keep the project management team up-to-date and aware of project objectives, even when a change of scope was imminent. The meetings were normally 1 to 3 hours long.

- Monthly characterization coordination meetings to plan and integrate the multi-disciplined characterization activities were also carried out. These involved staff from the characterization and construction teams and were normally 2 to 3 hours long. The focus was on detailed planning of work, instrumentation and data collection with respect to the characterization program activities in the shaft.

2.2.6.5 Survey Control

Accurate survey control was essential for the construction and characterization of the URL. The accuracy of field data obtained during the characterization program was directly related to the accuracy of the surveying done to identify test borehole collar locations, azimuths and plunges, and the survey points were used to locate features on excavation walls during geological mapping and photography operations. Surveying errors and inaccuracy can eliminate many benefits associated with careful field measurements of dip and dip direction of features found in diamond drill core and on excavation walls. The overall survey closure accuracy of $\pm$10mm was achieved. Borehole plunge and azimuth accuracy of $\pm$5% were achieved and this accuracy is adequate for most applications.

Survey control at the URL primarily comprised the location of control points for construction, excavation, ground support and the drilling of boreholes. Prior to commencing excavation work or the drilling of boreholes, a construction drawing illustrating the excavation or borehole design plan and section on a Mechanical Engineering (ME) drawing was prepared and approved under the direction of AECL's construction management group. This drawing was then issued to the excavation and/or drilling Contractors, or AECL crews, responsible for carrying out the work. Survey control points were installed and maintained to control excavation alignment and grade. Borehole alignment, azimuth and plunge control points were installed for each drill setup and the drill string orientation was checked after the hole had been collared. Survey data were stored during field activities and down-loaded into the URL CADD system data base for the preparation of detailed 3-D models of excavations, boreholes, geological information and structures. The approach taken is described by Chernis and Karklin (1994). The URL CADD system is described by Chernis (1993).

Warning notices were issued to notify Contractors and crews of pending breakthroughs into other excavations or boreholes during mining operations.

Specific survey control points for geotechnical mapping and other technical group activities were provided and as-built excavation and borehole surveys were completed to provide an accurate record of locations and configuration geometry.

2.2.6.6 Contract Change Control

A change control procedure was developed to provide a formal mechanism for considering potential changes in scope. Any project participant could initiate the change control procedure by submitting a trend report to the construction team's manager or resident engineer. The trend reports were one-page forms that contained the:

- originator's name and group;
- date;
- description of potential change;
- reason for change;
- cost impact (estimated or actual); and
- schedule effect.

Trend reports were not only intended to document adverse variances, but also to identify opportunities for cost saving and schedule improvement. When a trend report was received the construction and characterization teams reviewed the report to determine if the desired changes constituted a change in contract scope and was appropriate for the characterization program. If it did, a Change Order was approved by AECL and issued to the Contractor, resulting in a change to the "committed costs and schedule" defined in the contract.

2.2.6.7 Bonus Incentive System

In Canada, shaft-sinking crews are customarily paid an hourly-rate incentive bonus for performing above an expected or predetermined level of performance. It would have been very difficult, if not impossible, to attract experienced shaft-sinking miners to the shaft extension project without a competitive bonus incentive plan. A bonus plan was therefore set up for the shaft-sinking crews, with the bonus being controlled by the AECL project management team. Unlike the bonus plan used by the shaft-sinking Contractor in the upper shaft sinking project, the crew bonus in the lower shaft excavation was based on three criteria identified as the most significant objectives for the crews working on shaft bottom.

1. Quality of drilling and blasting work – following AECL approved designs.
2. Schedule objectives – achieving specified bi-weekly milestones.
3. Adequacy of shaft turnovers – quality and timeliness of the 08:00-hour turnover for characterization activities.

Shaft extension advance relied on breaking each full-face round to the full depth drilled. Blast designs had to be very closely followed by the crews to achieve the desired results; particularly during the early blast rounds while designs were being optimized. The blasts were designed to produce high-quality walls, with minimal damage to the surrounding rock mass. The face had to break with minimum bootleg, so that there would be a clean face for drilling the next round. Achievements in this area were referred to as quality of work, and a 40% portion of the crew bonus was awarded on the basis of the work done by the crews to achieve the desired design standard.

The project schedule was reviewed each two-week period with the Contractor's supervisor, and objectives for the coming period were laid out. Objectives were reviewed with the crews so they understood what was planned and expected. Achievements in this area were referred too as schedule objectives and a 30% portion of the crew bonus would be paid in full if the scheduled objectives were achieved. An improvement or reduction in schedule objectives resulted in a pro-rated adjustment in the portion of the bonus assigned to achieving the schedule objectives. When the schedule was exceeded the bonus was increased above the total bonus for achieving the schedule. This approach provided the crews with an opportunity and therefore incentive to catch up on any delays experienced during the project.

As part of the adequacy of shaft turnover objective, it was also necessary to have the face cleaned to concrete placement standards before the shaft turnover at 0800 hours each morning in preparation for the geotechnical program. Many tasks had to be completed in preparation for the geotechnical characterization activities carried out during each dayshift. Failure to complete part of the turnover resulted in delays to both the geotechnical activities and the overall project. A final 30% portion of the crew bonus was allocated for achieving effective turnovers in a timely manner, with increasing amounts for each day a satisfactory turnover was achieved.

2.2.6.8 URL Safety Program

To meet the objectives of the geotechnical characterization program during the shaft extension, it was necessary for AECL geotechnical personnel to work in the shaft. There were only a few individuals who had any previous shaft-sinking or underground mining experience. Also, the unfamiliar full-face blasting method utilizing a burn-cut was being introduced along with new equipment, e.g., a Galloway stage designed to integrate geotechnical characterization and shaft sinking activities, presented new concepts even for experienced shaft miners. It was therefore very important to set up good safety procedures and indoctrination programs for the AECL employees, AECL-contract staff and the shaft sinking Contractor's workers involved. Part of this responsibility was assigned to the shaft-sinking Contractor in Contract No. 8. Prior to starting the lower shaft construction project, the shaft-sinking Contractor prepared a safety manual titled "The Essentials of Safety During Shaft Sinking Operations" with information on orientation/indoctrination for existing and new personnel, regulatory requirements, role of the Health and Safety Committee, role of the mine rescue team, incident reporting, Workplace Hazardous Materials Information System (WHMIS), shaft sinking Contractor's safety rules, first aid procedures, protective clothing and footwear, used of safety belts and lanyards, check-in check-out procedure, traveling in the bucket, shaft signals, working on and operation of the Galloway stage, working on shaft bottom, climbing ladders, ventilation, refuge station procedures, and many other items concerning the project. A safety seminar lead by the AECL's resident engineer and the Contractor's safety specialist was held for all URL personnel once the manual was available.

The URL employee orientation/indoctrination was a three tier process in which general site indoctrination, facility orientation and shaft and underground orientation were provided by the URL Fire Safety and Security Officer. Special conditions, such as working on the Galloway stage, were explained by a knowledgeable person from the shaft sinking Contractor's group. The employee's immediate supervisor provided a detailed job orientation. Each step of the orientation was documented and a completed employee induction form was signed by the employee, his supervisor, and the URL Site Supervisor and filed for future reference.

In 1987 June, the shaft-sinking Contractor assumed responsibility for operation of the underground facilities, including the hoist and shaft. At this time, the URL fire emergency and evacuation procedures were reviewed with the Contractor's staff. These procedures provided for immediate evacuation of the shaft bottom to surface, with provisions for evacuation to a refuge station at the 240 Level, should immediate evacuation to surface not be possible.

A mine rescue team, composed of trained volunteers, was available to support underground fire fighting and recovery operation. The team maintained training and equipment, including self-contained breathing apparatus, throughout the project. Arrangements for backup support for mine rescue recovery operations were made with the Manitoba Mining Association. Additional teams could be called in on short notice from other underground mining operations in Manitoba to assist the URL mine rescue team.

Once the shaft extension project got underway, a safety program was initiated and continued on a regular basis. AECL's security and safety officer indoctrinated new staff and carried out weekly surface and underground inspections. The sinking Contractor held safety huddles for the sinking crews and AECL staff working in the shaft.

The safety program included a procedure for reporting and following up of safety incidents. Under this procedure, all safety incidents, whether or not they resulted in an accident or injury, were reported on an incident report form. All shaft-extension project staff were encouraged to report incidents and recommend corrective action. The incidents were reviewed by supervisors and line managers to ensure appropriate corrective action was identified and taken. The corrective action taken was noted on the incident report form and passed on to the URL Health and Safety Committee for information and review at monthly meetings.

Incidents that occurred during the contract included the following.

- Oil and water caused slippery conditions on the Galloway stage working decks during diamond drilling. No injury resulted.
- Cover plates on one of the Galloway stage working decks came loose during drilling. No injury resulted.
- No safe tie-in for safety lanyards for personnel taking instrument readings from the shaft-sinking bucket. No injury resulted.
- A small piece of loose was detached from the shaft wall by the shaft sprinkler system. No injury resulted.
- Slippery ladder rungs on the Galloway stage. No injury resulted.
- An employee was struck by a piece of falling loose resulting in minor abrasions.
- A box of diamond drill core fell from one of the working decks on the Galloway stage to shaft bottom while it was being transferred to the sinking bucket for hoisted to surface. No injury resulted.
- A rock burst from the brow of the 420 Level shaft station struck a supervisor on the head and shoulder. No serious or lost time injury resulted.
- A nut and bolt fell down the shaft landing near a worker on shaft bottom. No injury resulted.
- A scaling bar fell down the shaft to the bottom working Deck No. 5 of the Galloway stage hitting a worker on the hand. No serious or lost time injury resulted.
- A worker received a hand injury when his hand caught between a core tube and drill bit on at the 300 Level shaft station. This incident resulted in twelve days of lost time.

- A piece of loose rock fell from the shaft wall hitting a worker and badly bruising his leg. This incident resulted in two days lost time.
- Cutting galvanized metal on the shaft sets resulted in noxious fumes being reported. Precautions regarding the use of appropriate gas filters and control of ventilation were implemented.

The URL Health and Safety Committee consisted of representatives from AECL's staff working at the URL and met twice each month. Every second meeting was open to all the shaft extension project staff. All incidents were discussed and action was taken to follow up on any further requirements identified by the committee. Minutes were kept for the meeting to record issues and corrective actions. Frequently, relevant incidents occurring at other underground facilities were presented and discussed. Frequently, guest speakers were brought in to give brief presentations on topics of interest.

2.2.6.9 Excavation Maintenance and Monitoring

When good overall rock conditions are anticipated, as was the expectation during the shaft extension project, it is not necessary to install rock bolts on a pattern. A design-as-you-go approach to ground control was therefore adopted. This approach required a ground control engineer to inspect the freshly exposed walls and face after each blast. Any changes in rock type, major lithologic boundaries, structure, jointing, or other discontinuities were identified and, if any discontinuities were found, measurements were taken and various joint parameters were determined. The rock mass was then classified using the NGI Tunnel Quality Index (Hoek and Brown 1980) to identify and record the joint characteristics with respect to other locations at the URL. The ground control engineer also received information from the geological mapping and characterization crews who worked at the face on the day shift. The construction inspectors recorded the approximate size and location of loose material scaled off the shaft walls by the shaft sinking crews during the other shifts. This information was used to identify and design an appropriate ground control or ground support system for any problem areas. Ground control or support was then installed before the excavation advanced beyond the problem area.

Ground control and support procedures were prepared for the project staff and included the following.

- Ground control evaluation.
- Type I rock bolt specifications.
- Type II rock bolt specifications.
- Rock bolt instruction to Contractor.
- Screen installation record.
- Scaling record.
- URL Shaft Extension Excavation Log.

In case a structural ground support problem, such as an unstable wedge, was identified, a rock bolting pattern was designed by the ground control engineer using the "UNWEDGE" computer program (Hoek and Brown 1980). Stereographic techniques were used to determine mass, sliding direction, and magnitude and direction of force to be applied to stabilize the wedge. Once ground support measures were designed, instructions were issued to the Contractor and the AECL shift inspector. The AECL shift inspectors monitored the installation to ensure design specifications were followed.

Since the URL was developed to the 255-m depth with only a very minimal degree of ground support, it was felt initially that the shaft extension could proceed with minimal support as well. However, once the shaft extension excavation was underway; it became apparent that ground control measures, in the form of screening (i.e., wire mesh anchored to the walls with 0.6-m-long mechanical anchor bolts) on the rock walls, was required as protection for personnel. This was required primarily to deal with minor bursting and spalling of loose rock off the shaft walls, which was a recurring problem. For worker safety, screening and anchor bolts were installed to within 5 m from the shaft bottom at all times during the excavation cycle.

Elliptically-shaped openings are more stable than rectangular-shaped openings in rock that is subject to a high ratio of in situ stresses acting on the cross sectional opening. Therefore, the 300 Level and 420 Level stations were designed with curved walls and crowns to better accommodate the high stress ratios encountered. Figure 15 shows the 420-m-deep shaft station plan and profile.

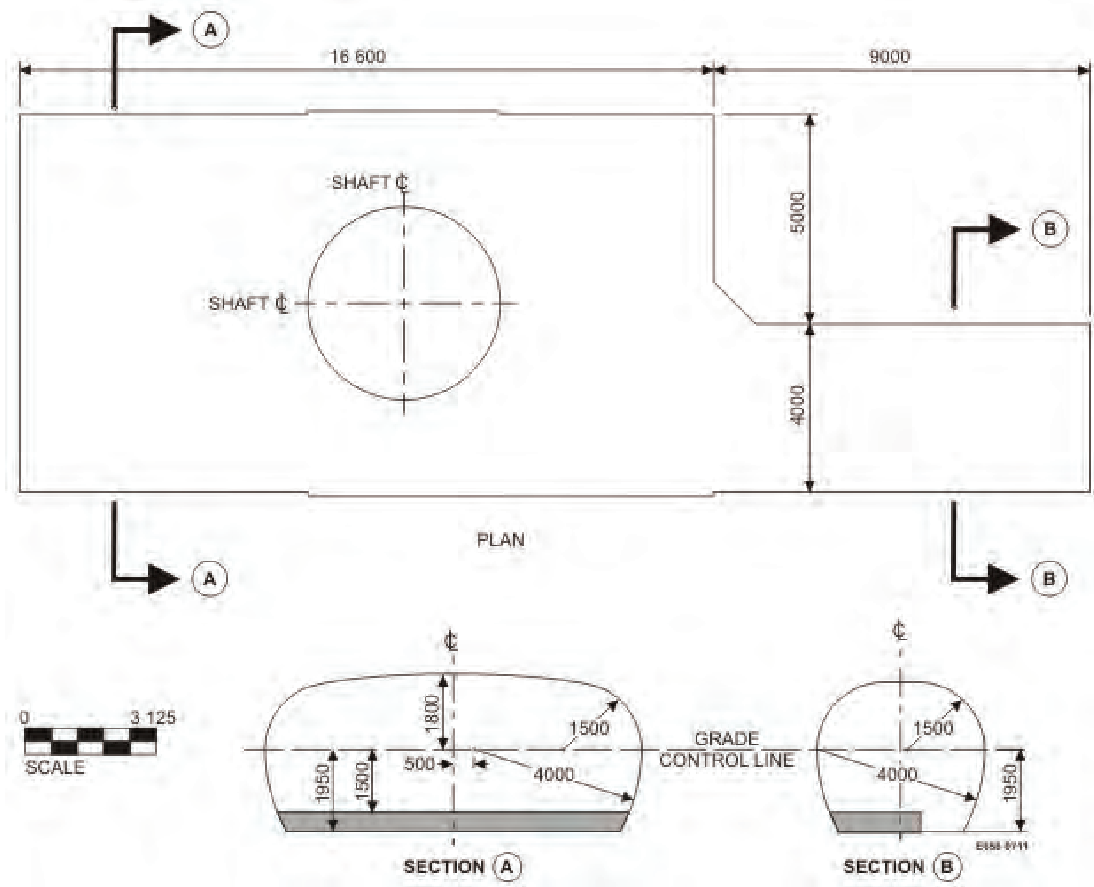


Figure 15: Station Profile and Plan

Although the exact stress magnitudes were not known at that time, the behaviour of the rock (e.g. breakouts below Fracture Zone 2 in the shaft) suggested a high horizontal stresses. To verify the stability of the profile laid out for the stations, a boundary element stress analysis was conducted using preliminary data for stress and strength available at the time.

2.2.7 Post-construction Stability of Excavation

The Manitoba Workplace Safety and Health Act requires that a shaft inspection book be kept that records the condition and inspection activities in operating mines within the province. The Shaft Inspection Record Books for the URL do not indicate any long-term instability. Upon review of the shaft inspection records, some references are made to small pieces of rock removed from timber sets. For the most part, it was determined that this material was primarily related to hoisting muck during mucking operations during shaft sinking and subsequent level development operations.

2.2.8 Lessons Learned

The “cost plus fixed fee” contract format was effective in meeting the needs of the owner and the Contractor in a situation when the owner has objectives either affect that the Contractor’s ability to advance the excavation/construction work without interruption of the work cycle (such as characterization of the rock and instrument installation) or represent unproven or unfamiliar technology (such as the use of full-face blasting with a burn cut). This form of contract has implications that must be satisfied by all parties to maintain an effective working relationship and meet the project objectives. The following aspects of the “cost plus fixed fee” form of contract are important.

1. The “cost plus fixed fee” contract format used to construct the URL lower shaft provided flexibility in accommodating an unproven excavation method and geotechnical characterization during sinking. The “cost plus fixed-fee” contract form provided for a reasonable sharing of the project risk, leaving the Contractor with those risks that are normally associated with the excavation and construction projects and the owner with the risks associated with the unproved blasting method and the geotechnical characterization activities. The cost plus fixed fee contract form was an economical way to allow owner imposition of an unproven excavation method and owner management of the uncertainty in the geotechnical characterization program without having the Contractor insert a major fee for risk and uncertainty in the contract.

2. Since the owner assumes more risk in using a cost plus fixed fee contract form, the owner must have an experienced contract management team to manage their risk and to control the project cost and schedule. In this form of contract, the Contractor’s risk is reduced and the Contractor’s opportunity for a larger profit margin is also reduced, as his flexibility to accelerate project progress and reduce the schedule is limited. Contractors are generally willing to accept a project that has a lower profit margin, providing the risk is minimized and an acceptable return on investment is provided.

   As the owner has assumed more risk, the owner must assemble a project management team with the expertise and experience to direct the project (e.g., control owner’s risk, control budget and schedule, provide the additional input to the project (e.g., blast round designs, ground control designs)). The owner must rely more heavily on his own resources.

The lower shaft contract at the URL provided a win-win situation for both Contractor and Owner. The Contractor's margin is pre-established in the tendering process and he is paid for extra work required. This reduces both the risk to the Contractor and the complexity of the "claims" process. By establishing a less adversarial relationship between the parties, the owner and the Contractor both have an objective of completing the project as soon as possible and at the lowest cost subject to achieving the owner's objectives (e.g., unproved excavation method, geotechnical characterization, etc.). Under these circumstances there is a more effective working relationship in which the Contractor can insert his knowledge and experience into project planning and implementation.

The incentive bonus system for the Contractor's staff adopted for the lower shaft construction identified project key issues related to the Owner's project objectives. In the lower shaft the bonus system was tied to three main objectives, quality of drilling and blasting, achieving schedule milestones, and adequacy of shaft turnover for characterization.

The drilling and blasting quality objective rewarded the contract miners for advances but the reward was related to how closely a given blast design was followed. The adequacy of timely shaft turnovers required that the Contractor provide a clean and safe working space and the required services for personnel performing characterization duties during the day shift. Bonuses awarded for this aspect were very useful to the Contractor as it reduced likelihood of characterization work running into the time designated as excavation advance periods for the Contractor.

General lessons learned in applying a cost plus fixed fee form of contract to construction of the URL lower shaft include the following.

- The owner must take more responsibility in project managing a cost plus fixed fee contract format.
- The owner must assume the risk resulting from the application of new methodologies or deviations from current practice, such as full-face shaft-sinking blast rounds utilizing a burn cut, geotechnical characterization during sinking operation, and the quality and stability of excavation surfaces.
- Effective implementation demands a team approach, fosters/encourages cooperation, good communications between parties and an open door policy, amenable to grievance identification and resolution for both sides.
- A clear process for change control must exist. The trend reports facilitated communication by necessitating a written explanation of the problem and recommendations for implementing the change required to correct the problem. The trend report provided early warning of possible issues that could possibly compromise the project success.
- A comprehensive safety program is important. The safety program must include everyone involved with the project (owner and Contractor employees). The safety program involved project meetings and daily meetings that included operations staff.

The cost plus fixed fee contract format was appropriate for the URL shaft extension project, since it provided a high degree of owner control and flexibility. The project management team was able to coordinate activities and achieve the geotechnical program objective, while excavation and construction work progressed.

The full-face blasting method worked well. Accurate drilling and following the proven blast designs developed during the project were critical to successful full-face blasting during the lower shaft sinking project. Shift inspection ensured that the designs were followed as intended.

Rapid shaft sinking was not the objective in the URL shaft extension project. The data compiled during the project, however, indicated that the schedule performance and possibly cost could be improved with longer rounds. The cycle time per metre of advance also depended on the size of the shaft and the equipment selected for the project.

2.3 240 AND 420 LEVEL

Upon completion of the upper shaft, the shaft sinking Contractor was retained to excavate an additional 100 m of lateral development work at the 240 Level to provide excavations for an instrument array in Room 209 and access for a bored ventilation raise to the surface from Room 211 (Kuzyk et al. 1986b) (later moved to Room 208 (Figure 3)). This work was completed in 1985 August. The majority of this section describes excavation work on the 240 Level as that is where most techniques were first used.

Controlled drilling and blasting was employed to excavate most of the tunnels and test rooms needed for geotechnical projects on both the 240 and 420 Levels at the URL (Kuzyk and Kwon 2006). Controlled drilling and blasting was optimized during the development of the 240 Level, initially using a pilot and slash method of advance, and later using a full-face method of advance (Kuzyk et. al. 1996). The full-face method of advance was utilized to excavate most of the headings on the 420 Level, where in situ stress conditions were more severe.

Controlled drilling and blasting is flexible and cost effective in hard, igneous rock and the experience from previous work at the URL indicated that the method could be used to excavate a geological repository in Canada. The reason for emphasis on control of wall quality in this project was that the reduction of excavation-induced damage in the perimeter of excavations would be important in a geological repository to limit the potential for excavation-induced damage to provide contaminant transport pathways in an axial direction around excavated openings. Controlled drilling and blasting methods were implemented and quality control procedures were developed to achieve high-quality wall surfaces (retaining 80% or greater drill-hole traces) and provide a profile conducive to long-term stability. (Kuzyk et al. 1986c and 1987).

For the most part, tunnels at the URL were relatively small, having dimensions of about 3.5 m wide by 3.6 m high. Tunnel profiles had curved crowns inverts to reduce stress concentrations around the perimeter and minimize stress related failure. The tunnelling work demonstrated that the drill and blast method is very versatile and adaptable and could be effective during the excavation of repository placement rooms. The controlled blast designs produced a very high quality tunnel wall with minimal bootleg.

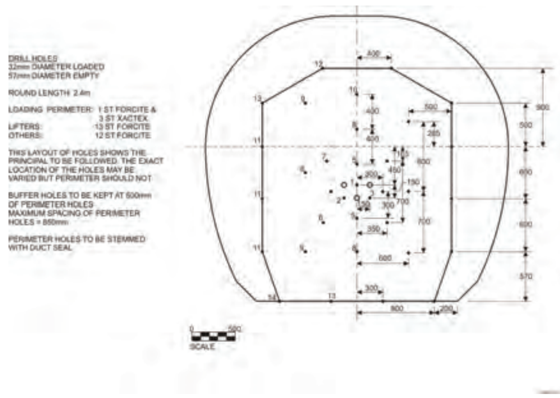
Quality control criteria for the controlled drilling and blasting operations were stringent and included:

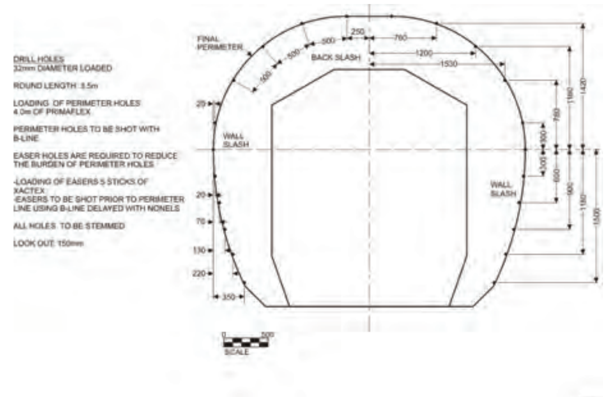
- detailed records for drilling and blasting operations of the:
  - drill-hole location (within 100 mm of design location along length), depth and alignment,
  - type of explosive, quantity of explosives per hole and detonation sequence, and
  - burden, spacing and powder factors;
- full shift inspection to ensure accurate location and alignment of drill holes;
- survey control to ensure accurate location and alignment of the tunnel; and
- simultaneous detonation of the perimeter holes.

Excavation procedures were also established to assess the need for ground control and/or ground support systems. These procedures involved mapping the rock mass conditions, stability analysis, designing rock support for the specific conditions encountered, stringent quality control inspection during installation, and post-installation testing. Ground control, comprised primarily of screening, was installed on the 420 Level in all headings as a precaution due to the high incidence of spalling and loose cause by higher in situ stress. It should be noted that screening is considered a safety precaution rather than ground support, as the bolts used to hold the screen in place were only 0.6 m long and provided minimal reactive support. Protective screening was not required on the 240 Level, but was installed in the electrical substations, pump station and in some test rooms were permanent equipment was installed. Where required for support of the rock mass, grouted rock bolts (typically 2 m or more in length) were installed in a few locations such as at station brows and stressed corners at tunnel intersections throughout the URL.

In the early stage of construction on the 240 Level, the pilot and slash method shown in Figure 16 was applied. This method was very effective and produced good results from an excavation damage perspective. However, the tunnel had to be advanced in two steps, first by excavating a pilot heading and then by following with a slash. The typical advance per blast with this method was about 2.4 m in the pilot heading (Figure 16(a)). To minimize setup time for drilling, mucking, scaling and the installation of services, several pilot blasts would be advanced before the final perimeter slash was excavated. Most slashes were 3.5 m in length. The slashes would generally be drilled in one setup but excavated in three blasts, one for each side and a third for the crown (Figure 16(b)). Usually, the side slashes would be blasted together and the crown blasted next.

Generally, at least four 2.4 m pilot rounds would be excavated first, followed by two 3.5 m slashes. In the single heading arrangement on the 240 Level, the overall rate of advance for the finished heading after slashes averaged about 0.6 m per shift. This performance could improve significantly in a multiple heading arrangement where separate crews could be organized for drilling, blasting, mucking and the installation of ground support and services. This practice is common in sublevel mining operations. Such an arrangement could be feasible in a repository where three or more emplacement rooms could excavated simultaneously.



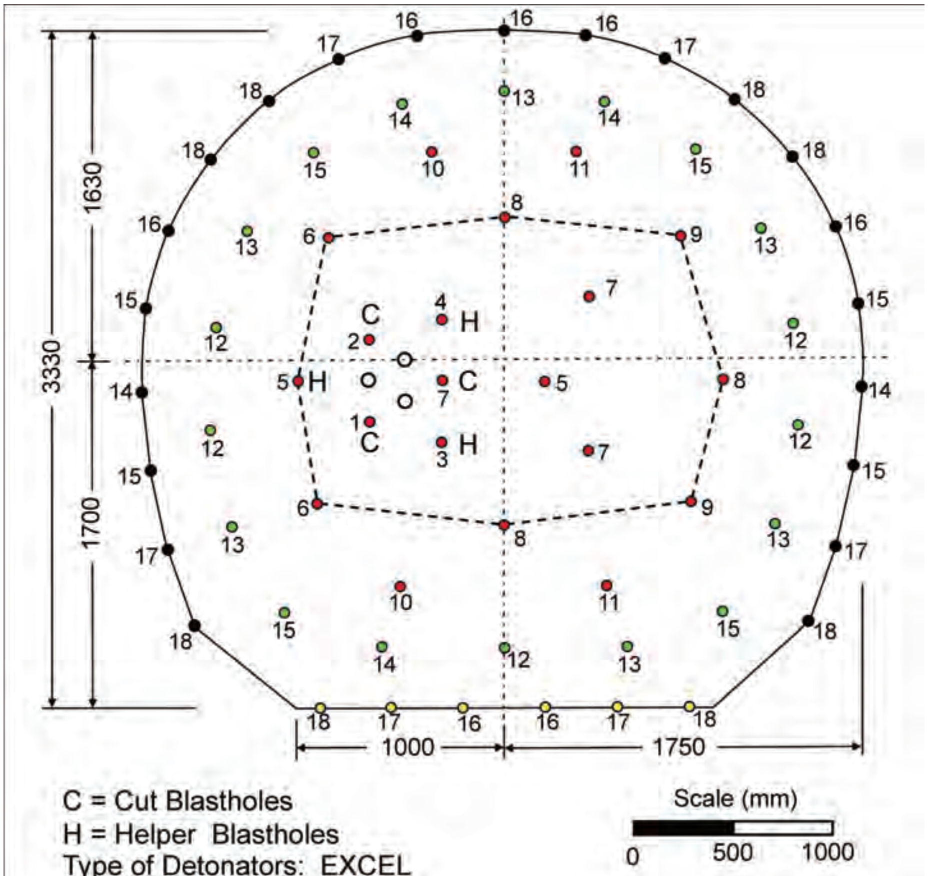


(a) Pilot round blast design

(b) Slash round blast design

Figure 16: Pilot and Slash Round Designs

For improved excavation productivity (advance rate), the full-face blasting method shown in Figure 17 was applied in the later stages of the 240 Level construction. This eliminated the need to advance the tunnels in two steps. Although the full-face method was not as effective at reducing blast-induced damage, the controlled drilling and blasting produced results that were still acceptable. With the application of the full-face method, the advance per blast increased to 3.5 m and the overall rate of advance to about 0.9 m per shift. A single boom electric-hydraulic jumbo drill was used to drill the blast holes.



Blasthole Type	Number	Loading
Relief Hole	3	100 mm Reamed Hole
Production	20	Amex II, Collars of 250 mm for Cut 300 mm for Easers 300 mm for Production
Lifter	6	Forcite 75% (F25 x 200), 200 mm Collar
Cushion	18	Lomex III, 400 mm Collar (1 x F25 Primer)
Perimeter	21	Prima-flex (2 x F25 Primer), Stemmed
	68	

Figure 17: Full Face Round Design

The following design aspects were determined to be useful in full face blasting.

- A triangular-shaped cut (Figure 17) is advantageous because the cut blast holes are more widely spaced. The cut blast holes were positioned to reduce the effect of sympathetic detonation or dynamic pressure desensitization caused by the earlier firing cut blast holes[2].

[2] Sympathetic detonation is the premature initiation of a charge in the blast round caused by an earlier firing charge – usually the result of the charges being placed too close to each other. Dynamic pressure desensitization results when the explosive in a charge is compressed by the shock wave of an earlier firing charge – resulting in failure of the charge to detonate properly. Each cut blast hole has two relief holes into which it can break and each of the

- Detonator delay sequence, charge mass per delay, burden and spacing were determined for specific blast patterns.
- Blast holes were bottom initiated with an additional primer charge to ensure a strong detonation front.
- Typically about 65 blast holes were drilled in each round (see Figure 17). Blast holes, generally drilled 38-mm diameter, were charged with various explosive products. Three relief holes were drilled and reamed to a diameter of 89 mm or 100 mm at the center of the face. The ability of the bottom element of each cut blast hole to break cleanly to the relief holes was improved by drilling the relief holes at least 300 mm deeper than the rest of the round.
- Of these 65 holes typically six lifter holes were drilled on the bottom of the round to form the tunnel floor. A higher energy explosive was used (Forcite 75% (25-mm-diameter)) to aid breakage under higher confinement. Only the bottom two cartridges are tamped in this case (Kuzyk et al. 1994).
- Typically 21 of the 65 blast holes were perimeter holes and these could be traced or decoupled or have low-strength explosives to lessen shock energy created by detonation.

Figure 17 shows a typical blast design. Figure 18 shows a slash round being drilled out and Figure 19 shows a full face round being drilled. Figure 20 shows loading operations.

Further developments of the excavation technology were undertaken at the URL during the excavation of Rooms 215 and 216 on the 240 Level, (Figure 3). A total of six longer rounds were excavated to evaluate the benefits of longer rounds. Blast rounds lengths were increased in increments of about 1-metre to a final length of about 8.7 m (Kuzyk 2003, Kuzyk et al. 2003, Mohanty and Joyce 1994, Kuzyk et al. 1994) to demonstrate that controlled drilling and blasting could be carried out in longer rounds (e.g., Figure 21).

A blast round that had more than 8 m of advance in a single blast was significant, as the tunnel advance was more than two-times greater than the largest dimension of the tunnel profile, e.g., 3.5 m wide by 3.6 m high. Excavation design typically assumes that a blast round advance to provide a stable tunnel is no more than the same advance as the largest dimension of a excavation profile.

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cut blast holes are “shielded” from the other two by the relief holes (Hagan et al. 1989, Mohanty and Joyce 1994, Kuzyk et al. 1994).

Figure 18: Slash Round

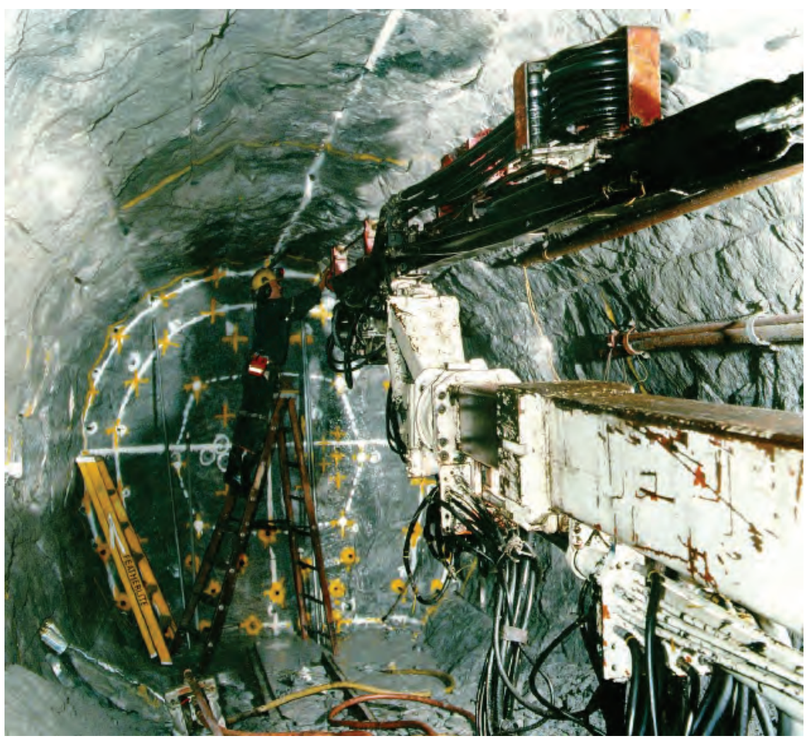


Figure 19: Full Face Round

Figure 20: Loading a Round

For the most part, blast designs for the longer rounds were consistent with those used for 3.5-m-long rounds. There were some exceptions, however. The drill pattern and charge configuration were adjusted to compensate for small changes in ground conditions.

The tunnels in which the long blast rounds were located were oriented perpendicular to the direction of maximum principal stress, which maximizes the deviatoric stress on the cross section and is the worst condition from a stability point of view. The maximum principal stress on the 240 Level was 26 MPa (azimuth 228° and plunging 8°) and the intermediate and minor principal stresses were 17 MPa (azimuth 135° and plunging 23°) and 13 MPa (azimuth 335° and plunging 65°) respectively, with the minor stress being sub-vertical. The rock mass quality was generally considered to be very good, having a Norwegian Geotechnical Institute (NGI) Tunnelling Quality Index (Q) between 50 and 250. The elastic modulus of the rock at the location was 55 GPa, Poisson's ratio was 0.18, density was 2670 kg/m^3, unconfined compressive strength was 200 MPa, and tensile strength was 11.1 MPa.

With the application of full-face controlled drilling and blasting in long rounds, very high quality tunnel wall with minimal bootleg was achieved. Only minor scaling of loose rock was required on the tunnel walls and crown. Ground support or protective screening was not required. However, from a practical standpoint, changes would have to be made to supporting tunnelling equipment to fully exploit this technology. This is particularly the case with the drilling and loading of the longer blast holes. An accurate drilling system capable of drilling small diameter blast holes is essential for reliable performance. Also, explosive products for the perimeter blast holes need to be suitable for loading in long blast holes. Products available at the time were not well suited for long blast holes.

Figure 21: Long Blast Round

2.3.1 Construction Contract

A service contract format was used to construct the 240 and 420 Levels. The scope of work required that the Contractor provide supervision, labour, equipment and services as required for underground mining related tasks. The type of work required included the following.

- Provision of labour, including hoist persons, cage tenders, and shaft persons, and supervision for the hoist operation.
- Provision of labour, equipment and supervision of minor installation, operation and maintenance of the mine hoisting system, such as -:
    - installation, commissioning, inspection and testing of shaft conveyances and hoisting ropes;
    - operation, adjustment and maintenance of shaft installations including cage-skip loading and dump mechanism, shaft sets and guides, intake ventilation system, and ventilation air propane heating system;
    - shaft bottom cleaning and spillage collection systems; and
    - shaft hydrogeological groundwater monitoring system.
- Provision of labour and supervision for installation and modifications to the shaft stations, and level development, such as installation of -:
    - station track work;
    - level ventilation ducting, fans, controls and fire doors;
    - temporary mine services, including compressed air, service water and telephones; and
    - temporary and permanent ground control and ground support equipment.
- Provision of supervision, labour, including experience drift and raise miners, and mining equipment for excavation of level development access drifts, research test rooms and diamond drilling and raise bore stations.
- Provision of supervision, labour and equipment for the installation of underground ventilation fans and ductwork.

- Provision of supervision, labour and equipment for the installation of electrical and mechanical services, including electrical power cables, power centres, conduits, distribution panels, and mechanical pipe work.
- Provision of supervision and labour for resolution of minor operations deficiencies and related tasks.

The general conditions under the service contract included the following requirements.

- The right for AECL to accept staff proposed for assignment based on qualifications of the individuals and the specific needs of AECL. All journeymen and tradesmen shall possess a valid certificate issued by the Province of Manitoba. All apprentices shall be indentured in an apprenticeship training program with the Manitoba Department of Labour, Apprenticeship Division. Any personnel found to be unacceptable by AECL shall be replaced.
- AECL may, by giving written notice to the Contractor, terminate all further work under the contract.
- AECL will at all times exercise general direction and control (to such an extent as AECL may determine) over the performance of the work and all orders and directions given by AECL shall be promptly and fully complied with by the Contractor. (This direction was provided by AECL's Resident Engineer located on site at the URL).
- The Contractor shall only perform work on such projects as AECL may request and shall not exceed the total estimated cost of this work, set prior to commencement without further written approval.
- The Contractor shall not sub-contract any of the work in connection with the projects assigned except with the prior approval of AECL. All such sub-contracts shall as far as possible include a termination clause as above.
- AECL reserved the right to provide labour, equipment and materials at its sole discretion.

Under the contract, AECL agreed to reimburse the Contractor for services rendered as per a schedule of Labour and Unit Rates, which included the following items.

1. Local labour service hourly rates for:
    a. Surface labourers;
    b. Underground labourers;
    c. Hoist persons;
    d. Cagetenders;
    e. Lead miners / Shift Forepersons;
    f. Jumbo drillers;
    g. Longhole / downhole drillers;
    h. Miners;
    i. Mechanics;
    j. Electricians;
    k. Equipment operators; and
    l. Supervisor / Superintendent.
2. Travel, room and board allowance (to be added to Item 1 rates) for non-local employees.

3. Hourly rate bonus allowance to reflect quality of work as well as productivity. This was to be determined biweekly with agreement of AECL's representative (Resident Engineer) located on site.
4. Equipment service rates, including:
   a. Atlas Copco Cavo 320 rubber tire wheel-mounted overhead loader with spare parts (monthly rental);
   b. Mancha trammer one and one-half ton locomotive complete with charger, spare parts and extra battery (monthly rental);
   c. Rail-mounted single-boom (or double boom) drill jumbo complete with spare drill, electronic alignment and parallel holding features or equivalent (monthly rental);
   d. Power cable for rail-mounted drill jumbo complete with connectors, ground fault and ground check systems (length 300 m);
   e. Down-the-hole drill, 100 mm (4 inch) diameter drill string, bits, tools and equipment (monthly rental);
   f. Longhole drill, 100 mm (4 inch) diameter by 50 m depth with bars, arms, drill string, bits, tools and equipment (monthly rental);
   g. Office trailer equipment, copier, tools, safety supplies, etc. (monthly rental)
   h. Mobilization and demobilization (lump sums);
   i. Non-local manpower mobilization cost to and from site (lump sum); and
   j. Mobilization and demobilization of additional equipment (freight rates).
5. Material charged at invoice cost plus specified percentage mark up.
6. The normal hours of work defined in the contract were from 08:00 hours Monday to 08:00 hours Saturday. The excavation work was carried out on a one, two- and/or three-shift per day arrangement as required by AECL. For normal hours of work, AECL paid the Contractor:
   a. the specified local labour hourly service rate for all hours worked;
   b. when applicable, an additional travel, room and board allowance, for up to 40 hours per week, at the specified hourly rate for all employees without a permanent residence within a 100 km radius of the URL site; plus
   c. when applicable, an additional hourly performance bonus allowance as authorized by AECL's representative Resident Engineer at cost. The bonus arrangement for crew performance was subject to approval of the Resident Engineer and was to reflect quality of work as well as productivity.
7. The overtime rate of time and one-half was paid for all hours worked, which exceeded the statutory workweek for construction employees. No payments were made for vacation or sick leave.
8. Service calls and call-ins were paid at a minimum charge of four hours time, or actual hours worked, which ever was greater.
9. Materials and supplies provided under the contract were invoiced at cost to the Contractor plus specified percentage mark up.
10. The Contractor was required to keep proper accounts and records of the cost of the work required under the contract, and all the expenditures or commitments made in connection with the work. He was required to keep all invoices, receipts, vouchers, purchase orders, time sheets/cards and documents open for audit and inspection by AECL at all times.
11. The Contractor was required to pay for all necessary permits or licenses required for the execution of the work.
12. The Contractor was required to maintain liability insurance as required to protect him and AECL from claims under the Workman's Compensation Act and from any other claims for damages from personal injury, including death, and from claims for property

damage which may have arisen from his operations under this contract. Insurance included:

a. General Public Liability to third parties of not less than $2,000,000 per occurrence; and
b. Vehicle coverage for Public Liability and Property Damage of not less than $2,000,000.

2.3.2 Estimated and Actual Construction Schedule and Cost

Upon completion of the upper shaft in 1985 May, the shaft sinking Contractor was retained to excavate 100 m of lateral development on the 240 Level. This work, referred to as Contract No. 6a, was completed in 1985 August in preparation for the boring of the ventilation-escapeway raise (Room 002) from Room 208 to surface. The raise was bored and equipped in 1985 September by a different Contractor under Contract No. 7.

Table 8 shows a comparison of costs associated with the excavation of the 130 Level and 240 Level shaft stations and the excavation work carried out on the 240 Level (part of Room 201 and Rooms 203 and 207) under Contract 6a prior to boring the ventilation-escapeway raise from the 240 Level to surface.

Table 8: Cost Comparison of Upper Shaft Stations and 240 Level Access Drift Excavation Costs

Item	Description	Contract 4 Shaft Stations	Contract 6a 240 Level Tunnels
1.	Excavated volume (estimated)	2,277	1,220
2.	Owner supplied explosives	$ 50,000	$ 36,000
3.	Contractor Costs	$ 438,900 (Note A)	$ 212,100
4.	Resident engineering support	$226,000	$ 127, 600
5.	Owner supplied material equipment & rentals	Nil (part of item 3)	$ 31,500
6.	Owner supplied labour	Nil	$ 3,100
7.	Total cost	$ 714,800	$410,400
8.	Contractor man-hours (Note D)	9,463	5,150 (Note B)
9.	Contract Cost / m^3	$ 192.7 / m^3	$ 202.3 / m^3 (Note C)
10.	Contractor man-hours / m^3 (Note D)	4.15 Man-hr / m^3	4.22 Man-hr. / m^3
11.	**Total cost / m^3**	**$314.0 / m^3**	**336.0 / m^3**

Note A – derived from Contract 4 Costs.
Note B – Includes owner-supplied labour required for shaft operation.
Note C – Total of items 3, 4 and 6.
Note D – Supervision supplied by Contractor not included.

The bulk of the 240 Level was excavated between 1985 May and 1990 October. The work was partially carried out with hand-held jackleg drills off a rail-mounted drill truck (Room 209), and a rail mounted twin-boom pneumatic drill jumbo (Rooms 205, 207, 208 and 211).

The cost of excavating the 240 Level Raise Bore Station (Room 208) was estimated to be $153,600, which included $31,700 for shift inspection by AECL staff (1985 Canadian Dollars).

Rooms 214, 215 and 216 were excavated between 1990 November and 1994 March. A total of 34 blast rounds were excavated for a total advance of 128 m or a total volume of 1300 m^3. Of the 34 blast rounds, 19 were 3.5-m-long rounds with the remaining consisting of slashes and long blast rounds ranging in length from 4.0 m to 8.7 m. The work was carried out on a two eight-hour shifts per day, five days per week basis with only a single heading available for excavation activities at any given time.

The initial excavation of the 420 Level commenced in 1990 May and continued to 1990 October. This work comprised the excavation of the incline and decline ramps and the upper and lower instrument galleries (Rooms 405 through 413). The volume excavated was 4,470 m^3. The total service contract labour, including overtime and bonus, was 7,353 man-hours at a cost of $337,500 or $70.75/$m^3$ (1990 Canadian Dollars). This part of the 420 Level excavation work was carried out on a three eight-hour shifts per day, five days per week basis with two headings available for excavation activities at any given time, which made it more efficient and cost effective then other excavation projects on the 240 and 420 Levels.

The increased price per cubic metre on the 420 Level reflects the increased cost of working in the more highly stressed rock environment. This is mainly because of the need for ground control on the 420 Level. In most cases, the ground control had to be replaced many times during advance because it was kept close to the face and damaged by subsequent blast rounds.

2.3.3 Construction Requirements

The 240 Level was planned by AECL so that fresh ventilation air could be circulated through the main access tunnel from the shaft to the return air ventilation raise. The main access tunnel was excavated in the shape of a ring and provided access to test rooms at various locations. The ring arrangement was beneficial to the characterization program as it provided access to a variety of in situ rock mass conditions available at the 240 m depth. One side of the ring provided close proximity to Fracture Zone #2 (a zone of highly-fractured rock) while the other side provided access to a moderately fractured rock zone with an interconnected fracture network. Access to single fracture splays and more competent rock without fractures was also available (Figure 1). This arrangement contributed to the success of the experimental studies. Since the ring could be excavated in two directions simultaneously from the shaft station, the configuration made excavation work a little more efficient as two heading could be worked simultaneously.

The 420 Level was developed with specific needs for large-scale experiments, such as the need for an incline and decline to allow instrumentation for an experiment to be installed from several directions. A raise connected the incline and decline and this facilitated air circulation on the level. In general tunnels were excavated (shape, orientation) to meet the higher stress conditions on this level.

On both the 240 and 420 Levels, tunnels and test rooms were excavated with curved shapes to improve stability and safety. Drilling was initially done on the 240 Level with hand-held jackleg and stoper drills. Tunnel blast rounds were drilled from a gear-truck designed to accommodate the jackleg drills. Several hand-held drills were purchased to ensure a sufficient supply for ongoing maintenance requirements. AECL also purchased the required drill steel and bits and bit sharpening equipment required, which optimized the drilling costs. Later, a two-boom pneumatic drill jumbo was rented from the Contractor to evaluate any improvements in drilling accuracy and performance that may be realized from mechanizing the blast round drilling operations. This proved to be beneficial, so AECL purchased a ‘state-of-the-art’ rail-mounted single-boom electric-hydraulic drill jumbo equipped with a SIG 100 drill and a spare drill. The drilling efficiency of the newer electric-hydraulic jumbo was better then the previous jumbo unit tested and could drill longer rounds (up to 5 m). On the 240 Level this rig was useable on track mounted equipment, which was well suited to the flat design of the 240 level. On the 420 Level the equipment was wheel mounted, which was well suited to the varied excavation profiles and the incline and decline work.

Permanent services on both levels included compressed air; service water, fire water, potable water and temperature controlled water (for geotechnical in situ stress measurement) diamond drilling water; electrical (110, 220 and 600 volt) power outlets at power centres; lighting; geotechnical data acquisition systems; and auxiliary ventilation for experimental test rooms and dead-end access tunnels. This provided the required services for experimental work.

2.3.4 Lessons Learned

During excavation of the 240 Level it was found that effectively integrating characterization activities with construction was as important as was during shaft development. The principle was therefore applied to all subsequent level development tunnelling work.

When tendering the service contracts, AECL purchased equipment where a cost saving could be realized over above the cost of renting equipment from the Contractor, e.g., a rail-mounted single-boom electric hydraulic drill jumbo, rubber-tired Cavo overhead loader, Mancha trammer a 2-cubic-yard scooptram, and a rubber-tired diesel-powered chassis for the electric hydraulic drill jumbo used on the 420 Level.

The service contract format was effective in providing AECL with flexibility, cost savings over other forms of contract and access to specialized and skilled mining crews as required for state-of-the-art equipment and methods used during URL level development.

Slashes were used to progressively widen the shaft stations on each level. The tunnels on the 240 Level and 420 Level were principally excavated using drill and blast techniques. On the 240 Level, the first tunnel excavations were by the pilot and slash technique. However, full-face excavation was adopted for most tunnels on the 240 Level and for those on the 420 Level, except Room 415 (See Figure 3).

The uncertainty and the variability in the natural environment are issues that are dealt with during the siting, construction and operation phases of a deep geological repository. It is helpful to have a structured approach to the design and construction of underground excavations, as the relationships among the facility design, excavation, construction, the rock mass and the groundwater systems are important to, and are often elements of, the research, demonstration and characterization objectives of the facility. A formal approach to design, construction and operation of the URL was important to achieve objectives, particularly if the studies are undertaken as part of a regulatory approval process.

3. EXCAVATION PERFORMANCE ON URL LEVELS

When developing a deep geologic repository, underground openings will need to remain in service for long periods of time. The Adaptive Phased Management (APM) approach (NWMO 2005) indicates that a repository may be open for long times (e.g., up to 300 years) before final sealing (NWMO 2005). This means that the access shafts, ramps and tunnels could be open for this period and must remain stable either by virtue of the rock strength and tunnel design or with assistance from engineered rock supports (e.g., rock bolts, etc.). To be effective, designs will have to be based on a characterization program that provides a understanding of the in situ rock mass properties and conditions.

The oldest URL excavations have been in existence since 1982. The URL provides a case study where:

- the rock conditions and in situ stresses are well known, and
- there is a range of ground conditions from a moderately jointed rock mass with moderate stresses to massive rock with higher stresses (Section 1.3).

Studies at the URL have identified that stability factors related to the rock mass include variations in geology, rock mass quality and in situ strength, and in situ stress.

- Subtle differences in lithology, grain size distribution, the fabric of the rock, and microstructure can affect macroscopic strength of the rock. The variations can result in variable mechanical properties (Read et al. 1997a, Everitt and Lajtai 2004).

- The unconfined compressive strength is not a true measure of long-term rock strength. The effective long-tem in situ strength is less than the unconfined compressive strength because the presence of microcracks, fractures and variations in geology at the scale relevant to in situ performance introduces weaknesses in the rock mass. Non-conventional laboratory tests are needed to determine the design values (Lau and Chandler 2004).

- The magnitude of in situ stresses relative to the rock's effective long-term strength and the orientation of the stress tensor relative to the rock fabric affect excavation stability. In general, low stress levels combined with sparsely fractured and high strength rock, such as the conditions observed at the 240-m depth of the URL, will result in stable excavations independent of the orientation of the opening relative to the maximum principal stress direction. Regardless of stress conditions, an understanding of the in situ stresses will improve the quality of the excavation both immediately following excavation and during the long-term transient period after construction.

Studies at the URL have identified that stability factors due to the excavation approach or layout include orientation relative to the stress tensor and tunnel shape.

- The orientation of the excavation relative to the principal stress directions, rock structures (e.g., bedding planes, dykes) and rock fabric (e.g., foliations) influence the development of excavation damage. These features can provide preferential surfaces for rock fracturing and formation of unstable rock slabs in the roof of the tunnel, or may result in damage at a specific location on the tunnel perimeter, particularly if a tunnel is oriented poorly with respect to the orientation of the features (e.g., poor alignment with joints may allow a wedge failure). However this was not an issue for either the 240 or 420 Level excavations in the URL due the limited and widely spaced or lack of fracturing in the rock mass.

- By selecting an excavation cross sectional shape compatible with the stress orientation and rock structure, the stress-induced excavation damage can be reduced and a more stable opening produced.

A study of excavation maintenance at Äspö Hard Rock Laboratory was conducted by Andersson and Söderhäll (2001). They noted that maintenance (scaling), during both construction and operation, was largely located in the widened curves of the access tunnel and in areas containing veins or intrusions of Småland granite or fine-grained granite. Their observations of the relationship of the amount of loose rock scaled scaling and local geological conditions appears to be similar to the URL experience.

One qualitative indicator of excavation stability is the amount of scaling required. When an underground opening is first excavated, the in situ stresses adjusted to the presence of the opening, and, depending on the conditions listed above, this may result in some damage and spalling of the rock on the excavation perimeter. The rock around the opening adjusts through progressive damage to produce a stable tunnel cross section for the current rock strength and stress conditions. In the URL, the openings stabilized over time and the amount of scaling required decreased with time for situations in which the in situ stress and environmental conditions remained stable. Variation in conditions, such as changes in temperature and moisture (e.g., standing water, humidity), may have caused some localized failure, which then required scaling on the upper portion of the excavation perimeter at a minimum to make areas safe for personnel to work under. However, excessive scaling can facilitate subsequent spalling as the loose but attached rock provides a small confining pressure that reduces the likelihood of further excavation damage activity under stable conditions (Martino 2000). Thus scaling must be done in moderation while still maintaining operationally safe conditions.

Scaling records were kept only for the levels at the URL. On the 300 shaft station and the 420 Level, scaling of the back was limited due to the screening installed to retain spalling rock, but this build-up of loose rock provided a confining pressure to inhibit further development of loose. Once the screen on the back was removed during recent decommissioning activities, some obvious drummy areas (i.e., areas of loose but not disconnected rock that make a hollow sound when tapped) were noted but these were not recorded. The walls, which were not screened during operations, did not require scaling.

The design cross section shape and final as built cross section shape of the excavated tunnels is provided in Figure 22 and Figure 23 for the 240 Level and 420 Level respectively. The differences between the design cross sections and the as built cross sections are a result of excavation round loading, the micro geology, and the response of the rock mass to the presence of the excavation. Room numbers are identified and can be located in Figure 3. This as built cross section shapes illustrate tunnel shapes that have been stable in the different stress environments and different orientations relative to the in situ stress tensors. The stress orientations for the 240 and 420 levels are provided in Table 9. In general, the orientation with the highest concentration of stress is perpendicular to the major principal stress (σ_1).

On the 240 Level, tunnels were generally designed and excavated for function and safety. On the 420 Level, tunnel cross section shapes were designed and excavated either for simple function and safety during operations, or to demonstrate the response a highly stressed, sparsely fractured rock to excavation by enhancing failure (Room 415) or encouraging excavation stability (Rooms 417, 418, and 421). Because of the nature of the URL, a large variety of excavation shapes were selected, to meet experimental, operational and stability requirements.

Table 9: In Situ Stress Tensors on the 240 and 420 Levels

Level	Stress	Magnitude (MPa)	Azimuth (°)	Plunge (°)
240	σ_1	26	228	8
	σ_2	17	135	23
	σ_3	13	335	65
420	σ_1	60	145	15
	σ_2	45	54	6
	σ_3	11	303	74

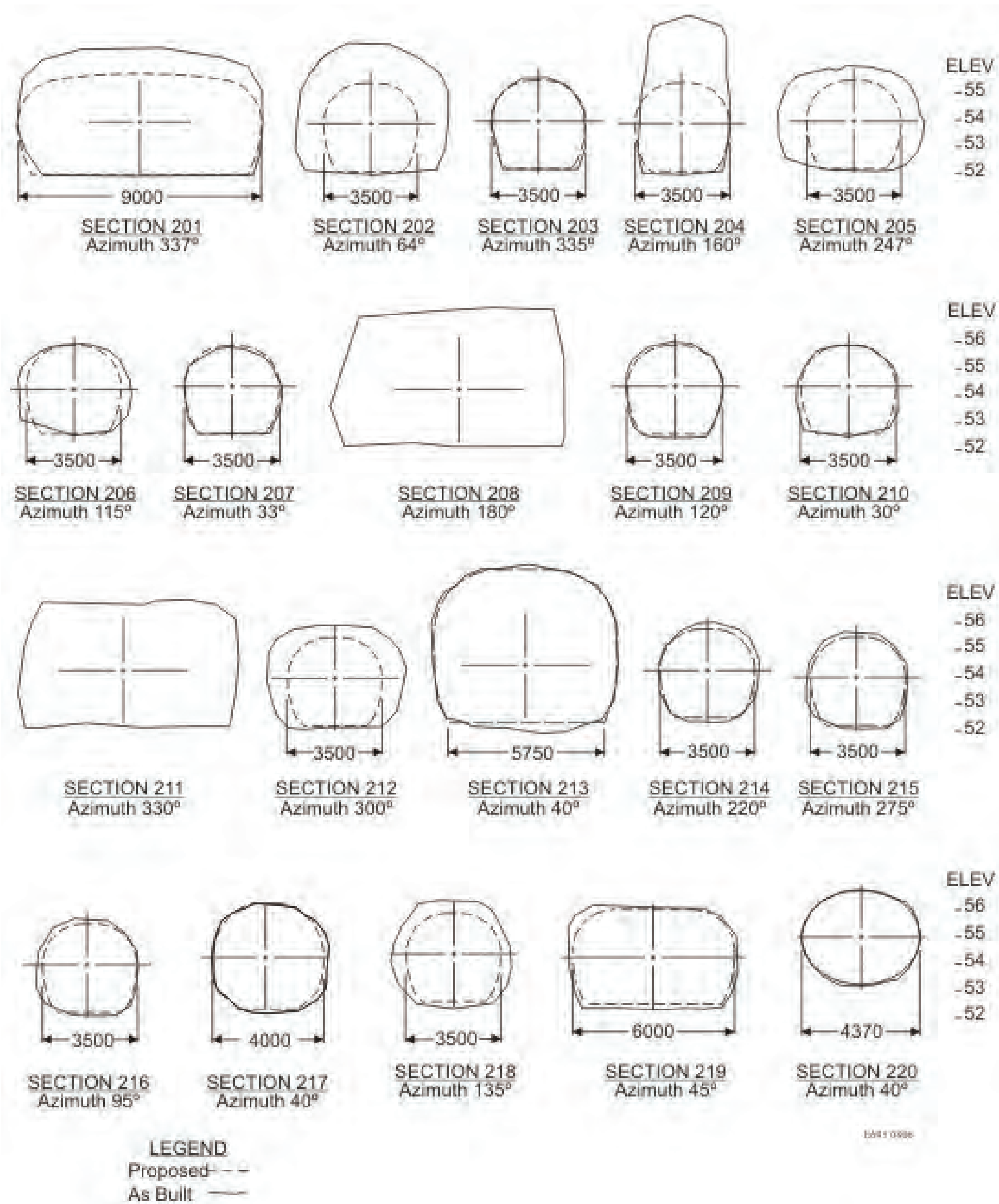


Figure 22: 240 Level Room Shapes. Elevations are above mean sea level. URL Shaft collar is at 289.9 m elevation.

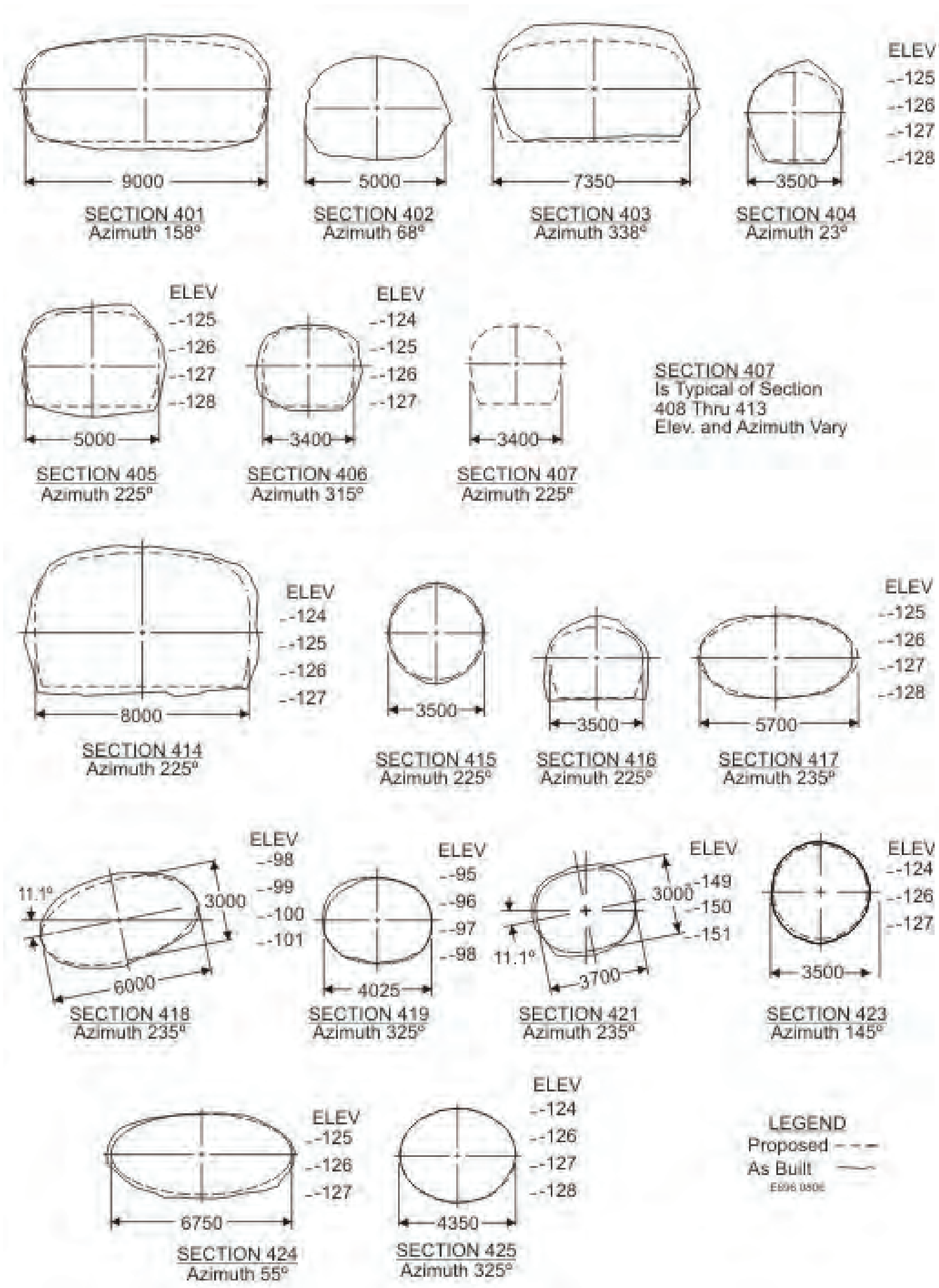


Figure 23: 420 Level Room Shapes. Elevations are above mean sea level. URL Shaft collar is at 289.9 m elevation.

Stability Observations by Room

In this section, scaling records for each horizontal excavation are related to known environmental conditions or other activities in the URL. In general the trend in all horizontal excavations was for decreasing scaling (assumed to be stress-induced damage activity) with time following excavation. In some tunnels, the initial adjustment period of the rock mass to the presence of the openings was not recorded, as scaling records were not kept during that early period. At this time the excavations were limited to the 240 Level and tended to be very stable with limited loose development or activity. When higher in situ stresses were encountered during the shaft extension, spalling and loose development in the shaft and lower levels became apparent, therefore scaling records were kept from the late 1980's to 2002 (scaling was a continuing activity and not all areas were scaled at once). Scaling records were not kept during decommissioning and removal of underground furnishings. Other tunnels show some increases over a period of time, which can be related to experimental work. Figure 3 provides a reference for the room numbers in the URL. The plots of volume of rock scaled versus time are presented in Appendix B.

130 Level Shaft Station

Room 130 is the drilling station nearest to the surface and likely to be most effected by changes in humidity and temperature. Temperature was recorded in shaft instrumentation arrays (e.g., Martino 1995) and for various experiments. The air humidity was recorded for the Buffer/Container Experiment (Graham et al. 1997)) but not for the facility as a whole. In general, based on observations, conditions tended to be more humid in summer months (May to August) and drier in the winter months. Intake air temperature (based on shaft arrays) was typically also higher for the summer months, and showed a cyclic pattern over the seasons. The amount of material scaled from Room 130 remained relatively constant and did not exhibit a reduction in scaled material with time. This may be due in part to its proximity to the incoming airflow and therefore the annual variations of temperature and humidity conditions, but the rock conditions were also different from other levels of the URL (more joint sets existed, and rock heterogeneity was greater). Additionally, controlled drilling and blasting was not employed at the 130 Level so more rock damage would have been present initially. It is important note that although scaling was required throughout the operational life of the URL, no increasing instability was noted (Appendix B, Figure B.1).

240 Level

Room 201 is the shaft station on the 240 Level and is one of the larger open spans at this depth. Scaling was not consistently recorded until four to five years after the first excavation of this opening. An intensive scaling effort occurred in 1989 and thereafter scaling was done only as required. An average of four to five small pieces of loose were scaled each year with occasional drummy (hollow sounding) areas noted. One corner of this room displays spalling structures (Figure 24), which are characteristic of highly stressed rock, similar to that seen in Mine-by tunnel (Room 405) on the 420 Level (Read 1996). This damage results from high stress concentrations at this inner corner. Despite this stress concentration, the damage remained isolated in the corner of the room and did not lead to further instability in the room (Appendix B, Figure B.2).

Room 203 is to the north end of 201. Generally only a few small pieces of loose were scaled from this room (Appendix B, Figure B.3).

Room 204 is a service area near the station. This room is at the start of the airflow path on the level and thus would be exposed to the largest variations of temperature and humidity on the level after Room 201. Multiple small pieces of loose occurred shortly in late 1988, and the amount of scaling generally decreased until 2001. The amount of scaled loose appeared to be increasing slightly through 2005, but the amount was minor. One or two larger pieces of loose were removed almost every year (Appendix B, Figure B.4).

Figure 24: Stress Induced Damage at Inner Corner of Room 201

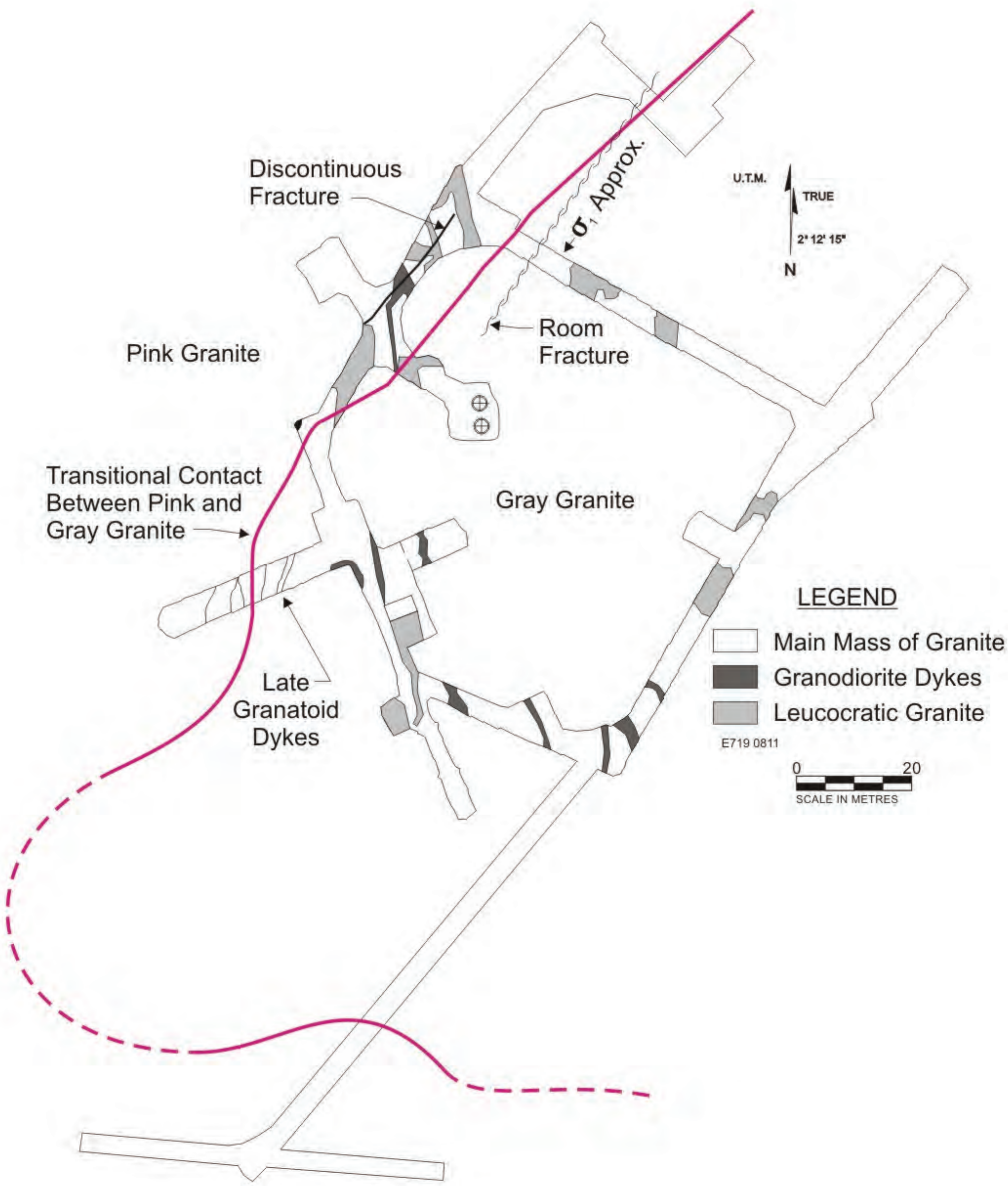
Room 205 is the refuge station and is located immediately off Room 201. The amount of scaled loose decreased over time. Occasional larger pieces were removed (Appendix B, Figure B.5).

Room 206 is one of the first sections of the loop tunnels on the 240 Level excavated and is along the fresh air circulation path, immediately after Room 204. The number of small pieces of loose removed increased slightly in the mid-1990's but then decreased again, this may be related to experiment work occurring in the mid-1990's on this level requiring an increased air circulation rate (Appendix B, Figure B.6).

Room 207 is another loop tunnel on the 240 Level and shows a similar pattern to Room 206 in terms of small loose pieces (Appendix B, Figure B.7).

Room 208 is the ventilation raise connection room (Room 002 to surface and Room 003 from the 420 Level) on the 240 Level and shows no clear pattern of the number of pieces scaled. This is likely due to the ongoing airflow and known changing humidity conditions in the room. Condensation was a common occurrence in portions of the room. It should be noted that in spite of this condition the room remained stable throughout its operational lifetime (Appendix B, Figure B.8).

Room 209 is another section of the air circulation loop on the 240 Level and shows a relatively large number of loose pieces with an increase in the number of small loose pieces scaled in the mid-1990's and an ongoing lower number of larger loose pieces. The number of large pieces decreased from 1989 (Appendix B, Figure B.9). This tunnel was where full face blasts were first used at the URL, with excavation advances of one-third to one-half the tunnel diameter.



Note: See Figure 3 for Room Number

Figure 25: Geology of the 240 Level

Room 210 is another tunnel on the 240 Level loop and had similar amounts of loose material to Room 209 and showed a similar decrease in the number of large pieces from 1989 (Appendix B, Figure B.10). Room 210 and 209 are roughly perpendicular to each other.

The geology of the 240 Level is shown in Figure 25. An anisotropy in the orientation of microcracks was determined to exist based on geologic mapping and core samples (Figure 26). The microcracking does influence the stress orientations at this depth (Martin 1993(b)) but does not appear to have influenced the stability of the tunnels, perhaps because the stresses at the 240 Level at depth are relatively low.

Room 211 was an experiment room off the main loop and was in pink granite near FZ2. Room 211 generally showed a limited very limited amount of loose scaled (Appendix B, Figure B.11). An increase in the number of small pieces scaled was recorded after 2003. This may be related to URL decommissioning activities in the area.

Room 212 was a small experiment room off the main loop. It was fully screened to protect the experiments in the room and does not have any scaling records.

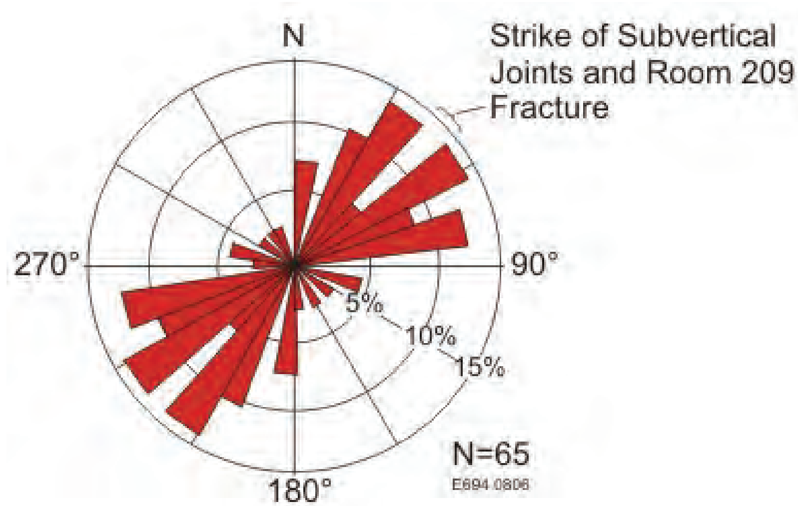


Figure 26: Rose Diagram Showing Orientation of Microcracks on the 240 Level. N is the number of microcracks in the sample plotted on the rose diagram. The percentage rings indicate the percent of the microcrack sample at a given orientation.

Room 213 is a large chamber that hosted the Buffer/Container Experiment. A large amount of loose was scaled in 1989 to 1990 (Appendix B, Figure B.12). The room was excavated in 1989 September and borehole drilling occurred between 1990 May and September. After 2001, a few larger pieces of loose were scaled each year. This scaling may relate to the extension of the room by construction of Room 220, the Blast Damage Assessment tunnel, and the associated redistribution of stresses and additional activities in Room 220 for several years.

Room 214 is an access tunnel to the Moderately Fractured Rock (MFR) experimental area and has a relatively high number of small pieces of loose scaled. Several peaks in the material scaled occurred in 1991, after construction, and in 1994 to 1995 during work on the MFR experiment (Figure B.13). These peaks in scaling are thought to be related to diligence in scaling rather than instability in this tunnel.

Room 215 and Room 216 are instrument galleries for the MFR and had much lower amounts of scaling compared to Room 214 (Appendix B, Figures B.14 and B.15). These tunnels were excavated using long blast round technology but there is no evidence collected that suggests an effect on the stability from this method of excavation.

Room 217 was constructed in 1994 and extends from room 209 to provide access to an experimental area. It starts with a higher amount of loose scaled which decreases over time (Appendix B, Figure B.16). This room has a wider shape at one end and is in close proximity to a vertical fracture. The wider shape may have contributed somewhat to the amount of loose but does not appear to have effect long term stability.

Room 218 is an experimental area immediately adjacent to a vertical fracture. One end of this room is wider. The amount of loose was generally moderate and decreased with time; however, in the year after construction one scaling activity produced a high (26 pieces) of loose (Appendix B, Figure B.17). It is thought that the proximity to the fracture, wider cross section and need to install equipment in a stable area all contributed to the higher amount of scaling in that period.

Room 219 extends from Room 218. Room 219 had a limited amount of loose material scaled and showed a decreasing trend except for a minor increase in scaling in 2002-2003. This likely occurred during removal of equipment from this room (Appendix B, Figure B.18).

Room 220 was excavated in 2001 and only required a very limited amount of scaling before access to the room was closed in 2005 and scaling was no longer performed (Appendix B, Figure B.19).

300 Level Shaft Station

The 300 Level is the shaft station at 300 m depth. It is in the higher in situ stress regime at the URL. Unlike the rooms on the 240 Level, mesh screening was installed on the crown and included grouted anchor bolts at pre-determined locations are used for ground control to provide safe working environments and rock support on this level. It shows a clear decrease in loose material scaled with time (Appendix B, Figure B.20).

420 Level

Rooms on the 420 Level are in the higher in situ stress regime at the URL. Like the 300 Level, mesh screening was installed on the crown and included grouted anchor bolts at pre-determined locations are used for ground control to provide safe working environments and rock support on these levels.

Room 401 is the shaft station on the 420 Level. A relatively large amount of loose material was scaled during the first years after construction (Appendix B, Figure B.21). After that period, the amount of loose material culled decreased significantly to a level of a few pieces per year. This indicates that although the level experienced higher stress conditions, the excavation remained stable.

Room 402 is the electrical substation and had a relatively large amount of scaling immediately after excavation but no scaling since that time because the walls and back were screened to protect the equipment (Appendix B, Figure B.22). The room is relatively small and the door generally kept closed. The room is kept at a slightly elevated constant temperature by the electrical substation equipment in it.

Room 403 is attached to Room 401 and is basically an extension of Room 401. As with Room 401 and 402, a relatively large amount of scaling was done initially with only minor amounts of scaling required after that time (Appendix B, Figure B.23).

Room 404 is the access tunnel to the ventilation raise (Room 003) from the 420 Level. As this tunnel receives exhaust air from the level (slight variations in temperature and humidity) some additional loose material may be expected. The room is also an escape route and may receive more attention from miners performing the scaling for this reason. The amount of loose scaled follows a similar pattern to the surrounding rooms but does show an increase in loose material scaled beginning in 2001 (Appendix B, Figure B.24).

Room 405 was an instrument gallery for the Mine-by Experiment (Room 415) and then an access tunnel to the Tunnel Sealing Experiment tunnel (Room 425). The amount of loose material scaled varies with time but shows no clear pattern or decrease in scaled material (Appendix B, Figure B.25). However, no scaling was required after the completion of the TSX. See also Rooms 417, 423 and 424, which were access tunnels closer to the TSX experiment.

Rooms 406, 407, and 408 form sections of the incline ramp and connect to Room 409 at the top of the incline. All are on the exhaust end of airflow on the level. All showed similar patterns of slowly decreasing amounts of loose material with time (Appendix B, Figures B.26 to B.29).

Rooms 410, 411, and 412 form sections of the decline ramp and connect to Room 413 at the bottom of the decline. All are on the intake section of airflow on the level. All showed similar patterns of slowly decreasing amounts of loose material with time (Appendix B, Figures B.30 to B.33). Rooms 410 and 411 show similar patterns of an initial decrease followed by several years of approximately constant levels of scaling and then another decrease. Room 412 shows a more rapid decrease in the amount of loose scaled with time. Room 413 showed a much lower level of loose material scaled with time.

Room 414 is a large opening near the level station that had very little loose material scaled from it (Appendix B, Figure B.34).

Room 415 is the Mine-by Experiment tunnel, a circular experimental tunnel (Read 1996) designed to have a large excavation response. This room was mechanically excavated using line-drilling and hydraulic rock splitters. Notches formed in the floor and roof and required scaling. This material was not counted in the scaling totals for this room. The scaling data only include work done to ensure safety in the room (Appendix B, Figure B.35). The amount of loose material showed a rapid decrease to a minor level, indicating that even in a tunnel purposely excavated to have a strong excavation response, the rock around the tunnel will stabilize. During the excavation of Room 415, a series of surveys was conducted that showed the evolution of the break outs in the tunnel with time, prior to closure of access to the tunnel and final survey was done, which showed only minor changes over a period of twenty years (Figure 27). Considering that the tunnel was specifically designed to create a large excavation response in the stress conditions at the 420 Level, its ongoing stability is a positive indicator.

Room 417 was an access tunnel to the Tunnel Sealing Experiment, but was excavated as part of the Excavation Stability Study (Read et al. 1997a). Two-thirds of the tunnel had an oval cross-section with the long-axis of the oval being horizontal. It is worth noting that the maximum principal stress alignment was perpendicular to the tunnel axis but 14 degrees off horizontal, resulting in localised stress concentrations in the tunnel roof and floor. The room experienced a large slab failure due to variations in rock type in one location. Overall the amount of loose material scaled was low and decreased over time (Appendix B, Figure B.36).

Room 418 was part of the Excavation Stability Study as well and was excavated using three different oval cross-sections, similar to Room 417 but with the long axis of the oval oriented 14° from horizontal to be aligned with the major principal in situ stress direction. The amount of loose scaled from the room showed a similar pattern to that of Room 417 but did have some additional scaling of small pieces of loose in 1999, 2000 and 2001 (Appendix B, Figure B.37).

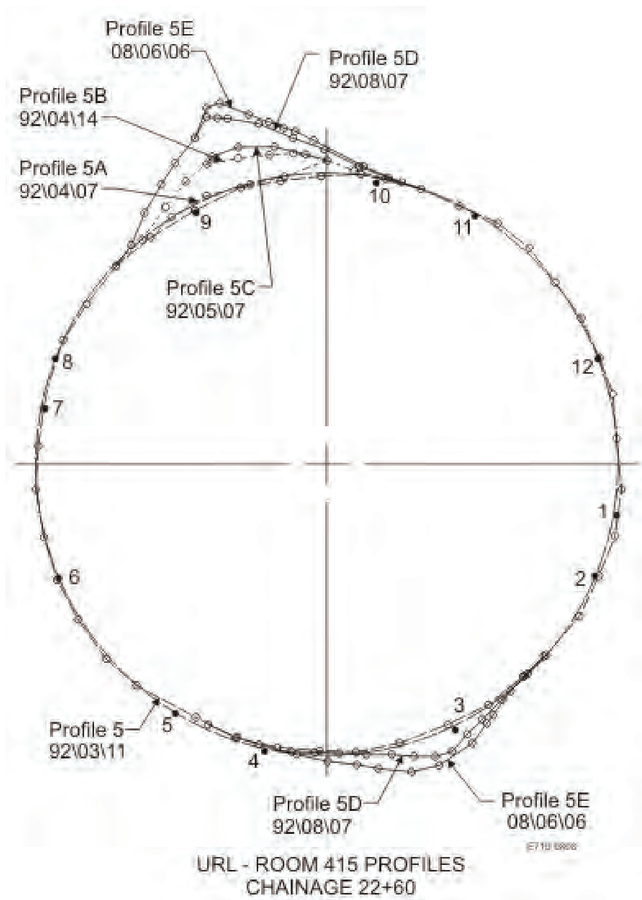


Figure 27: Time Lapse Survey in Circular Tunnel on the 420 Level of the URL Showing Only a Minor Change in Profile Over 17 Years

Room 419 extended 90 degrees from the end of Room 418. The amount of scaling in this room was low but showed no clear pattern with time (Appendix B, Figure B.38).

Room 421, also part of the Excavation Stability Study, was excavated using three different oval cross-sections, the long axis of each being oriented at an angle of 14° from horizontal and aligned with the major principal stress. These tunnels had a very low width-to-height ratio relative to other tunnels in the Excavation Stability Study (compare with Room 417 and 418 in Figure 23) resulting in higher stresses in the roof and floor, although these stresses were not concentrated at a point but distributed over a larger area. Room 421 showed no clear pattern to the amount of loose material scaled with time. There was a slightly higher amount of larger loose pieces scaled from this room; this may be because Room 421 was used as the sump for the level and moisture conditions tended to be higher and more variable than other locations (based on observations of the presence of water at the end of the room and damp areas on the walls) or due to higher stresses from the tunnel shape (Appendix B, Figure B.39).

Room 423 is a circular tunnel and was an access tunnel that paralleled the TSX tunnel, Room 425. In Room 423 only minor scaling was required and there was an increase in the amount of loose scaled from 2002 to 2004 when the adjacent TSX tunnel (Room 425) was heated (Appendix B, Figure B.40).

Room 424 is a wide access tunnel having an oval cross-section connecting Rooms 423 and 425. Only minor scaling was required in this room (Appendix B, Figure B.41).

Room 425 records have a large gap in scaling when the room was filled with sand and pressurized water during the TSX (Appendix B, Figure B.42). The amount of loose scaled does not include spalled rock caused by heating of the tunnel that was removed during decommissioning of the experiment (Martino et al. 2008). Once this spalled material was removed only minor scaling was required during the remainder of decommissioning. During heating and saturation of the TSX tunnel, water pressure provided a confining pressure to the roof, sand fill in the tunnel also provided a confining pressure by its weight. It is believed that the sand settled slightly during the experiment and when the tunnel was depressurized, and after the sand was removed, the spalling began to occur in the roof. The spalling was of limited extent and self stabilized. This suggests that in the changing temperature conditions some localized development of spalling and loose will occur, but the rock mass will remain stable.

Microseismic and Acoustic Emission Monitoring

During the later half of the URL operation a microseismic system was in operation and was capable of recording hammer blow energy release levels from rock damage on the 420 Level. The majority of this information remains to be analyzed but evidence from the Mine-by tunnel (Martino et al. 1993) shows that the amount of activity occurred largely around the face of the advancing tunnel and then decreased with time. This indicates a stabilizing tunnel, even when that tunnel (Room 415) was designed to elicit a large excavation response.

The decreasing energy release events (i.e., damage) with time is also shown by acoustic emission systems capable of detecting energy released during microcrack formation having similar seismic energy released as that of a pencil lead break. One such system was installed in the Heated Failure Tests (Read et al. 1997b) and the results indicated a decrease in events with time, even though the rock mass had been heated.

The combination of information from scaling records, microseismic response and acoustic emission response data indicate that the URL tunnels show no evidence of increasing instability. The URL provides evidence that tunnels excavated in stable rock masses, whether moderately or highly stressed, are stable for 20 years and should remain stable over longer time periods if environmental conditions are stable.

4. CONCLUSIONS

Construction of the URL in a previously undisturbed and well-characterized granite rock mass was without precedent. Further, the decision to integrate geotechnical studies with the excavation and construction activities required a significant planning and coordination effort. Some of the lessons learned during the construction of the URL are identified below.

Excavation Method

It was recognized during the planning of the excavations for the URL shafts, levels and test rooms that a drilling and blasting method of excavation would be more versatile and cost effective that a mechanical boring method in granite rock. It was also deemed important that damage to the rock mass surrounding the excavations caused by excavation and subsequent stress redistribution could be both determined and minimized. The excavation induced damage could form hydraulic pathways that might complicate or reduce the effectiveness of tunnel and shaft sealing systems, thus, ultimately becoming an issue for repository sealing.

Attempts at controlled drilling and blasting during the initial shaft-sinking project (surface to 255-m depth) were only moderately successful. The rectangular configuration and the benching method of excavation used by the shaft-sinking Contractor did not facilitate the application of controlled drilling and blasting principles. Subsequently, during the excavation of the 240 Level and shaft extension (255-m to 443-m depth), the excavation designs were adapted to facilitate application of the methodology for controlled drilling and blasting.

A pilot-and-slash technique was initially used for excavations on the 240 Level, which significantly reduced damage to the rock mass surrounding excavations and the need for permanent ground control and ground support. The pilot and slash excavation method was time consuming, e.g., temporary services (air and water lines, ventilation, rail, etc.) and in some cases ground control had to be re-done after the slash was taken. However, the pilot and slash method gave good results in terms of reducing excavation damage. Next, a full-face method was developed, which maintained an acceptable level of quality in terms of limiting excavation damage, and improved productivity. The careful full-face blasting method was important in dealing effectively with high rock stresses encountered in the shaft extension, the 300-Level shaft stations and the initial 420 Level developments.

Observational Method

The acceptance of the “Observational Method” was an important factor in realizing success in the project. This iterative approach to design and construction, which is applied to underground civil engineering projects, encouraged pre-planning for expected, and to some extent unexpected, situations that could arise as the project progressed.

The Observational Method was the basis for managing the development do the URL but in some cases, particularly as it applied to the geotechnical characterization activities, it was applied as “design as you go”. The potential for a situation to arise was considered but the approach to dealing with the situation was left open until the in situ conditions could be evaluated first-hand and the solution that had the least impact on the R&D objectives could be developed.

One example of this was the application of appropriate ground support. It was important, for the sake of the geotechnical studies, to provide unimpeded access to the rock surfaces during excavation. Under normal conditions, ground support could interfere with geological mapping and the acquisition of geotechnical data. By developing the appropriate ground control when the situation could be assessed directly, the R&D objectives could be better satisfied.

Shaft Sinking Equipment and Methodology

Innovative equipment can facilitate achieving the R&D objectives and can more effectively integrate the R&D activities and the excavation project. A Galloway Stage was designed to integrate geotechnical and construction activities during sinking of the lower shaft. The design for the stage evolved from experience gained during the upper shaft-sinking project. The Galloway stage minimized the effort needed to ensure a safe working environment was provided for the technical and construction groups an facilitated both groups in conducting their work.

The shaft configuration was changed from a rectangular cross-section using a traditional bench blasting method to circular cross-section using a full-face blasting technique employing a burn-cut. The full-face method allowed for variable length blast rounds, a simpler shaft profile and a relatively flat face at shaft bottom, which were beneficial to the geotechnical characterization activities. The circular shaft cross section better suited the highly stressed rock conditions anticipated below the 240 Level. The controlled drilling and blasting results, in terms of the quality of the rock surface, during the shaft extension were superior to the benching method used previously.

Contract Format

The decision to use a cost plus fixed fee contract format in the shaft extension project was based upon experience gained during the initial shaft sinking and recommendations from independent Contractors. Unlike the unit rate contract used in the upper shaft, this format reduced the risk to the Contractor (e.g., fixed fee was assured to the Contractor, blast design risk was assumed by AECL, the Contractor understood much more clearly the R&D aspects of the scope of work, the miners' bonus was based on objectives of both the Contractor and AECL) and provided more control for AECL's construction management group (e.g., implications of delays were better defined). The contract provided for a change control process to accommodate design changes and was flexible enough to better accommodate the R&D activities.

Incentive System

An incentive system based upon quality of work, meeting schedule objectives and cooperation and support for the geotechnical program proved to be an important factor in guiding the Contractor's crews during the URL shaft extension project.

In Canada, the Contractors offer a bonus system to attract suitable miners for shaft sinking and tunnelling. The bonus systems are customarily based upon speed of construction using standard methods, as opposed integrating with and supporting an R&D component to the project. The normal bonus approach was used during the excavation and servicing of the upper shaft and this put the Contractor's construction objectives and AECL's R&D objectives in conflict.

In the shaft extension project, to encourage cooperation and acceptance of new techniques, a quality-based incentive system for the miners was devised. Once the Contractor and the miners realized the nature of the URL project and understood the bonus incentives established for achieving project objectives, they were fully cooperative. With reliable records and assurance that the crews were precisely following the intended designs (see Quality Control below), the blast design engineer was able to progressively refine the designs to achieve better results.

Quality Control and Inspection

Quality control and inspection procedures implemented by AECL subsequent to the upper shaft sinking contract were instrumental in controlling the work done by the contractors and insuring that technical objectives were achieved, AECL's quality control inspectors were placed on each shift to work with the Contractor's crews. They kept independent, accurate records of the work done and the results achieved.

Excavation Stability

Overall, the tunnels at the URL required only a small amount of scaling in order to maintain safe conditions in over 20 years of operation. The minor amount of scaling demonstrates that the tunnels were stable and showed no evidence that damage was increasing with time and in fact, with stable environmental conditions, generally showed a decrease to a minimal level of scaling.

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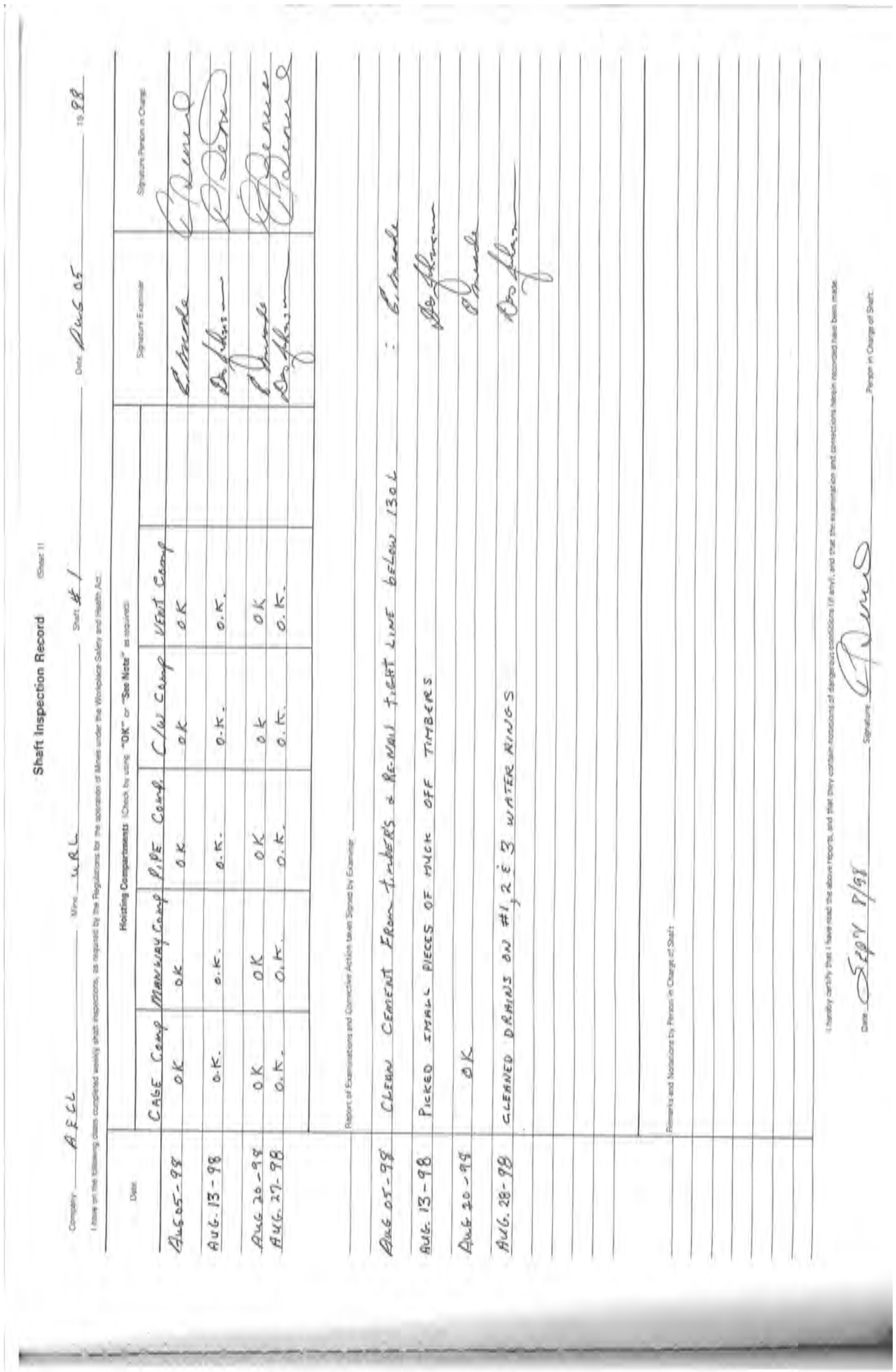
APPENDIX A: SAMPLE WORK MANAGEMENT AND QUALITY CONTROL INSPECTION FORMS

CONTENTS

Page

A.1 SAMPLE PAGE OF PROVINCIAL LOGBOOK



Shaft Inspection Record (Sheet 1)

Company: AECL Mine: URL Shaft: #1 Date: Aug 05 19 98

I have on the following dates completed weekly shaft inspections, as required by the Regulations for the operation of Mines under the Workplace Safety and Health Act.

Date	Hoisting Compartments (Check by using "OK" or "See Note" as required)						Signature Examiner	Signature Person in Charge
	CAGE Comp	MANWAY Comp	PIPE Comp.	C/W Comp	VENT Comp			
Aug 05-98	OK	OK	OK	OK	OK			
AUG. 13-98	O.K.	O.K.	O.K.	O.K.	O.K.			
Aug 20-98	OK	OK	OK	OK	OK			
AUG. 27-98	O.K.	O.K.	O.K.	O.K.	O.K.			

Report of Examinations and Corrective Action taken Signed by Examiner

Aug 05-98	CLEAN CEMENT FROM TIMBER'S & RE-NAIL TIGHT LINE BELOW 130L
AUG. 13-98	PICKED SMALL PIECES OF MUCK OFF TIMBERS
Aug 20-98	OK
AUG. 28-98	CLEANED DRAINS ON #1, 2 & 3 WATER RINGS

Remarks and Notations by Person in Charge of Shaft

I hereby certify that I have read the above reports, and that they contain notations of dangerous conditions (if any), and that the examination and corrections herein recorded have been made.

Date: Sept 8/98 Signature: ____ Person in Charge of Shaft

Figure A.1: Sample Page from Shaft Logbook

A.2 EXCAVATION ADVANCE FORMS

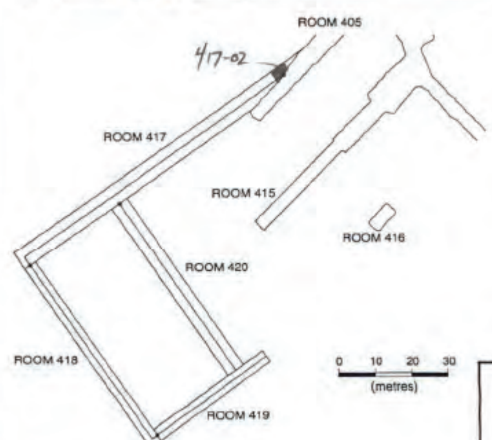
These forms are examples of the forms used to record blasting information during shaft and tunnel advance

AECL RESEARCH
IVS EXPERIMENT
TUNNEL EXCAVATION
ACTIVITY RECORD

Date: 95-12-20
Inspector: SK
Contractor: J.S.Redpath
Shift Leader: EM

ACTIVITY	9 10 11 12 13 14 15	TOTAL	REMARKS
1. Survey / Mark-up		3	PICK-UP 418-02, MARK-UP 418-03 PICK-UP 417-01
2. Blast Round Inspection		1/4	418-02
3. Blasthole Drilling 41, 43, 45, 51 mm diameter		3 3/4	COMPLETE DRILLING ON 417-02 SLASH
4. Reaming - 100 mm			
5. Loading and Hook-up			Blast # : Time :
6. Blast Monitoring (BMX)			
7. Ventilate / Washing			
8. Scaling			
9. Mucking		1 1/4	MUCK-OUT 418-02 FOR SURVEY.
10. Ground Control			
11. Photography			
12. Experimental Delays			
13. Operations Delays			
14. Mechanical Delays			
15. Other Delays			
16. Lunch		1/2	
17. Other Work (specify)			
18. Other Work (specify)			

COMMENTS:



Signed: SKeith
AECL INSPECTOR

Figure A. 2: Sample Page from Tunnel Excavation Activity Record

AECL RESEARCH
IVS EXPERIMENT
TUNNEL EXCAVATION
SHIFT SUMMARY

Date: 95-12-20
Inspector: SK
Contractor: J.S.Redpath
Shift Leader: EM.

ADVANCE	BLAST NUMBER	417-02		
	CHAINAGE START	2.10 m	m	m
	CHAINAGE END	— m	m	m
	CHAINAGE ADVANCE	— m	m	m

DRILLING	NUMBER OF PRODUCTION BLASTHOLES DRILLED	9		
	NUMBER OF CUSHION BLASTHOLES DRILLED	13		
	NUMBER OF PERIMETER BLASTHOLES DRILLED	11		
	NUMBER OF LIFTERS DRILLED	6		
	NUMBER OF RELIEF HOLES DRILLED	0		
LOADING	NUMBER OF PRODUCTION BLASTHOLES LOADED	—		
	NUMBER OF CUSHION BLASTHOLES LOADED	—		
	NUMBER OF PERIMETER BLASTHOLES LOADED	—		
	NUMBER OF LIFTERS LOADED	—		
BLASTING	BLAST TYPE (FULL FACE, SLASH, PILOT, REBLAST)	SLASH		
	BLAST NUMBER AND TIME 417-02	—		
	PERCENT HALF-BARRELS	— %	%	%

MUCKING	REMOVAL OF MUCK (Number of Skips or Particulars)	MUCK OUT 418 FOR SURVEY.
	SCALING (Hours and Location)	✓
	FACE CLEAN-UP (Hours and Location)	✓ 418
GROUND SUPPORT	MESH ANCHORS INSTALLED	—
	RESIN BOLTS INSTALLED	—
	AREA OF WIRE MESH INSTALLED	— m^2

COMMENTS:

SIGNED: S Keith

Figure A. 3: Sample Page from Tunnel Excavation Activity Log Summary of Drilling

ATOMIC ENERGY OF CANADA LIMITED
URL SHAFT EXTENSION

BLAST SUMMARY
AND EVALUATION

BLAST NUMBER: 101
SHAFT DEPTH (m): 432.74
DATE OF BLAST: 09/07/88
TIME OF BLAST: 03:30
DESIGNED BY: D. ONAGI

COMMENTS: 1ST 3.5 M ROUND SINCE BLAST #73. ELECTRONIC CAPS USED. PARTICULAR ATTENTION PAID TO THE DRILLING, ALIGNMENT & LOADING OF THIS ROUND (AND SUBSEQUENT 3.5 m ROUNDS). DELAY PATTERN IN RING # 3 WAS DESIGNED DIFFERENTLY FOR THIS ROUND. THE 6 #16 DELAYS WERE DISTRIBUTED AROUND THE RING INSTEAD OF ALL BEING GROUPED TOGETHER. BLAST WAS MONITORED BY CL

CALCULATIONS:

AVERAGE DRILLED DEPTH	=	3.51 METRES
SURVEYED ADVANCE	=	3.37 METRES
DESIGN VOLUME	=	58.3 CUBIC METRES
VOLUME EXCAVATED	=	60.1 CUBIC METRES
TOTAL LENGTH OF BOREHOLE TRACES	=	51.0 METRES
PERCENTAGE OF HALF BARRELS	=	47%
PERCENTAGE PULL	=	96%

EXPLOSIVE CALCULATIONS:

Description	Design	As Built
WT. XACTEX (PERIMETER)	31.68 KGS	31.57 KGS
WT. F25x200 (PERIMETER)	13.06 KGS	13.06 KGS
WT. F25x200 (PRODUCTION)	44.06 KGS	41.82 KGS
WT. F32 (PRODUCTION)	104.30 KGS	106.00 KGS
TOTAL WEIGHT:	193.10 KGS	192.45 KGS
POWDER FACTORS:	3.31 KG/CU.M	3.30 KG/CU.M
POWDER FACTOR (ANFO):	3.23 KG/CU.M	3.22 KG/CU.M
WALL SHEAR FACTORS:	0.88 KG/SQ.M	0.88 KG/SQ.M

CONCLUSIONS: BLAST QUALITY EXCELLENT. 96% PULL & 47% HALF BARRELS. PROBABLY A COMBINATION OF ELECTRONIC DETS & ATTENTION PAID TO DRILLING & LOADING. PROBLEM OF "CROSS TALK" BETWEEN DET'S APPEARS TO HAVE BEEN ELIMINATED BY SHORTENING LEG WIRES FROM 15 TO 5 M.

01000-U-88

SIGNATURE: Dwayne Onagi

URL OPS
VOLUME 51
PAGE 197

88-U-00010

89-U-00010

Figure A. 4: Sample Page of Shaft Blast Round Summary

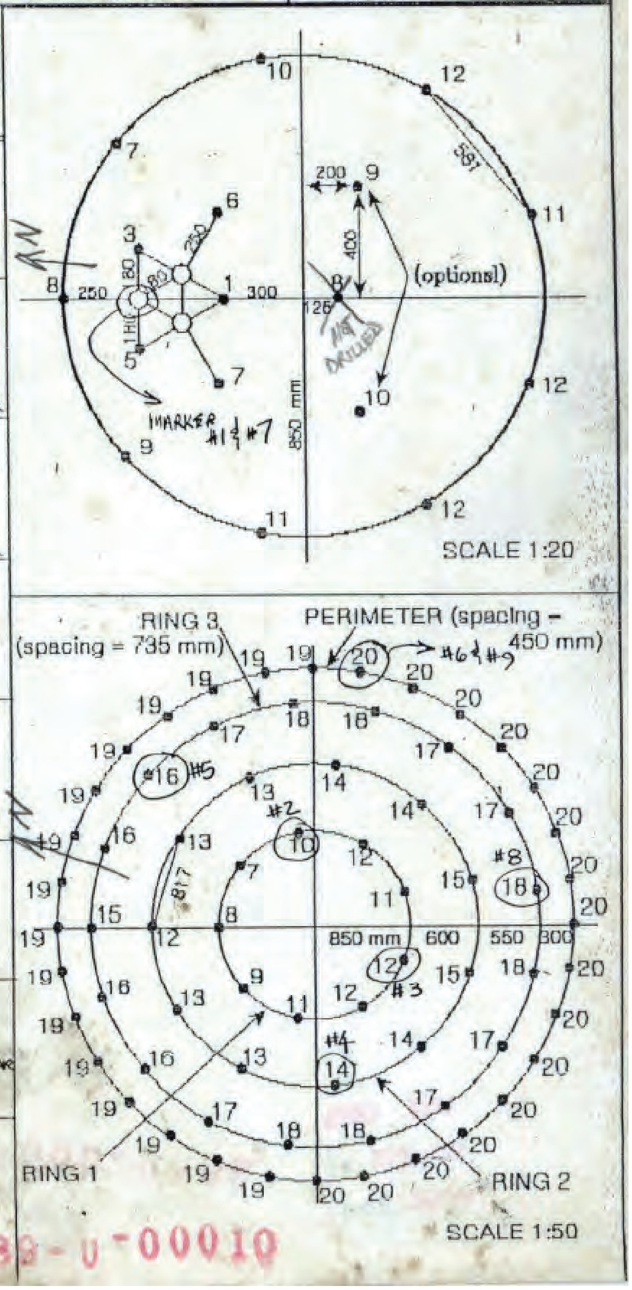
ATOMIC ENERGY OF CANADA LIMITED
URL SHAFT EXTENSION

BLAST DESIGN NUMBER : 101

Blast No. : 101
Date: 88-07-08
Designed by: D PO.

Contractor: J. S. Redpath

LOCATION	HOLE DIA. (mm)	HOLE DEPTH (m)	DIP (DEG.)	NO. STICKS	LOADING DETAILS
RELIEF	89	3.7	90	1/3 F3	TO CLEAR WATER FROM RELIEF HOLES
CUT	88	3.5	90	9 F3	UNTAMPED EXCEPT FOR LAST STICK 100 MM COLLAR
HELPER	38	3.5	90	8 F3	UNTAMPED EXCEPT FOR FIRST 2 AND LAST 400 MM COLLAR
RING 1	38	3.5	90	9 F3	TAMP ALL 500 MM COLLAR
RING 2	38	3.5	90	9 F3	TAMP ALL 500 MM COLLAR
RING3	38	3.5	90	19 F2	TAMP EVERY 3 STICKS 400 MM COLLAR
PERIMETER	38	3.5	87	3/4.5 F2/x	UNTAMPED 200 MM COLLAR



COMMENTS
- ELECTRONIC CAPS
- NOTE CHANGES IN DELAYS

F3 = 32mm Forcite
F2 = 25mm Forcite

Figure A. 5: Sample Page of Shaft Blast Round Summary - Hole Layout

Atomic Energy of Canada Limited
URL Shaft Extension

LOADING PERMIT

Date: 88/JULY/09
Shift: GRAVEYARD

Contractor: J. S. Redpath Ltd.
Shift Leader: L. WILSON

BLAST NO.: 101

	YES	NO*
Hole Pattern as per Design (i.e. deviations <100mm)	☑	☐
Hole Alignment Accurate (i.e. deviations <2°)	☑	☐
Hole Location and Inclinations recorded	☑	☐
Correct and sufficient Explosives available	☑	☐
Correct and sufficient Detonators available	☑	☐
Blasting Box available and in working order	☑	☐

*If NO specify the Corrective Action

CORRECTIVE ACTION REQUIRED

Signed: ______ Inspector   Acknowledged: ______ Contractor

CORRECTIVE ACTION CARRIED OUT   Signed: ______ Inspector

	Forcite 32mm	Forcite 25mm	Xactex	Primaflex	B-Line	32x200
Explosives Down:	4 BOX	3 BOX	3 BOX	___	___	1 BOX 175
Explosives Up:	0	53 STICKS	1 BOX 48 STICKS	___	___	43 STICKS
Explosives Used:	220 STICKS	412 STICKS	152 STICKS	___	___	84 ST

Comments:

89-U-00010

BLAST IS APPROVED FOR LOADING EXPLOSIVES AT 1:00 HOURS

VOLUME 51
PAGE 201

Acknowledged: ______ Contractor   Signed: ______ Inspector

Figure A. 6: Sample Page of Shaft Blast Round Summary – Loading Permit

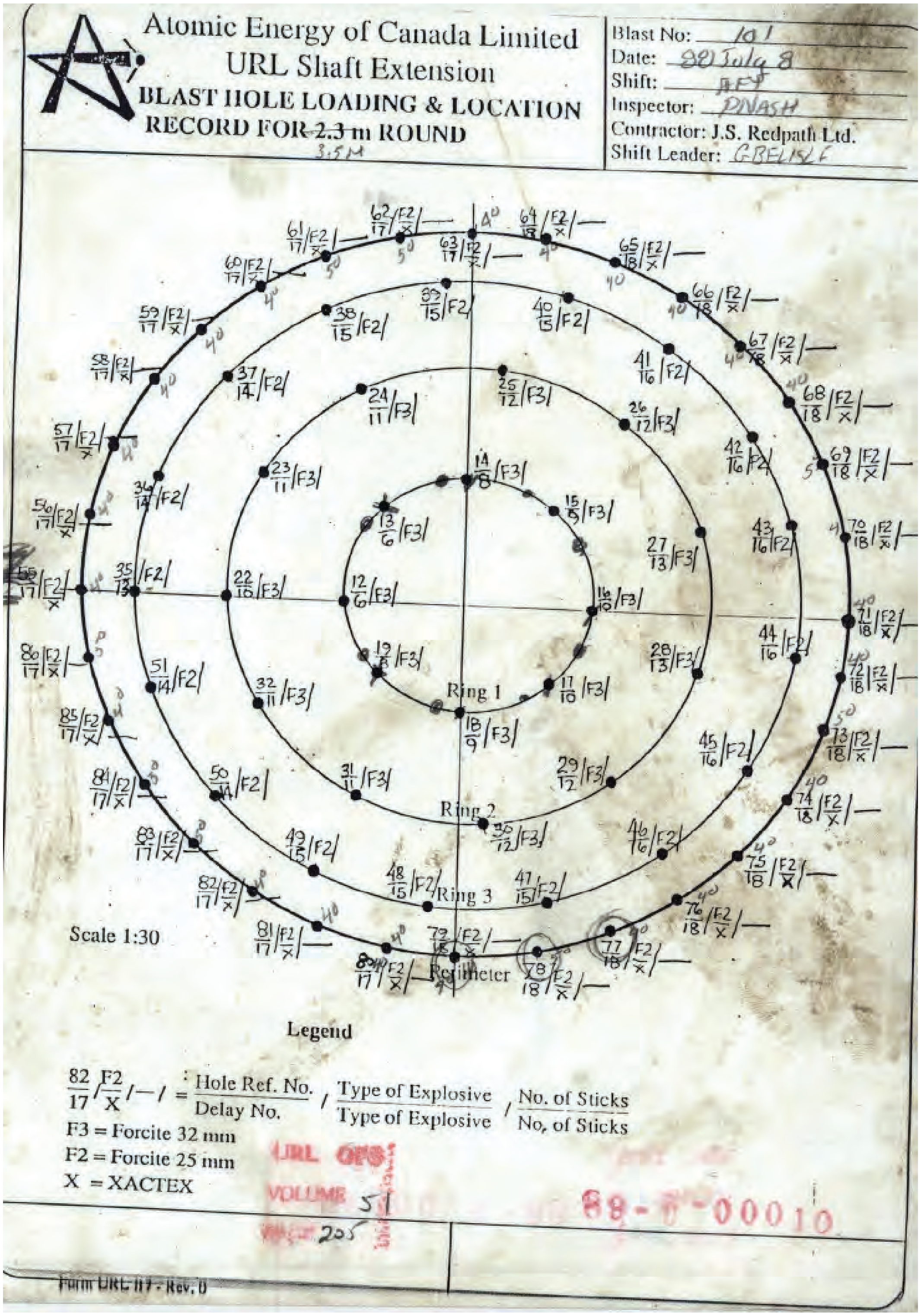


Figure A. 7: Sample Page of Shaft Blast Round Summary – Loading Locations

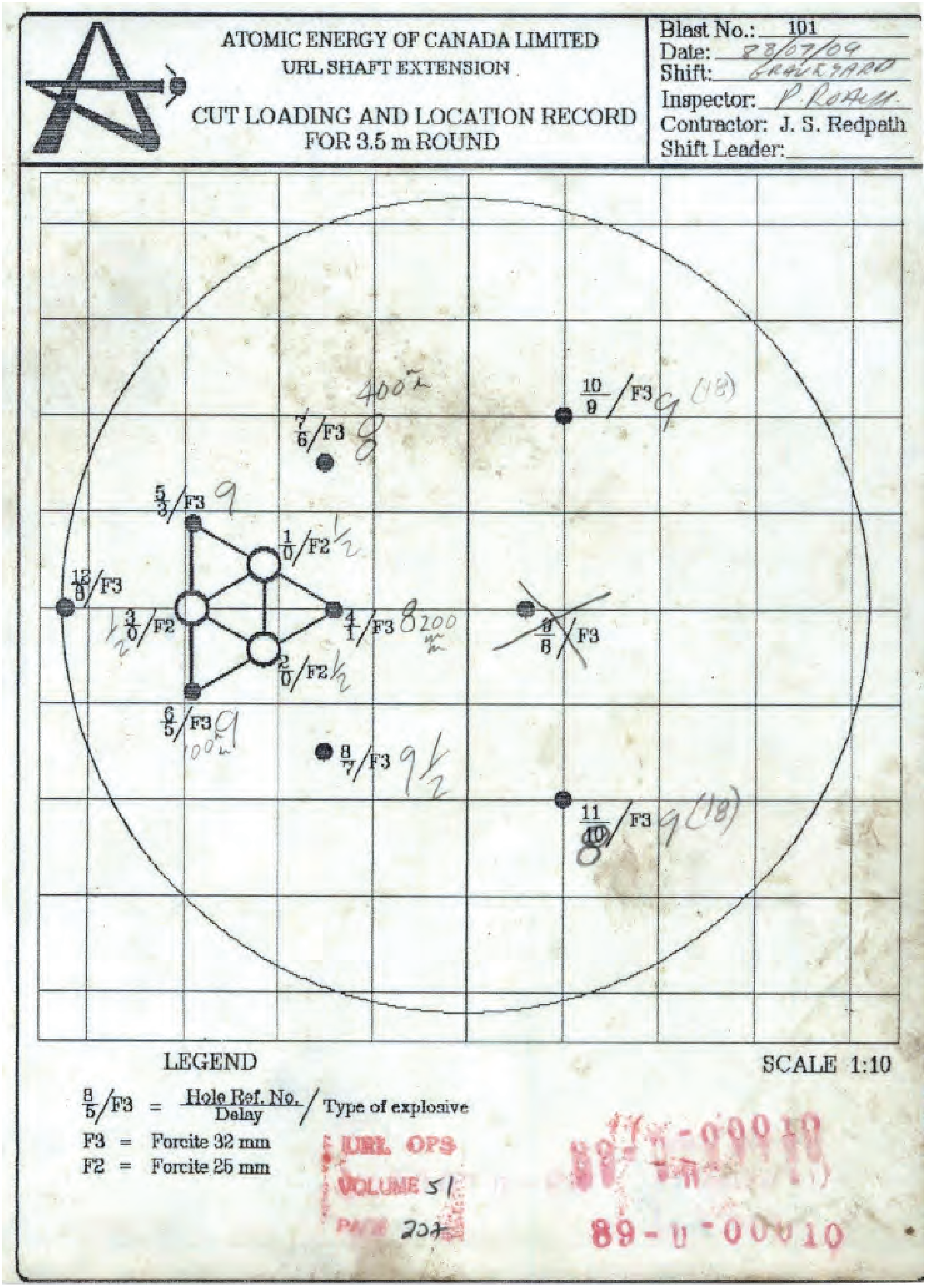
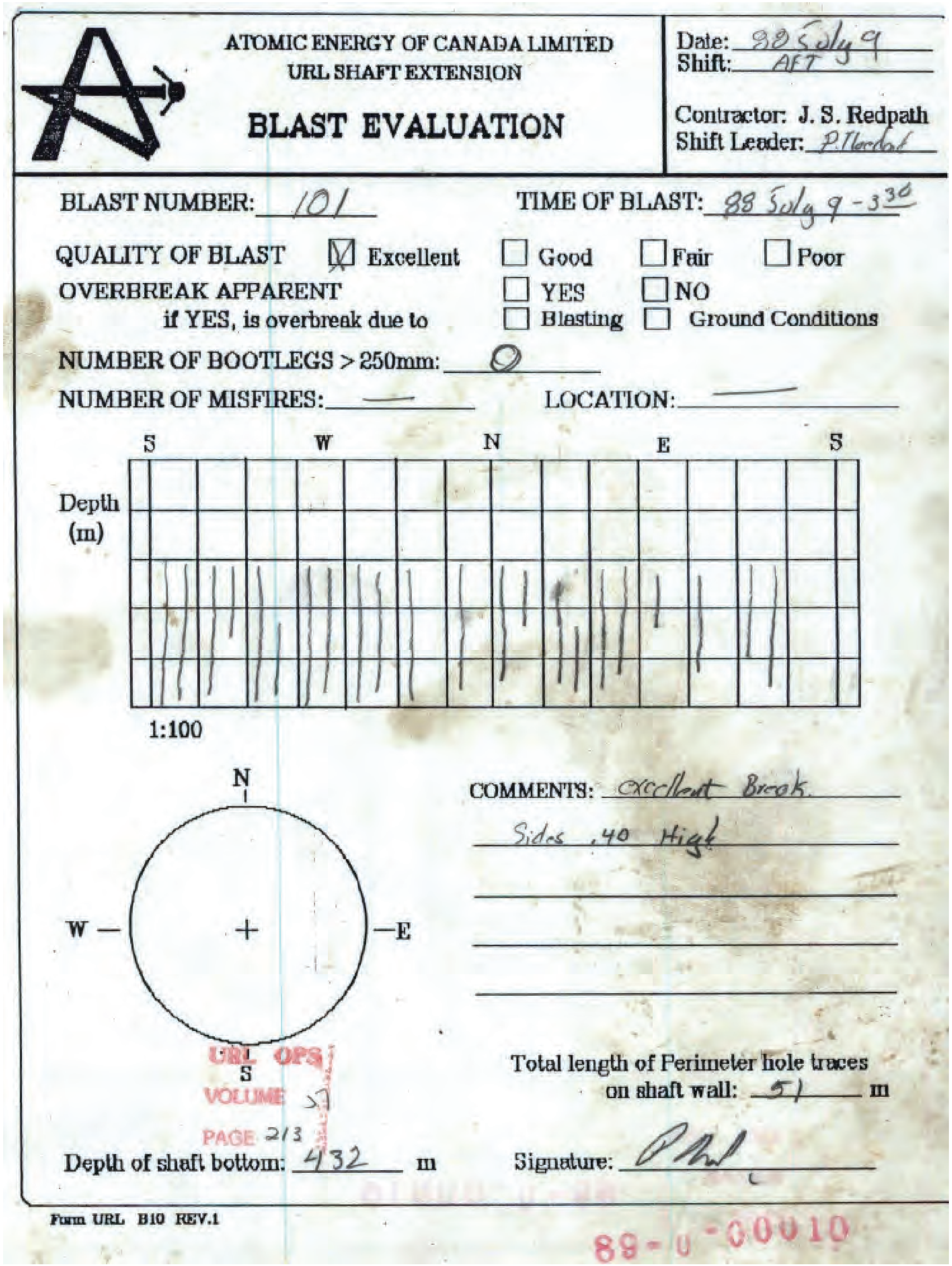


Figure A. 8: Sample Page of Shaft Blast Round Summary – Cut Hole Loading



ATOMIC ENERGY OF CANADA LIMITED
URL SHAFT EXTENSION

BLAST EVALUATION

Date: 88 July 9
Shift: AFT

Contractor: J. S. Redpath
Shift Leader: P.Tlordat

BLAST NUMBER: 101

TIME OF BLAST: 88 July 9 - 3 30

QUALITY OF BLAST ☒ Excellent ☐ Good ☐ Fair ☐ Poor

OVERBREAK APPARENT ☐ YES ☐ NO

if YES, is overbreak due to ☐ Blasting ☐ Ground Conditions

NUMBER OF BOOTLEGS > 250mm: 0

NUMBER OF MISFIRES: —

LOCATION: —

COMMENTS: excellent Break
Sides .40 High

Total length of Perimeter hole traces on shaft wall: 51 m

URL OPS
VOLUME 57
PAGE 213

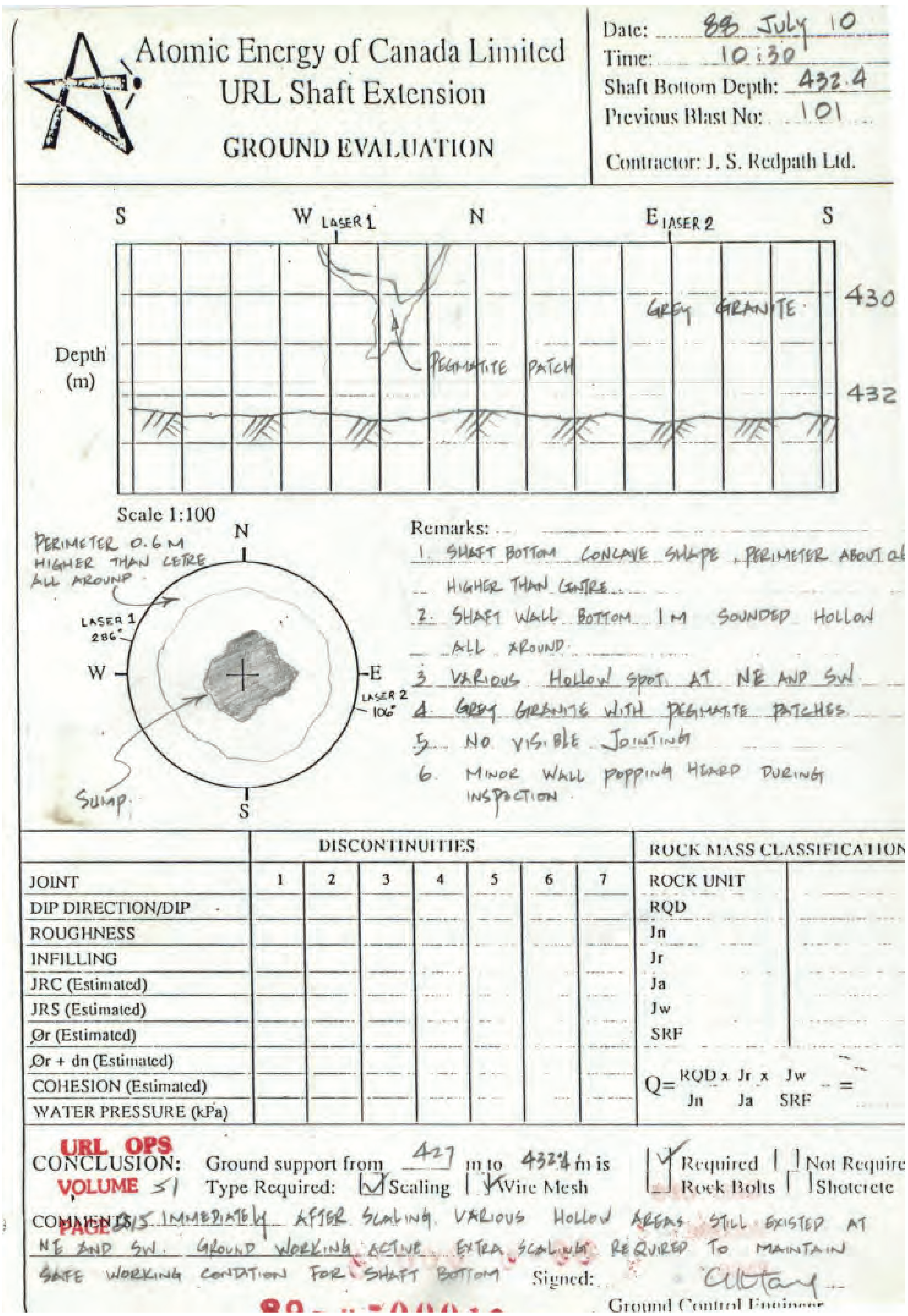
Depth of shaft bottom: 432 m

Signature: P.Tl

Form URL B10 REV.1

89-U-00010

Figure A. 9: Sample Page of Shaft Blast Round Summary – Blast Evaluation



Atomic Energy of Canada Limited
URL Shaft Extension
GROUND EVALUATION

Date: 88 JULY 10
Time: 10:30
Shaft Bottom Depth: 432.4
Previous Blast No: 101
Contractor: J. S. Redpath Ltd.

S W LASER 1 N E LASER 2 S

Depth (m)

430
432

GREY GRANITE
PEGMATITE PATCH

Scale 1:100

PERIMETER 0.6 M HIGHER THAN CETRE ALL AROUND

N
LASER 1 286°
W
E
LASER 2 106°
S
SUMP

Remarks:
1. SHAFT BOTTOM CONCAVE SHAPE, PERIMETER ABOUT 0.6 HIGHER THAN CENTRE
2. SHAFT WALL BOTTOM 1 M SOUNDED HOLLOW ALL AROUND
3. VARIOUS HOLLOW SPOT AT NE AND SW
4. GREY GRANITE WITH PEGMATITE PATCHES
5. NO VISIBLE JOINTING
6. MINOR WALL POPPING HEARD DURING INSPECTION

	DISCONTINUITIES						
JOINT	1	2	3	4	5	6	7
DIP DIRECTION/DIP							
ROUGHNESS							
INFILLING							
JRC (Estimated)							
JRS (Estimated)							
Ør (Estimated)							
Ør + dn (Estimated)							
COHESION (Estimated)							
WATER PRESSURE (kPa)							

ROCK MASS CLASSIFICATION	
ROCK UNIT	
RQD	
Jn	
Jr	
Ja	
Jw	
SRF	

$Q = \frac{RQD}{Jn} \times \frac{Jr}{Ja} \times \frac{Jw}{SRF} =$

URL OPS
CONCLUSION: Ground support from 427 m to 432.4 m is [✓] Required [ ] Not Required
VOLUME ≤1 Type Required: [✓] Scaling [✓] Wire Mesh [ ] Rock Bolts [ ] Shotcrete

PAGE
COMMENTS: IMMEDIATELY AFTER SCALING, VARIOUS HOLLOW AREAS STILL EXISTED AT NE AND SW. GROUND WORKING ACTIVE. EXTRA SCALING REQUIRED TO MAINTAIN SAFE WORKING CONDITION FOR SHAFT BOTTOM

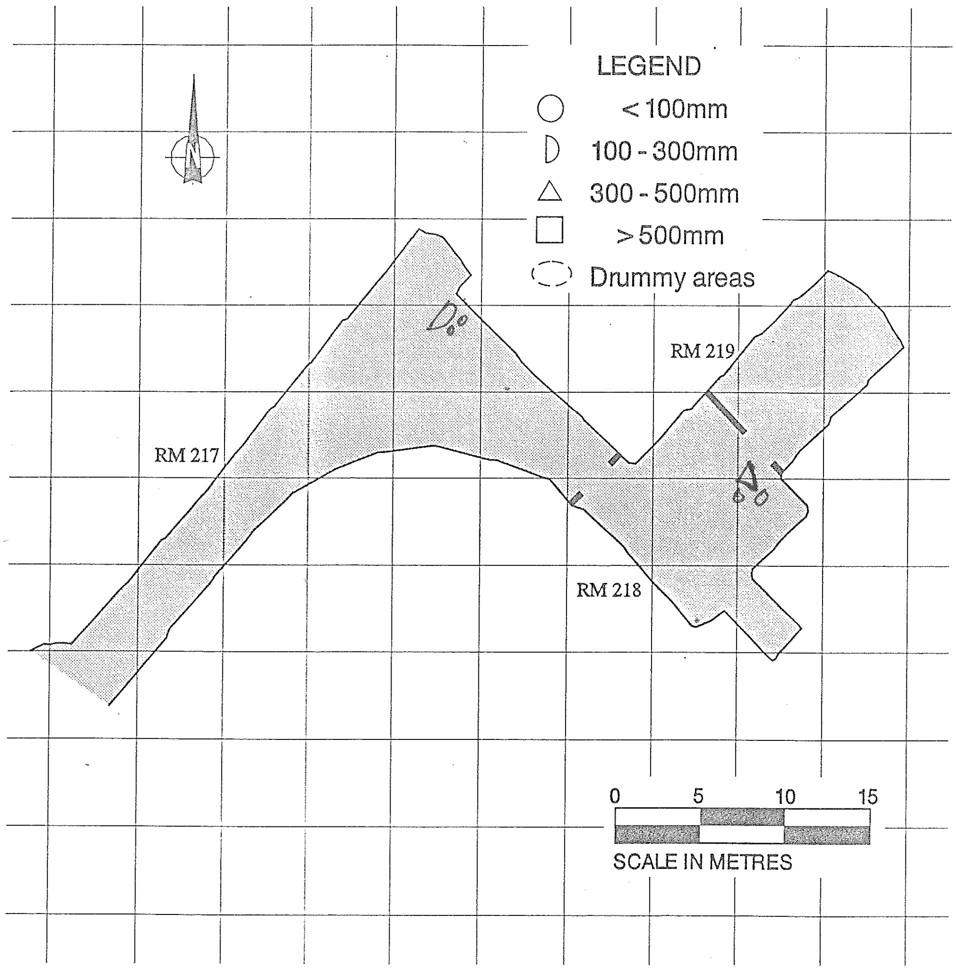
Signed: [signature]
Ground Control Engineer

Figure A. 10: Sample Page of Shaft Blast Round Summary – Ground Evaluation

A.3 EXCAVATION MAINTENANCE FORMS

AECL
240 LEVEL
SCALING RECORD

Date Jun 16/06
Shift
Contractor: J.S. Redpath
Signed



Comments: ______________________________

E3939510

APPENDIX B: SCALING RECORDS

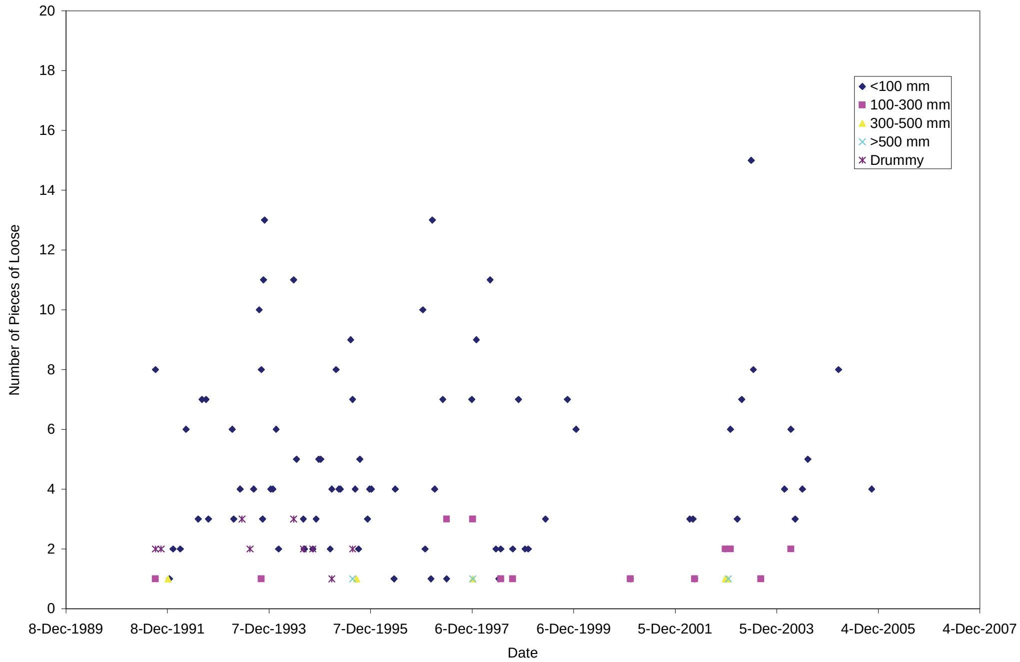


Figure B. 1: Amount of Loose Rock Scaled from the 130 Level of the URL.

The 130 Level is the shallowest of the excavations at the URL and hence is most the exposed of the underground rooms to changes in temperature and humidity conditions on an annual basis.

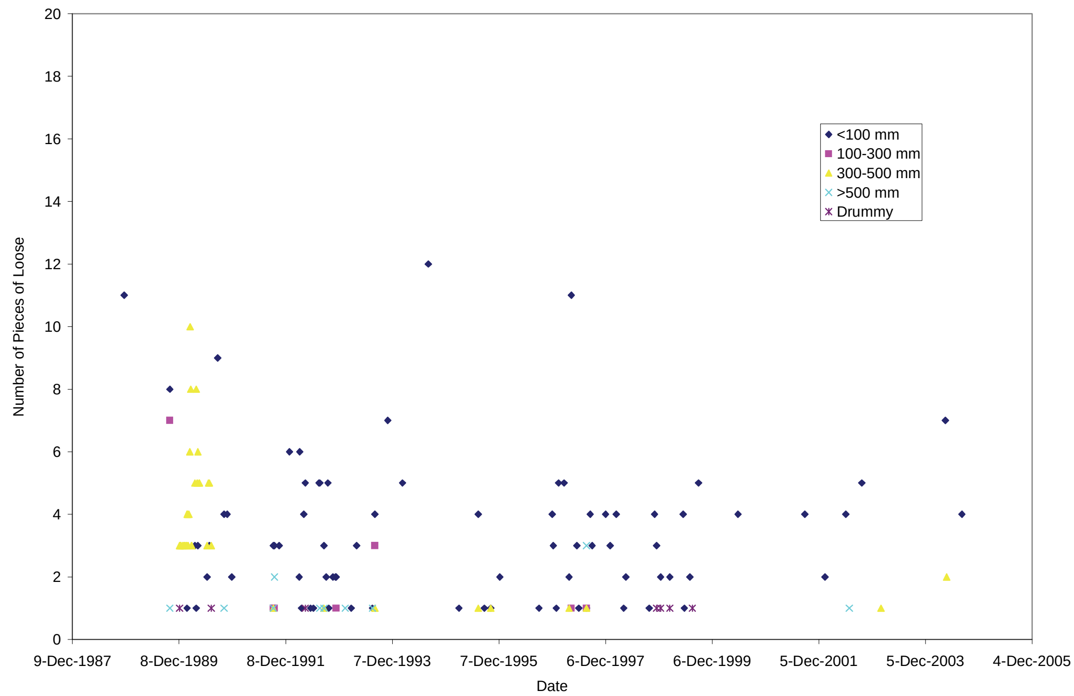


Figure B. 2: Amount of Loose Rock Scaled from Room 201 on the 240 Level of the URL

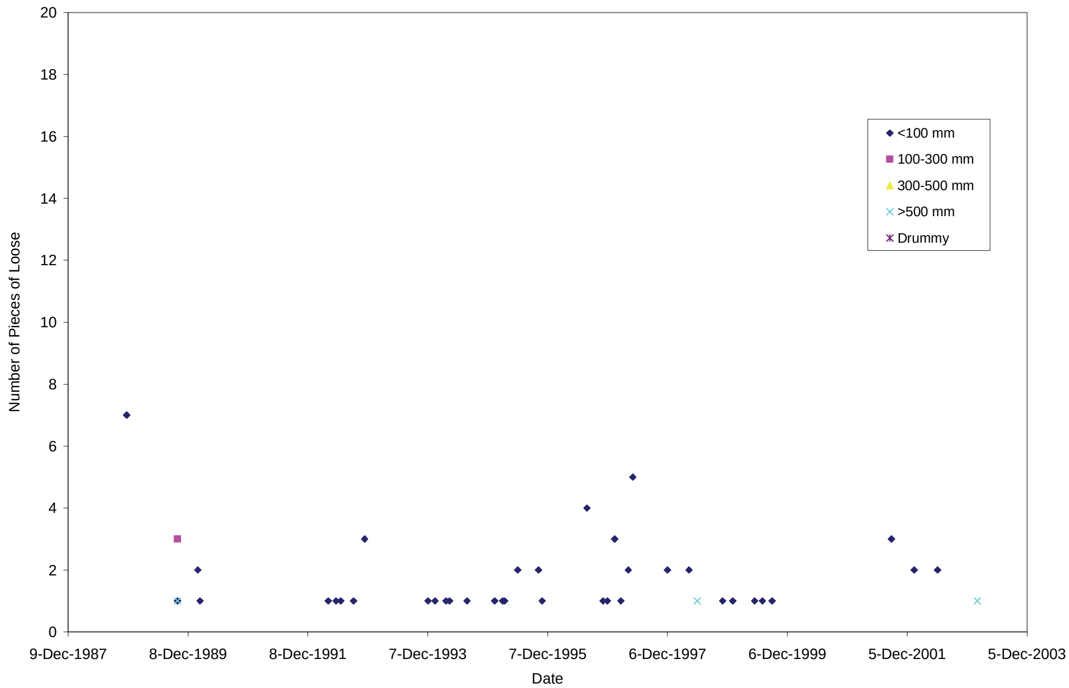


Figure B. 3: Amount of Loose Rock Scaled from Room 203 on the 240 Level of the URL

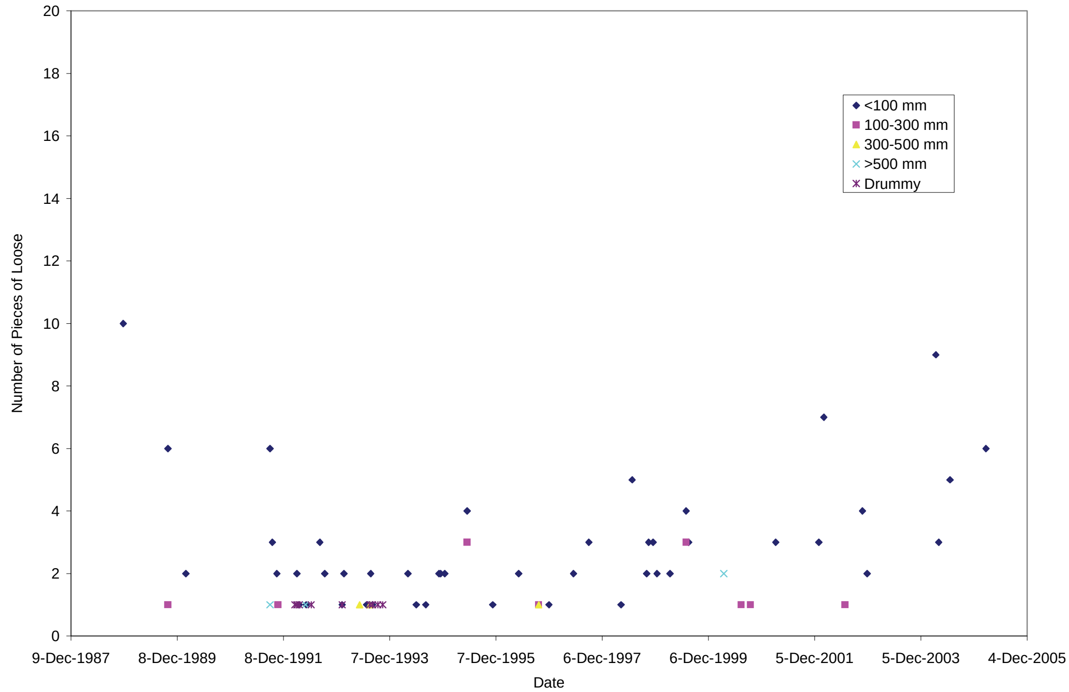


Figure B. 4: Amount of Loose Rock Scaled from Room 204 on the 240 Level of the URL

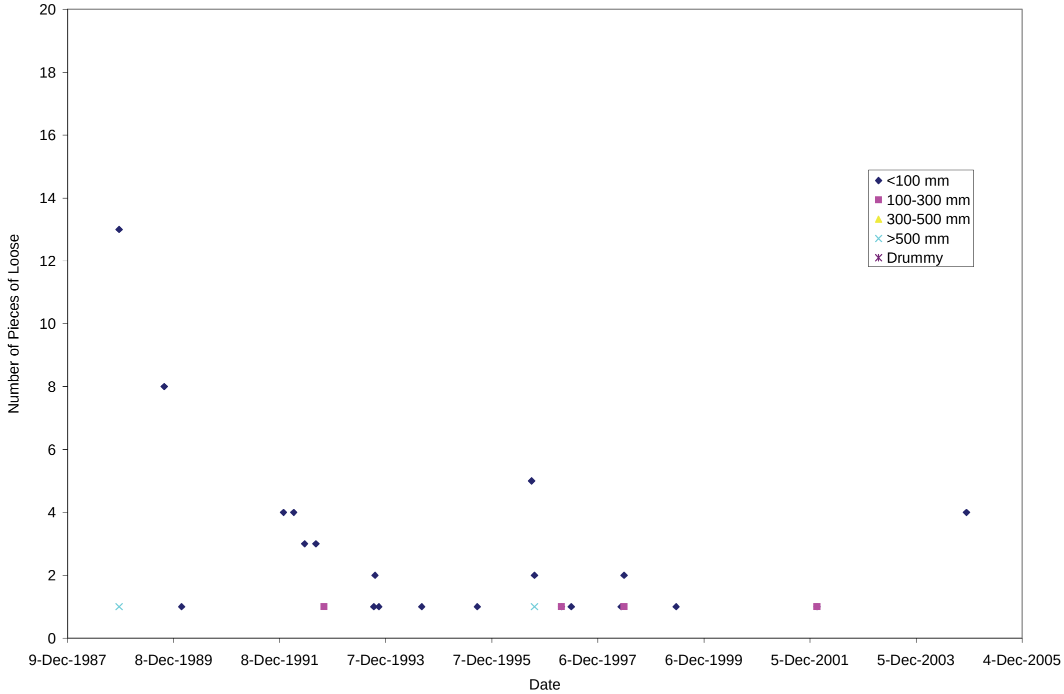


Figure B. 5: Amount of Loose Rock Scaled from Room 205 on the 240 Level of the URL

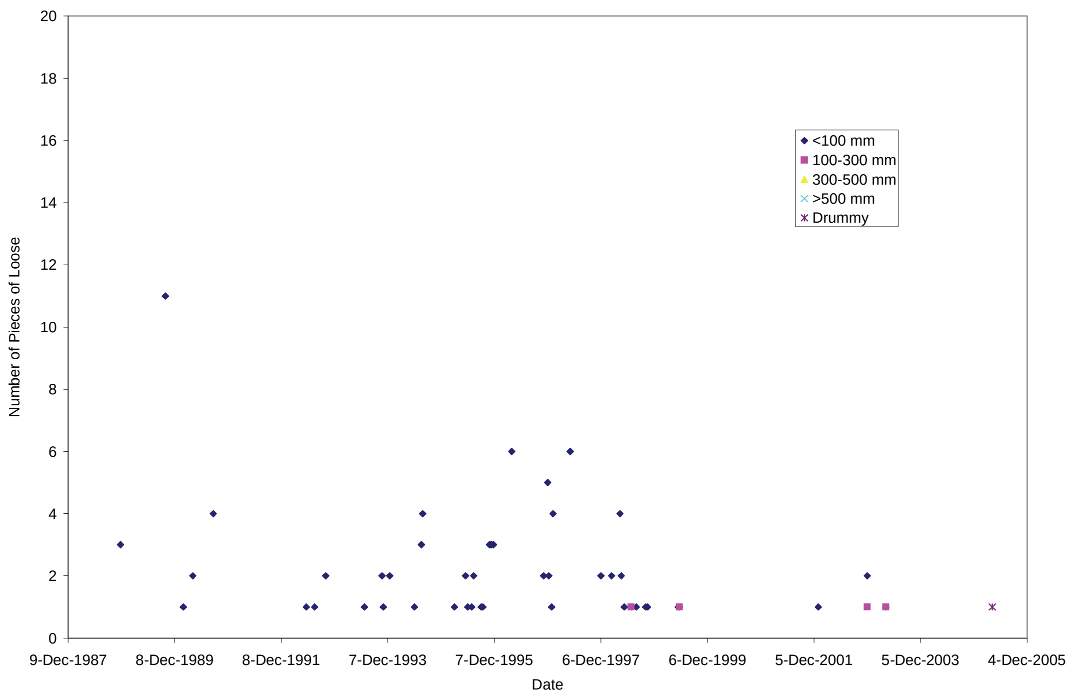


Figure B. 6: Amount of Loose Rock Scaled from Room 206 on the 240 Level of the URL

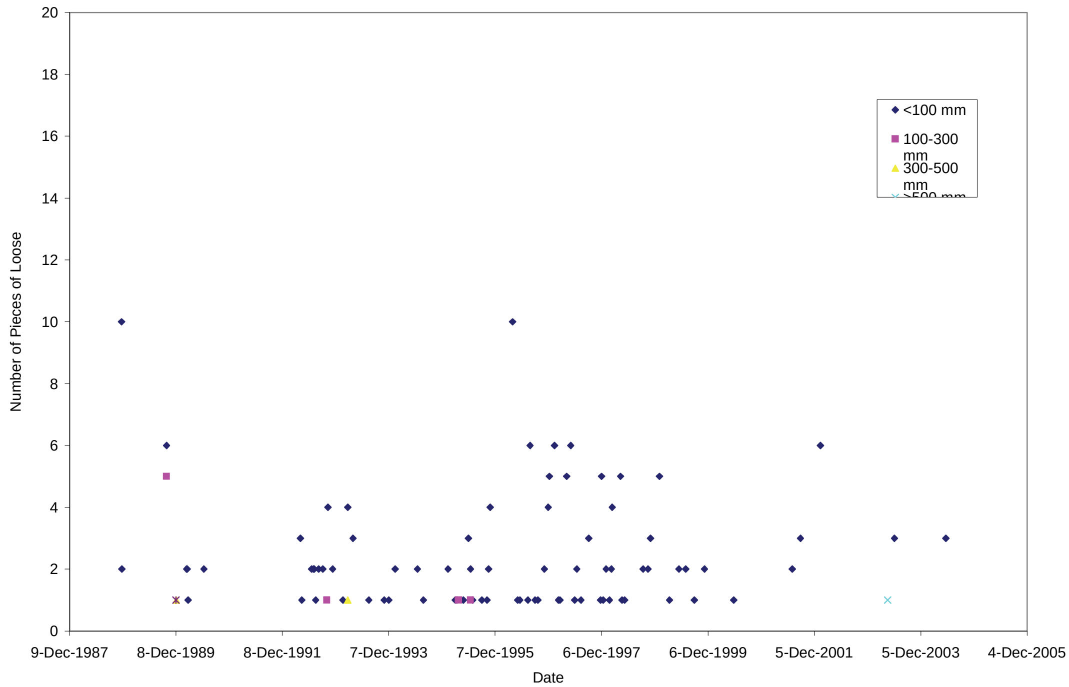


Figure B. 7: Amount of Loose Rock Scaled from Room 207 on the 240 Level of the URL

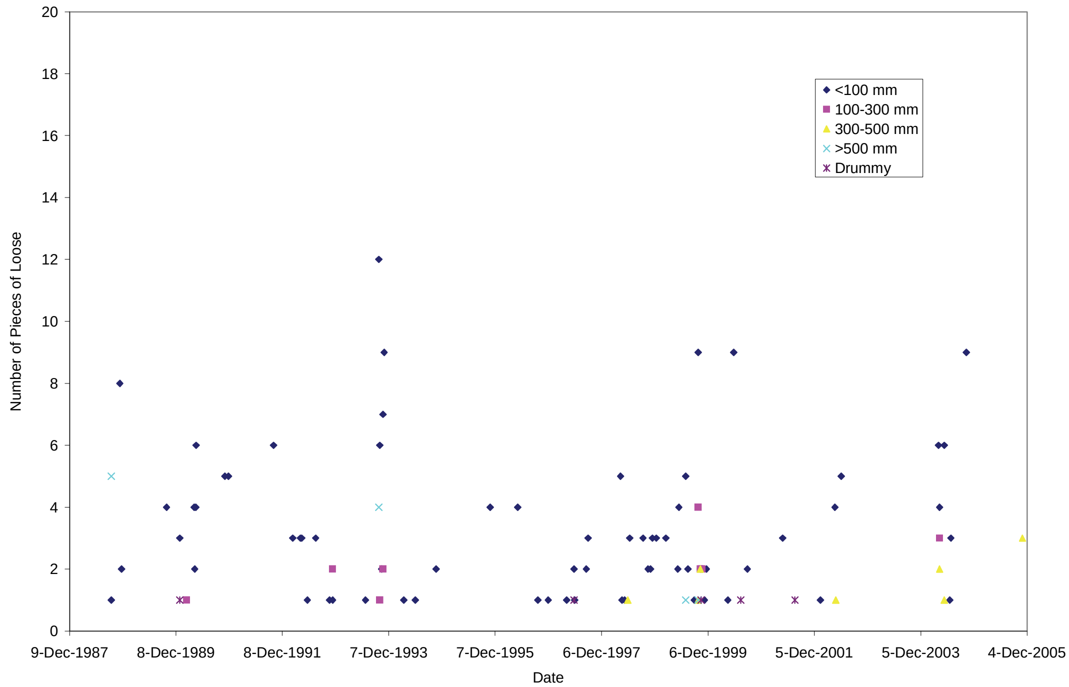


Figure B. 8: Amount of Loose Rock Scaled from Room 208 on the 240 Level of the URL

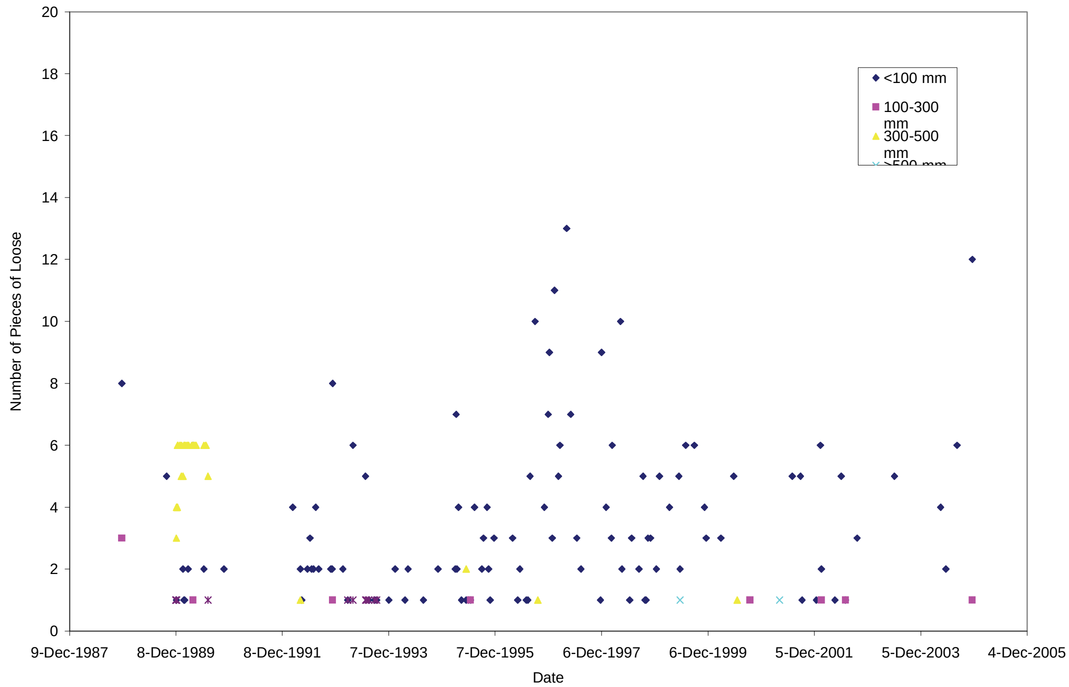


Figure B. 9: Amount of Loose Rock Scaled from Room 209 on the 240 Level of the URL

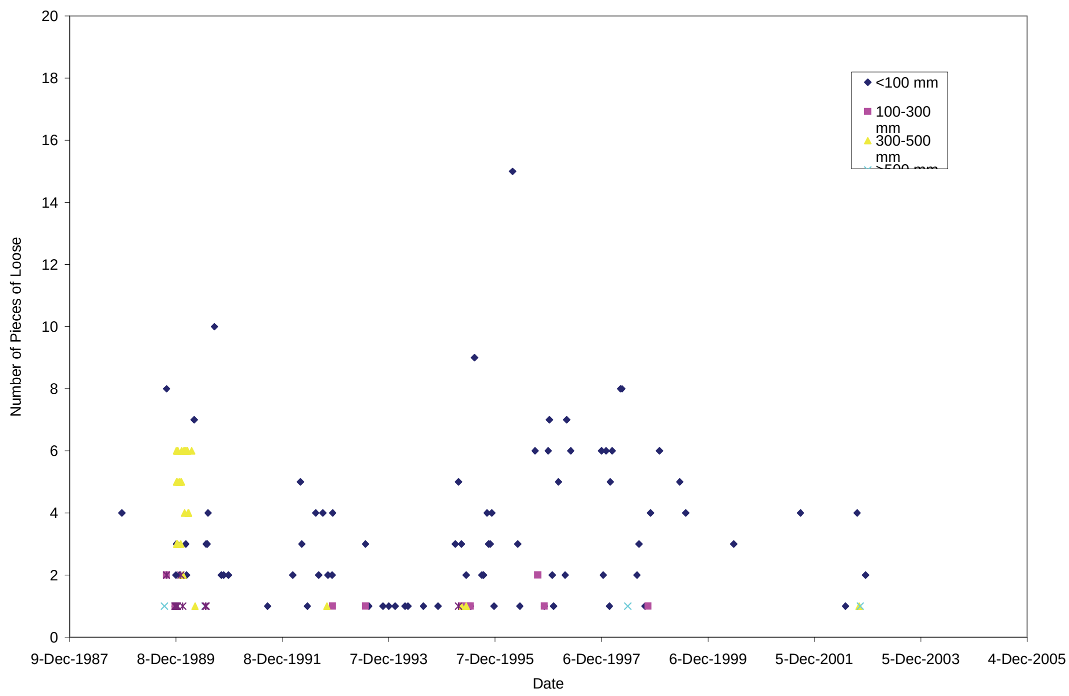


Figure B. 10: Amount of Loose Rock Scaled from Room 210 on the 240 Level of the URL

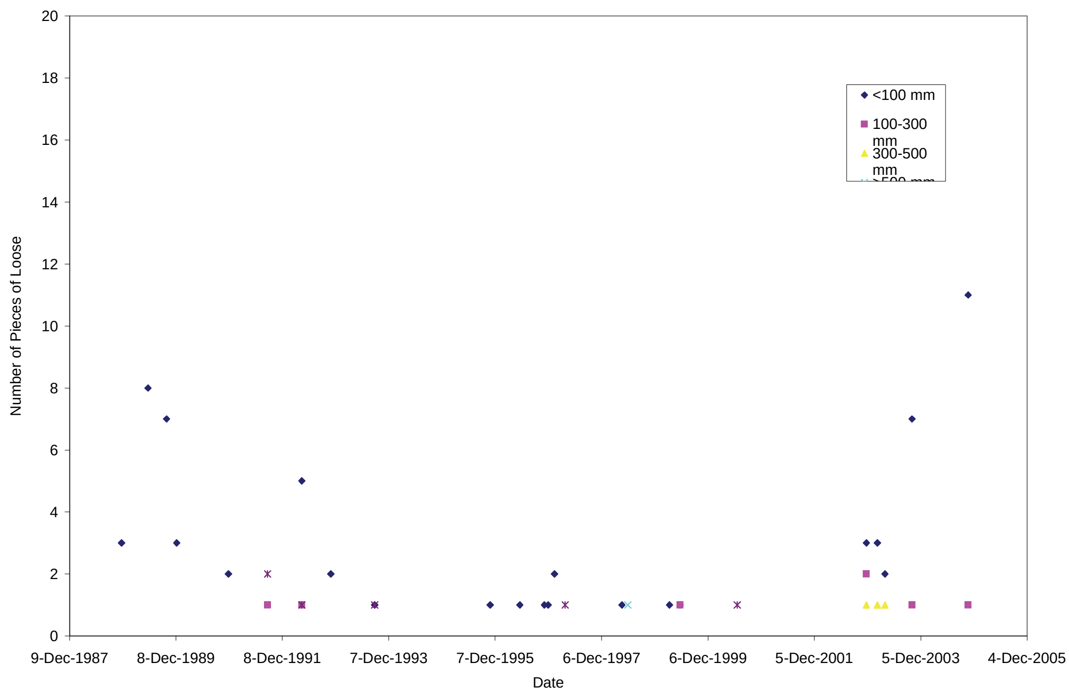


Figure B. 11: Amount of Loose Rock Scaled from Room 211 on the 240 Level of the URL

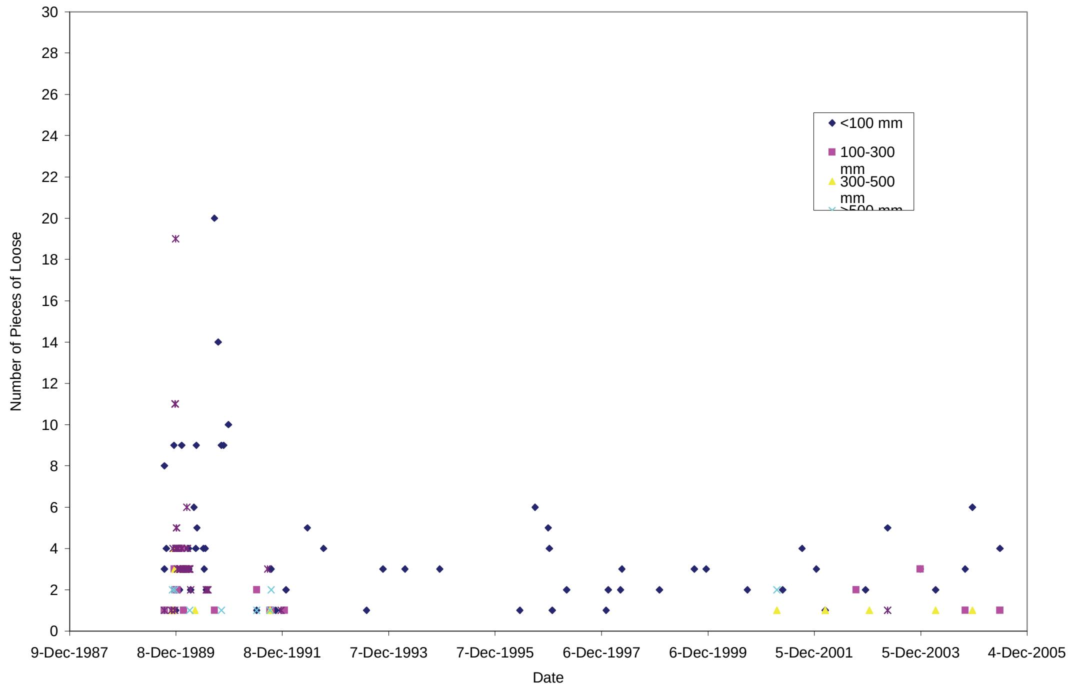


Figure B. 12: Amount of Loose Rock Scaled from Room 213 on the 240 Level of the URL

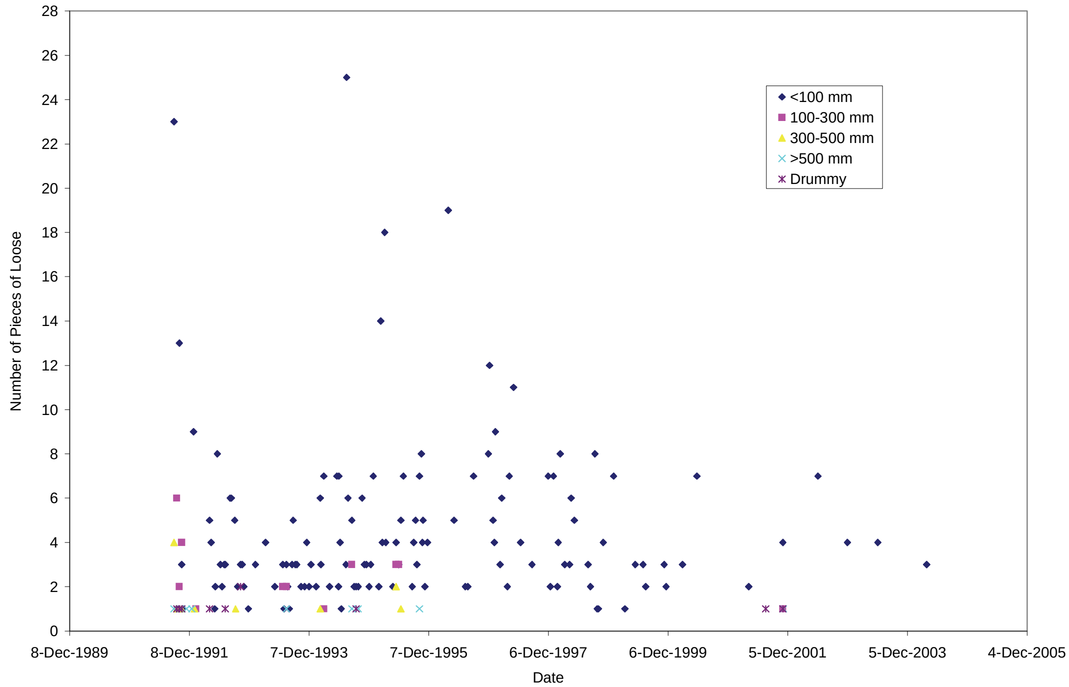


Figure B. 13: Amount of Loose Rock Scaled from Room 214 on the 240 Level of the URL

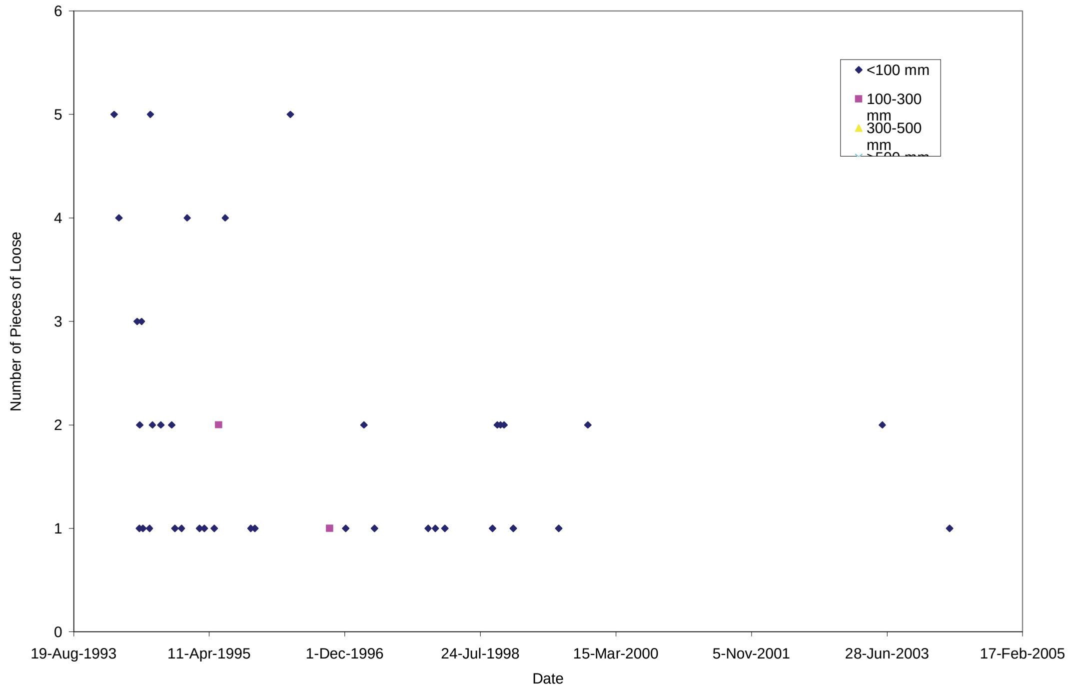


Figure B. 14: Amount of Loose Rock Scaled from Room 215 on the 240 Level of the URL

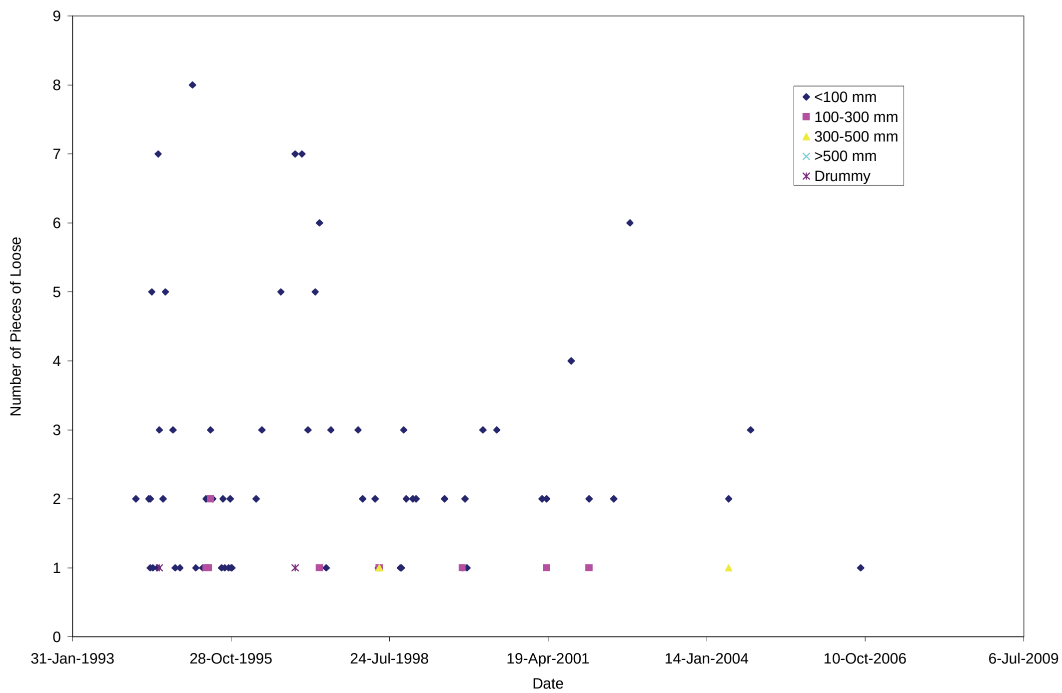


Figure B. 15: Amount of Loose Rock Scaled from Room 216 on the 240 Level of the URL

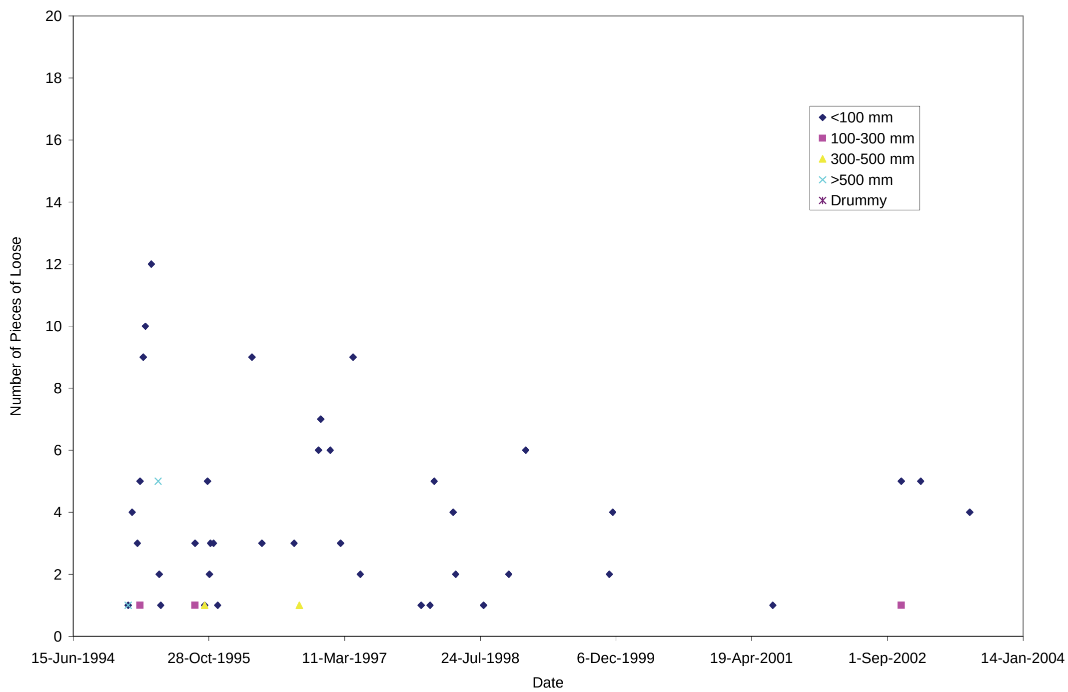


Figure B. 16: Amount of Loose Rock Scaled from Room 217 on the 240 Level of the URL

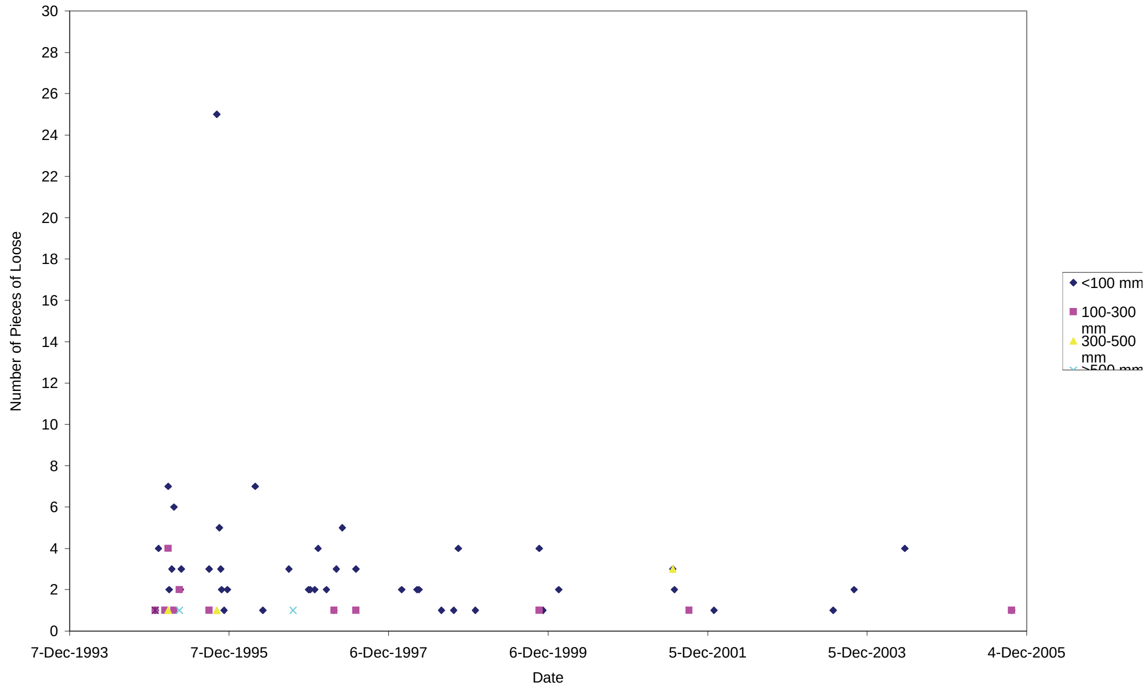


Figure B. 17: Amount of Loose Rock Scaled from Room 218 on the 240 Level of the URL

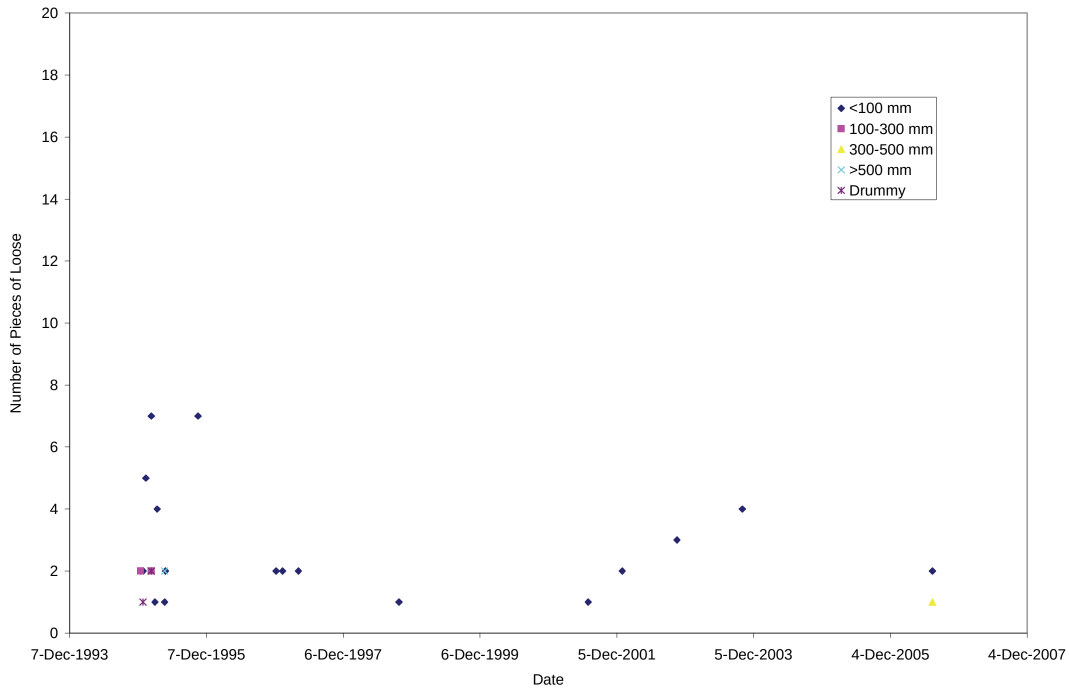


Figure B. 18: Amount of Loose Rock Scaled from Room 219 on the 240 Level of the URL

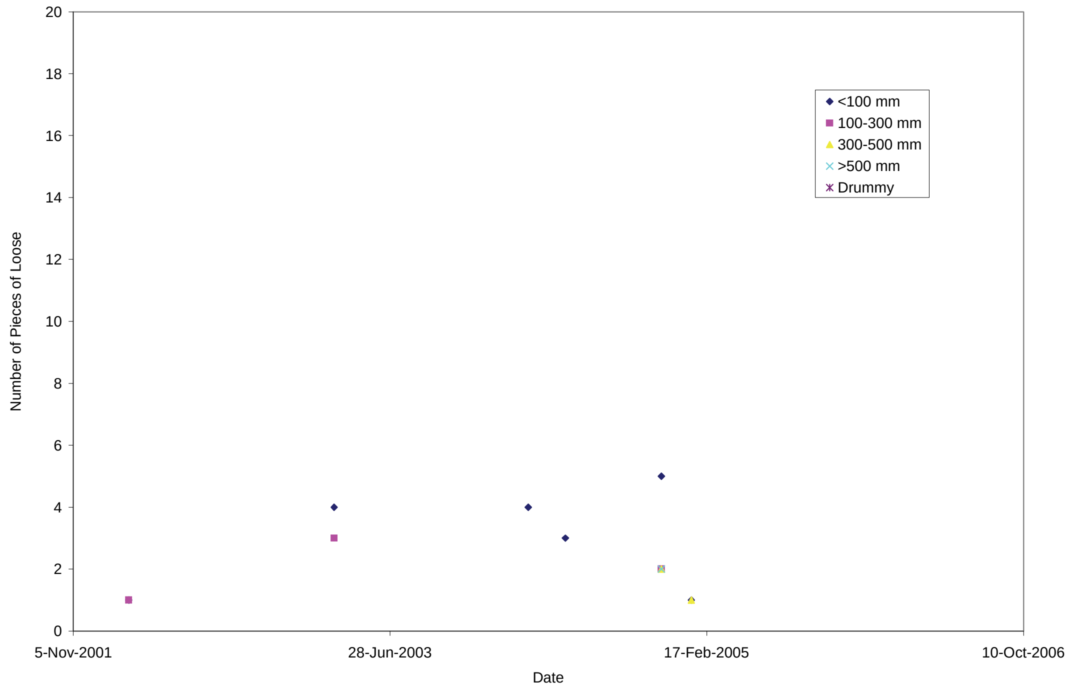


Figure B. 19: Amount of Loose Rock Scaled from Room 220 on the 240 Level of the URL

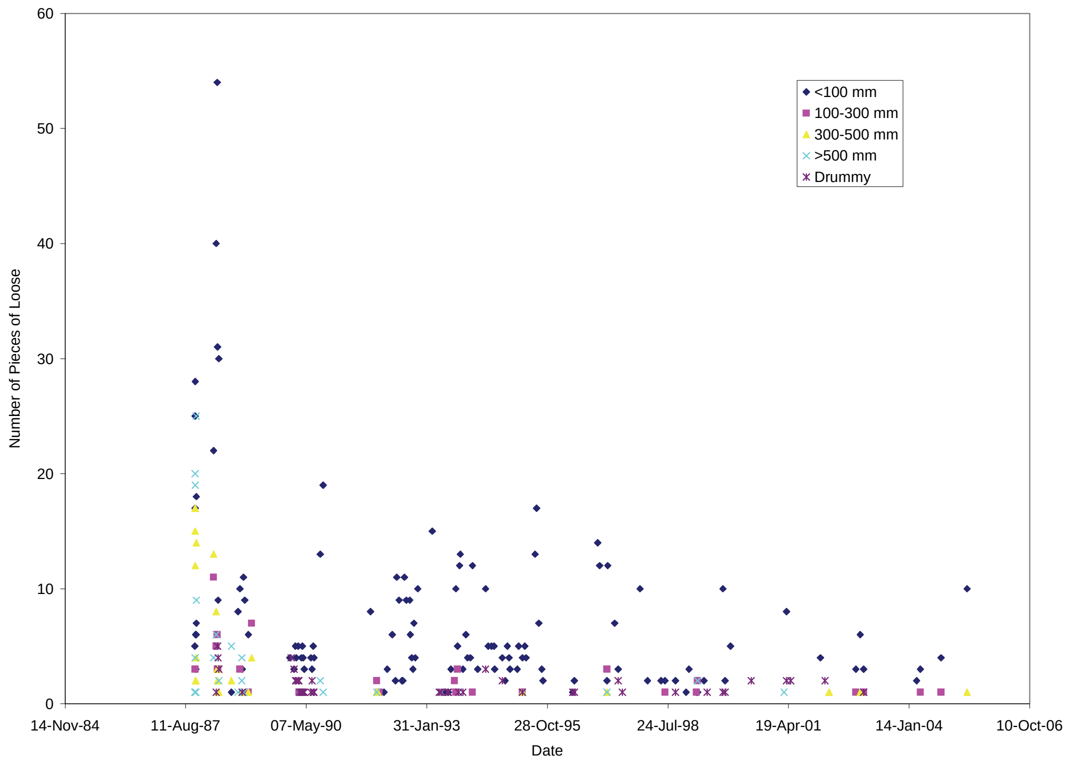


Figure B. 20: Amount of Loose Rock Scaled from the 300 Level of the URL

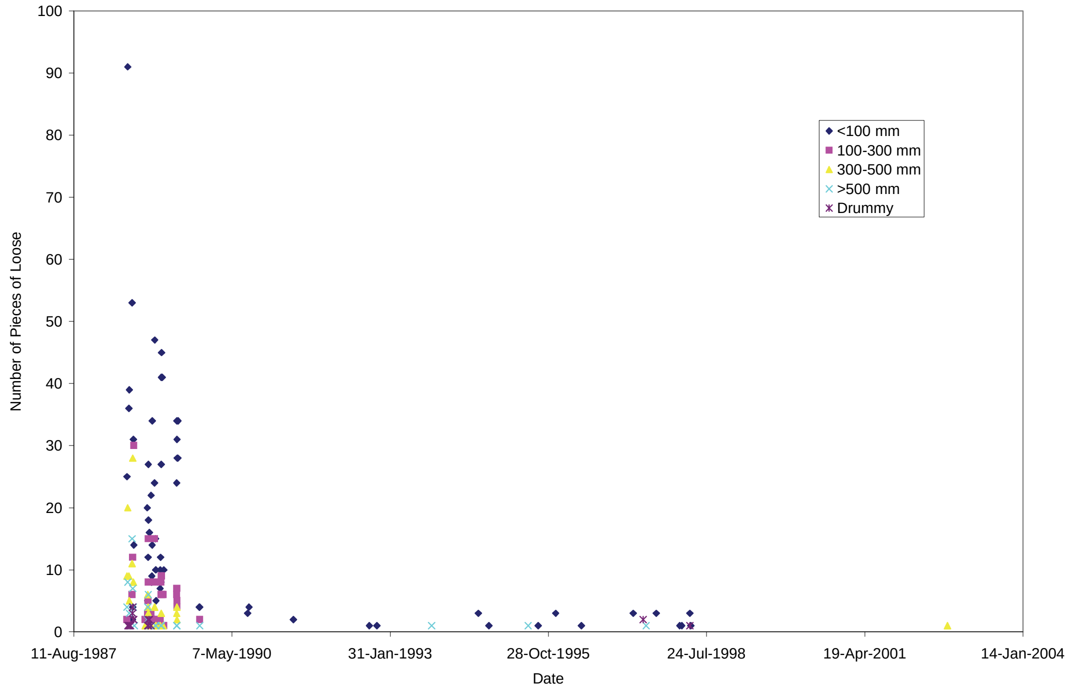


Figure B. 21: Amount of Loose Rock Scaled from Room 401 on the 420 Level of the URL

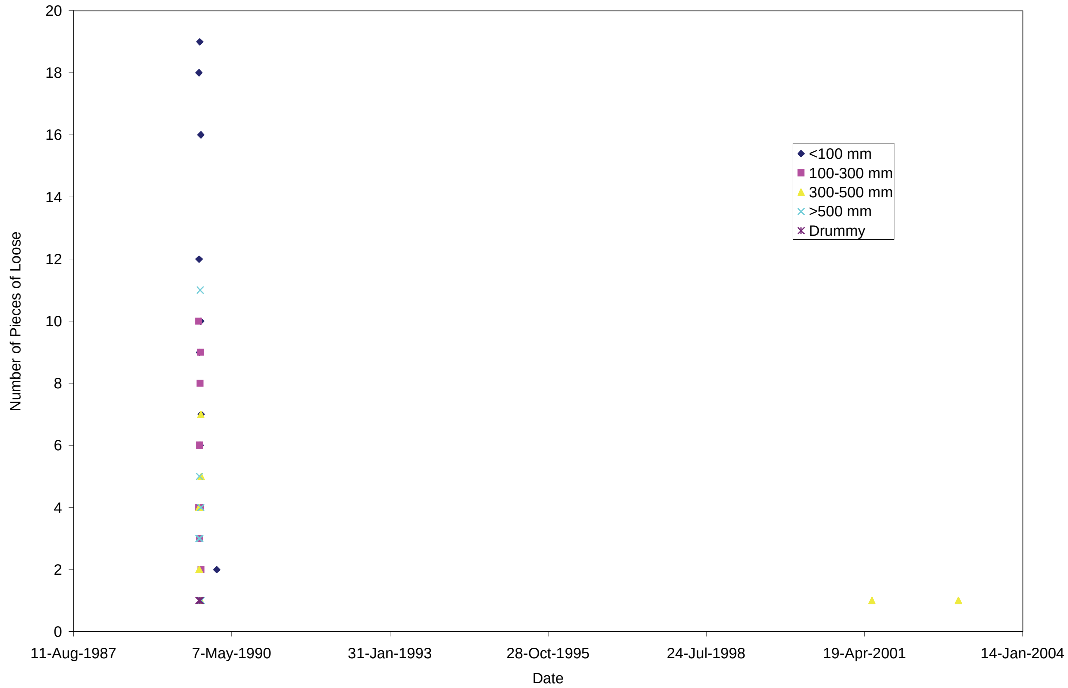


Figure B. 22: Amount of Loose Rock Scaled from Room 402 on the 420 Level of the URL

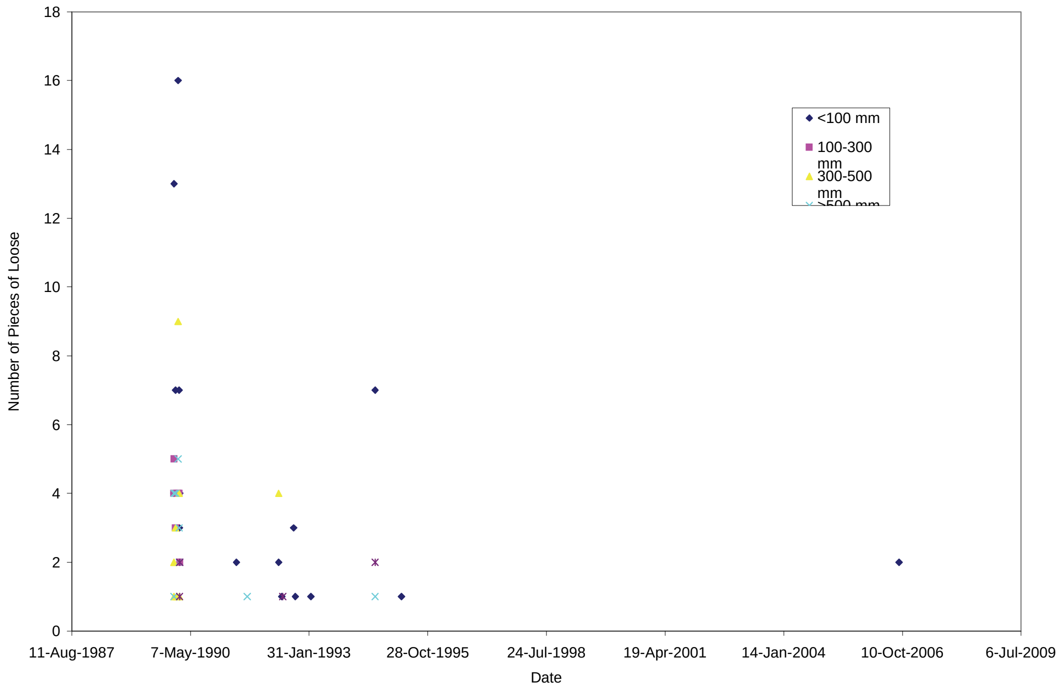


Figure B. 23: Amount of Loose Rock Scaled from Room 403 on the 420 Level of the URL

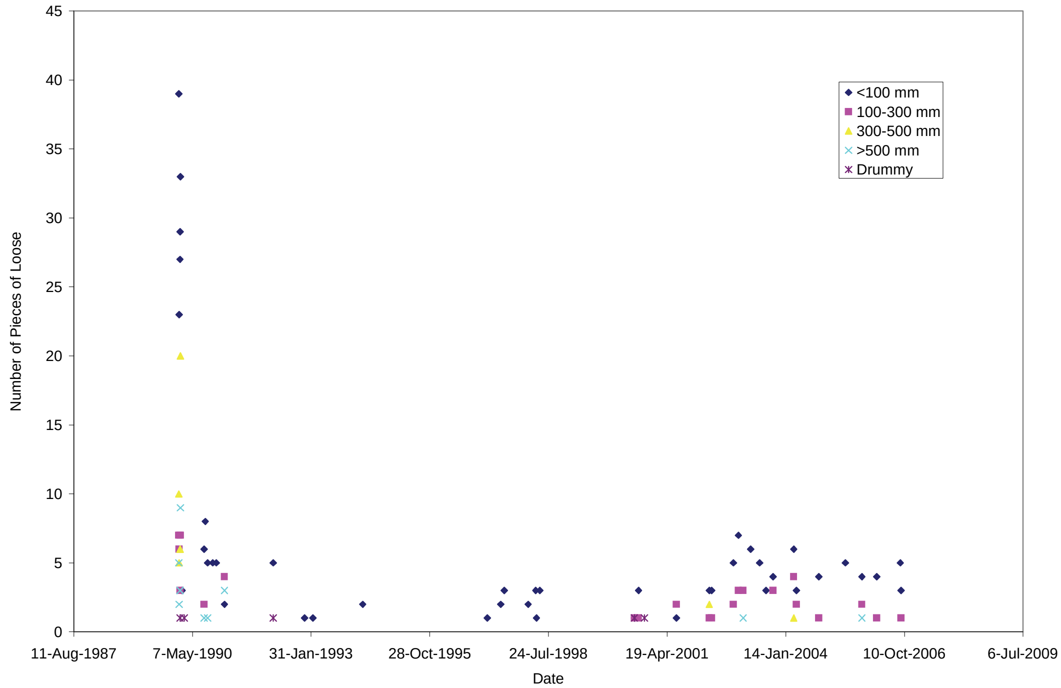


Figure B. 24: Amount of Loose Rock Scaled from Room 404 on the 420 Level of the URL

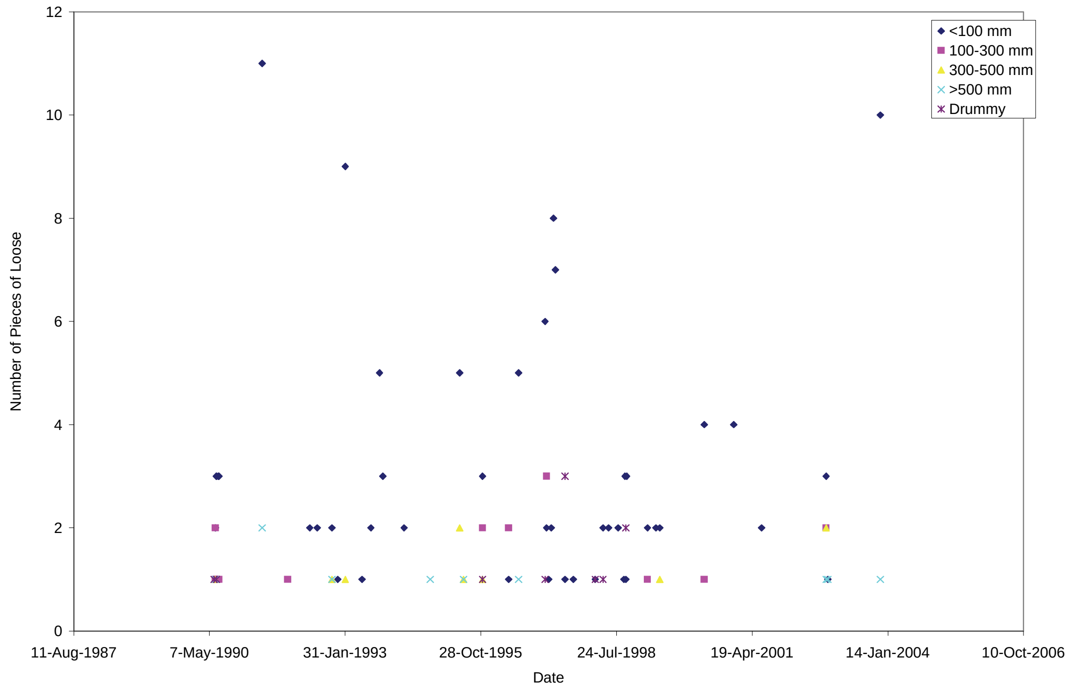


Figure B. 25: Amount of Loose Rock Scaled from Room 405 on the 420 Level of the URL

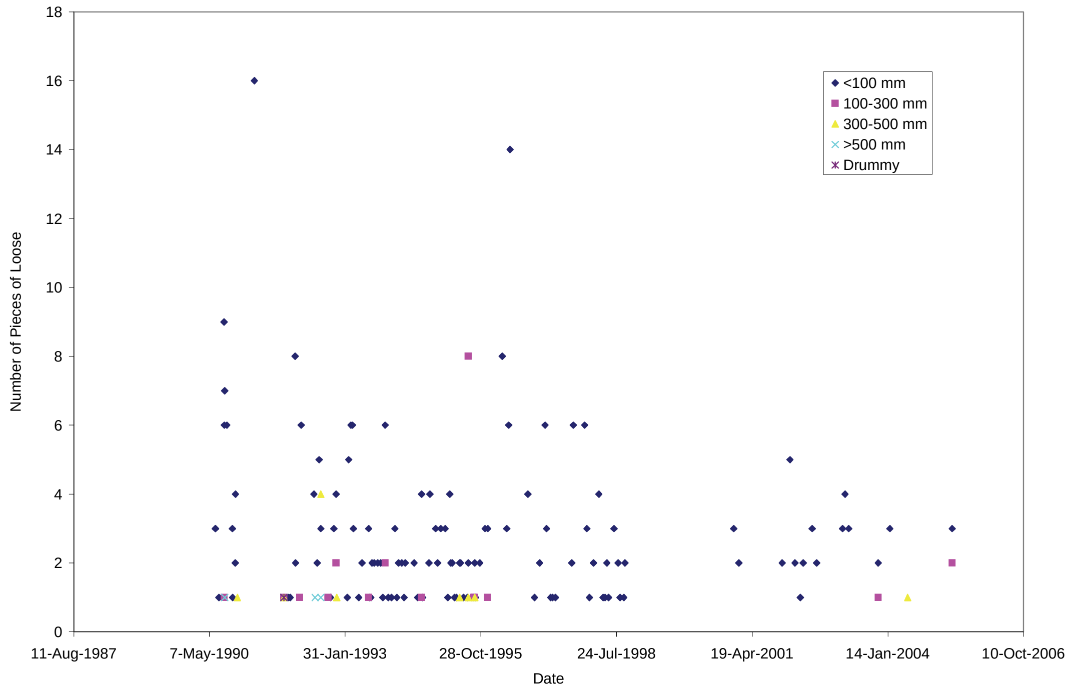


Figure B. 26: Amount of Loose Rock Scaled from Room 406 on the 420 Level of the URL

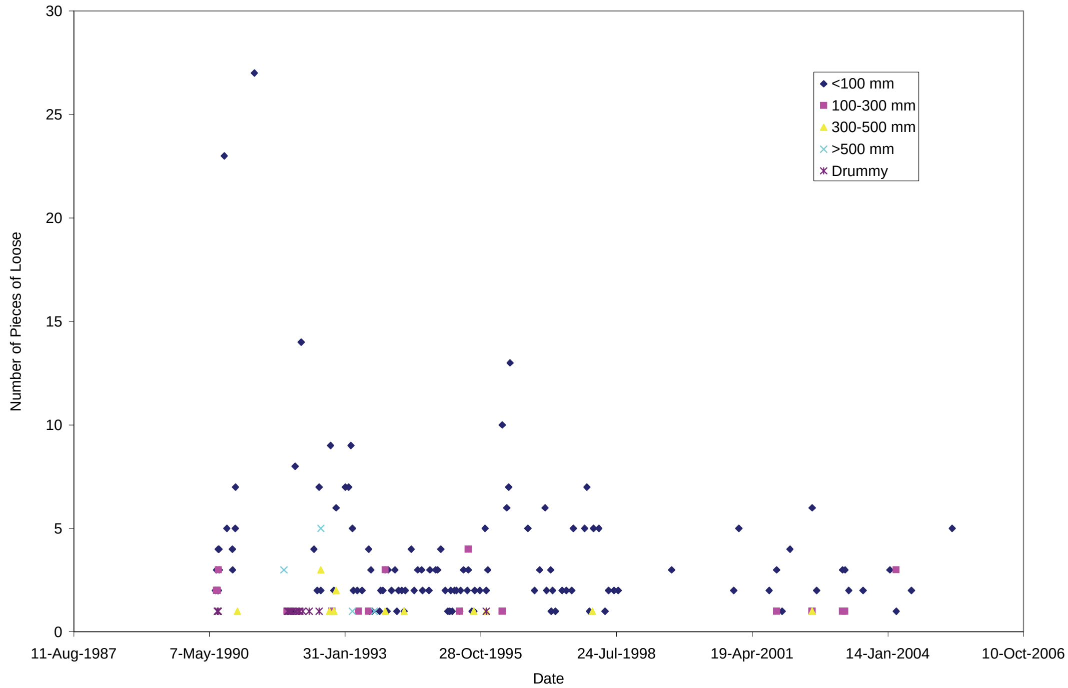


Figure B. 27: Amount of Loose Rock Scaled from Room 407 on the 420 Level of the URL

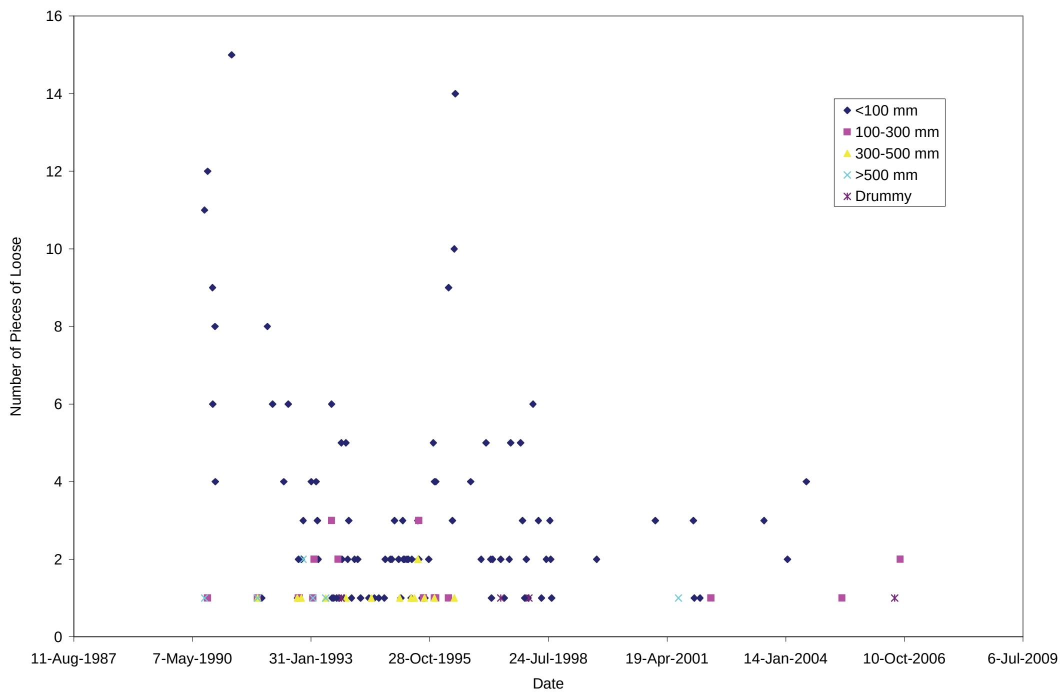


Figure B. 28: Amount of Loose Rock Scaled from Room 408 on the 420 Level of the URL

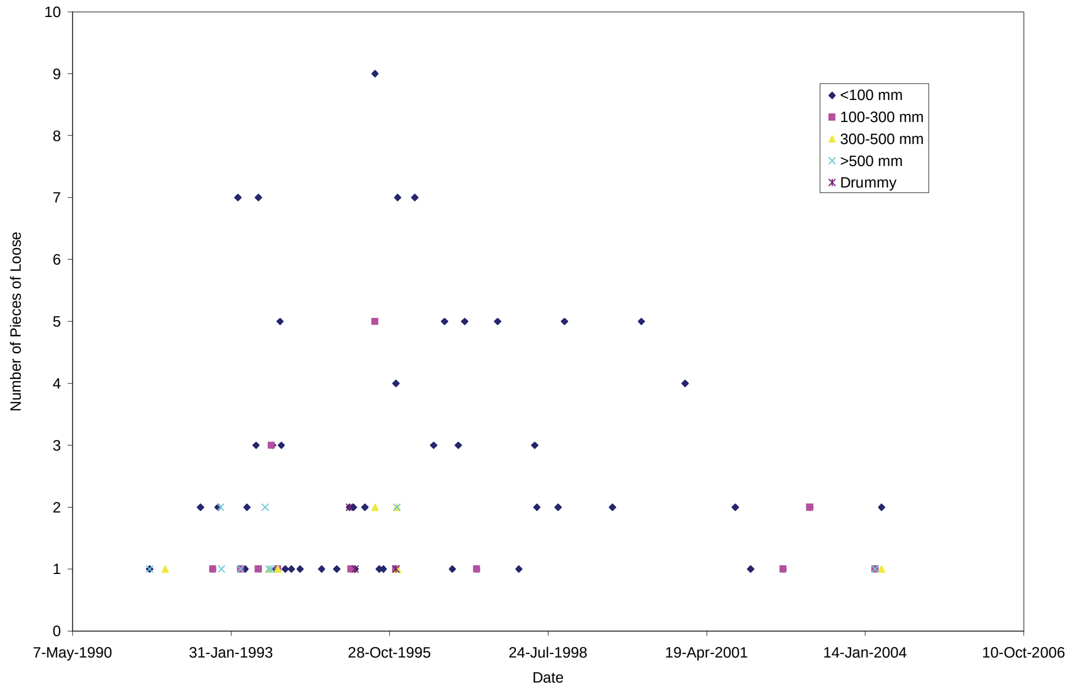


Figure B. 29: Amount of Loose Rock Scaled from Room 409 on the 420 Level of the URL

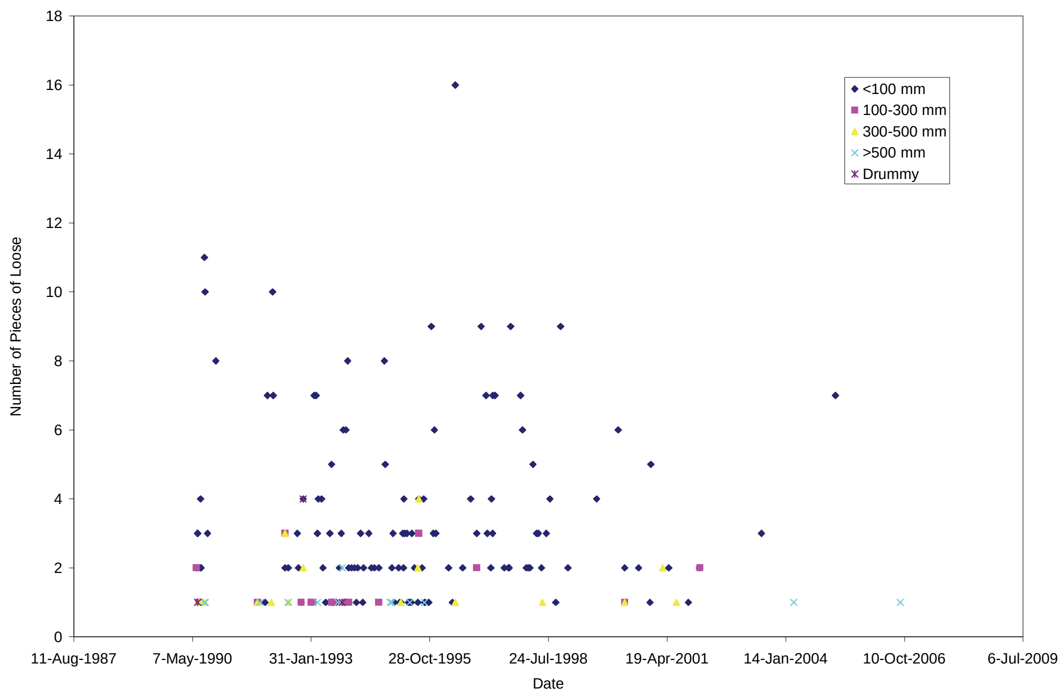


Figure B. 30: Amount of Loose Rock Scaled from Room 410 on the 420 Level of the URL

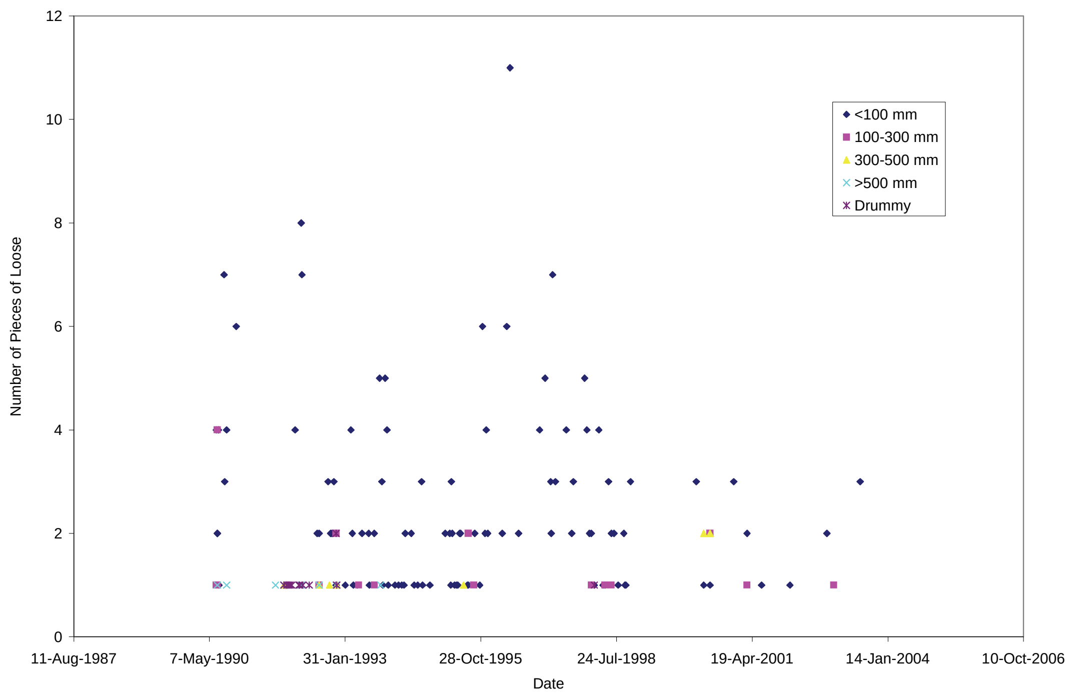


Figure B. 31: Amount of Loose Rock Scaled from Room 411 on the 420 Level of the URL

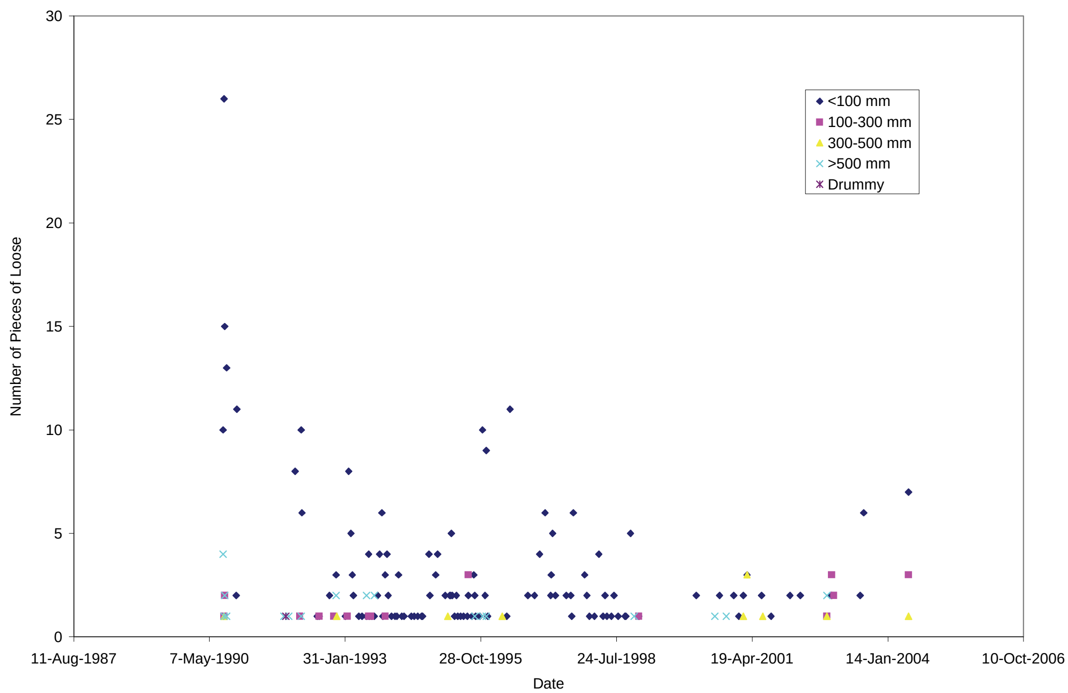


Figure B. 32: Amount of Loose Rock Scaled from Room 412 on the 420 Level of the URL

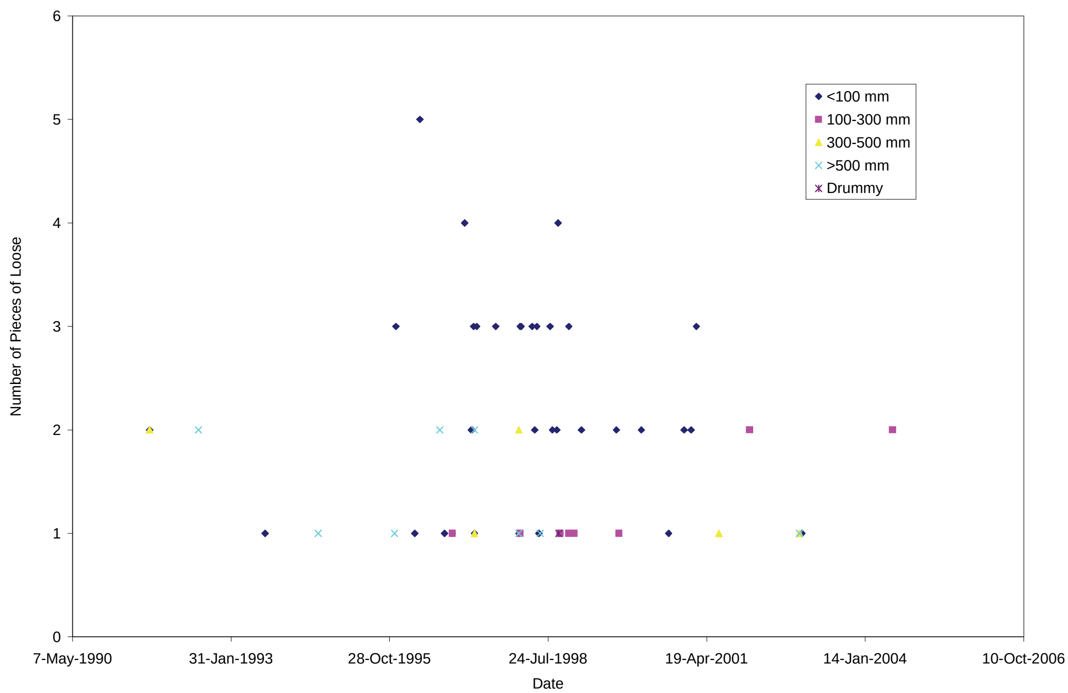


Figure B. 33: Amount of Loose Rock Scaled from Room 413 on the 420 Level of the URL

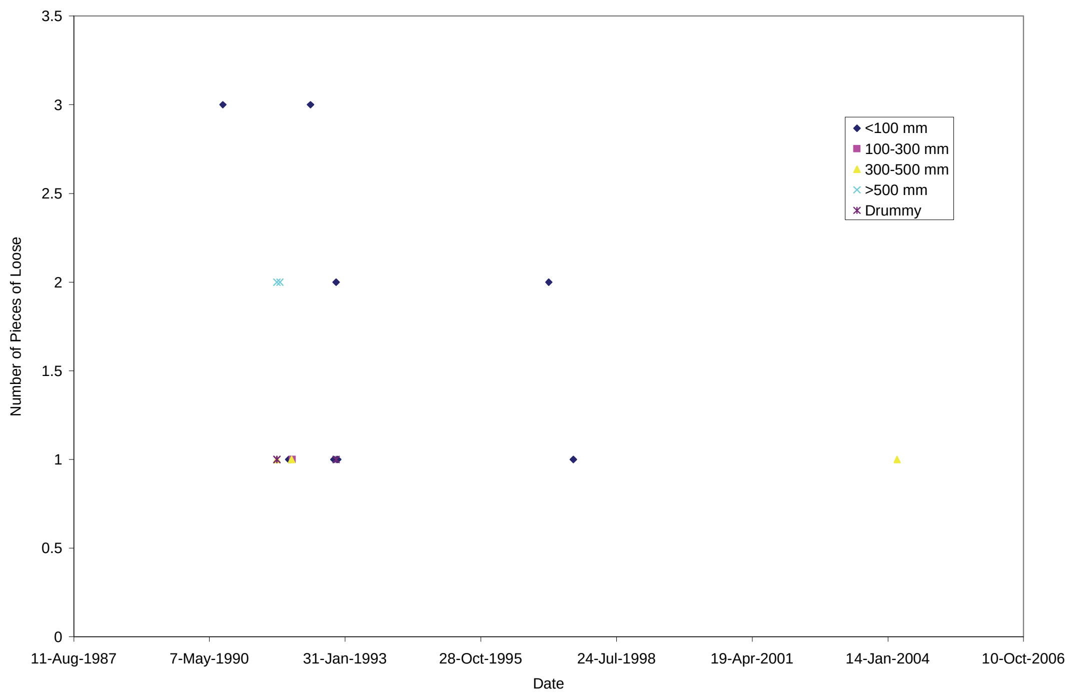


Figure B. 34: Amount of Loose Rock Scaled from Room 414 on the 420 Level of the URL

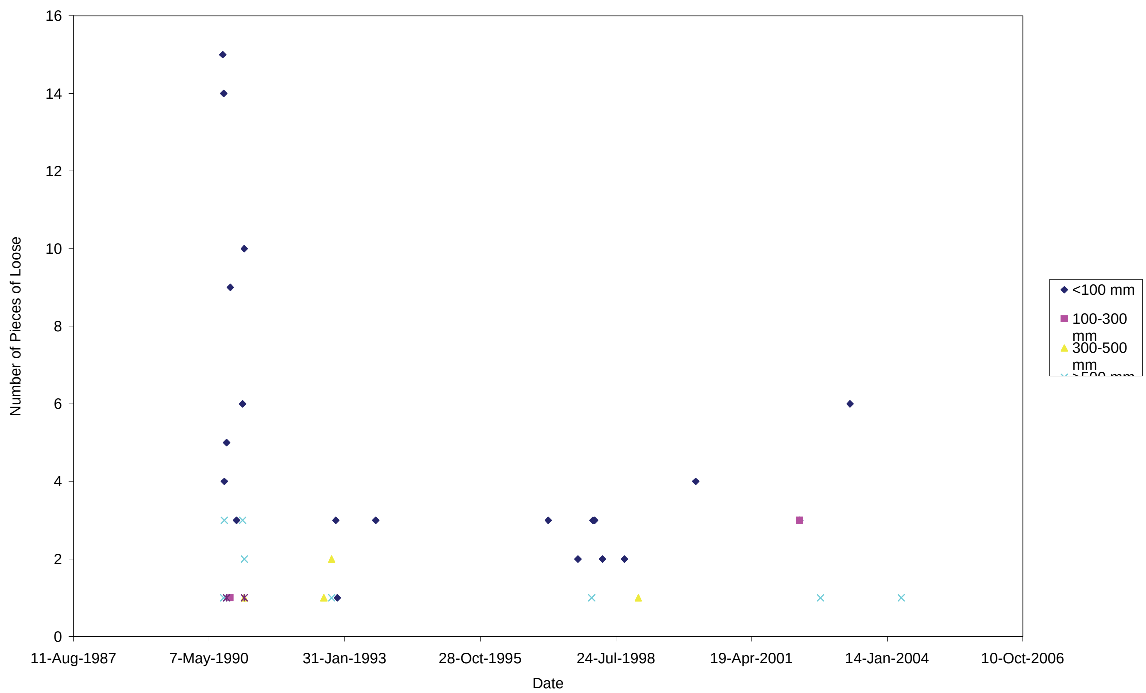


Figure B. 35: Amount of Loose Rock Scaled from Room 415 on the 420 Level of the URL

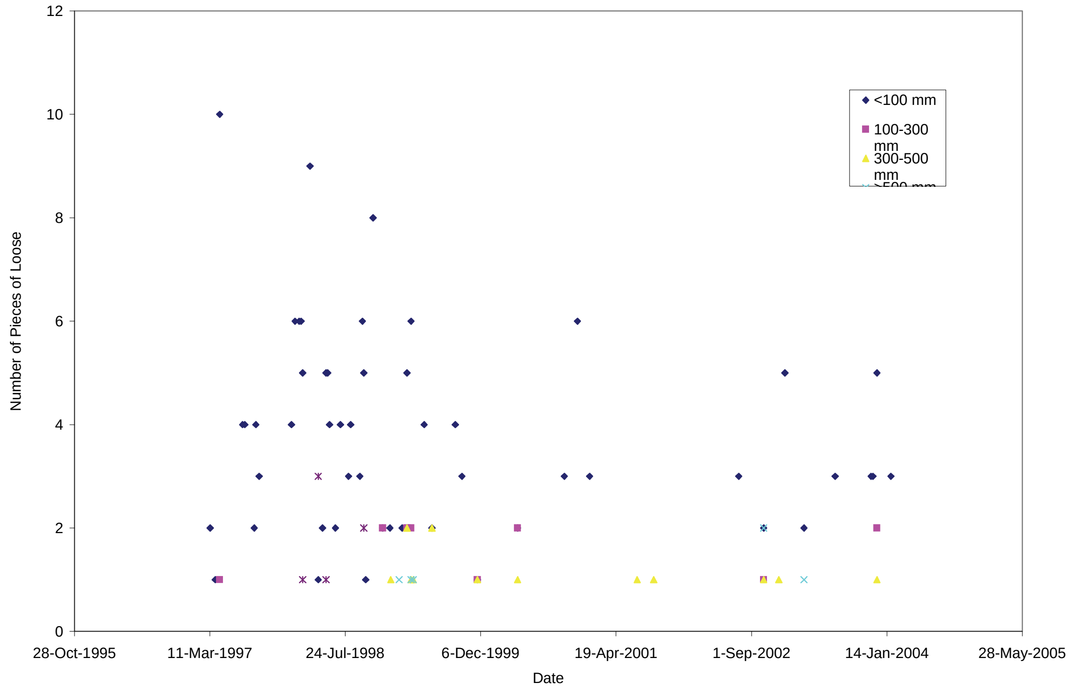


Figure B. 36: Amount of Loose Rock Scaled from Room 417 on the 420 Level of the URL

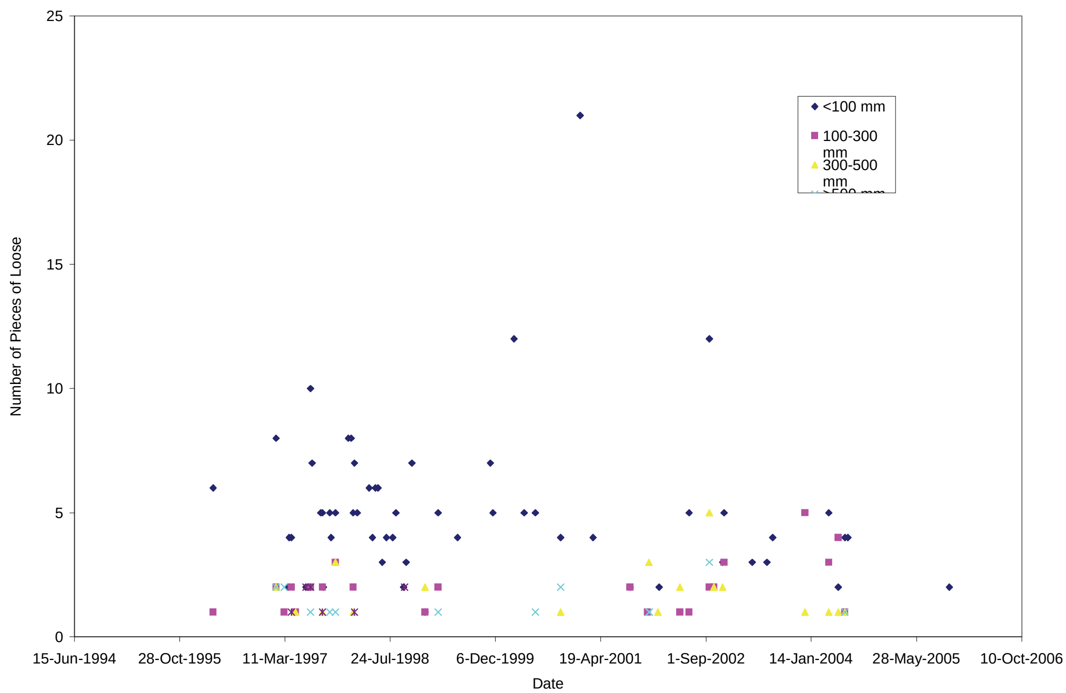


Figure B. 37: Amount of Loose Rock Scaled from Room 418 on the 420 Level of the URL

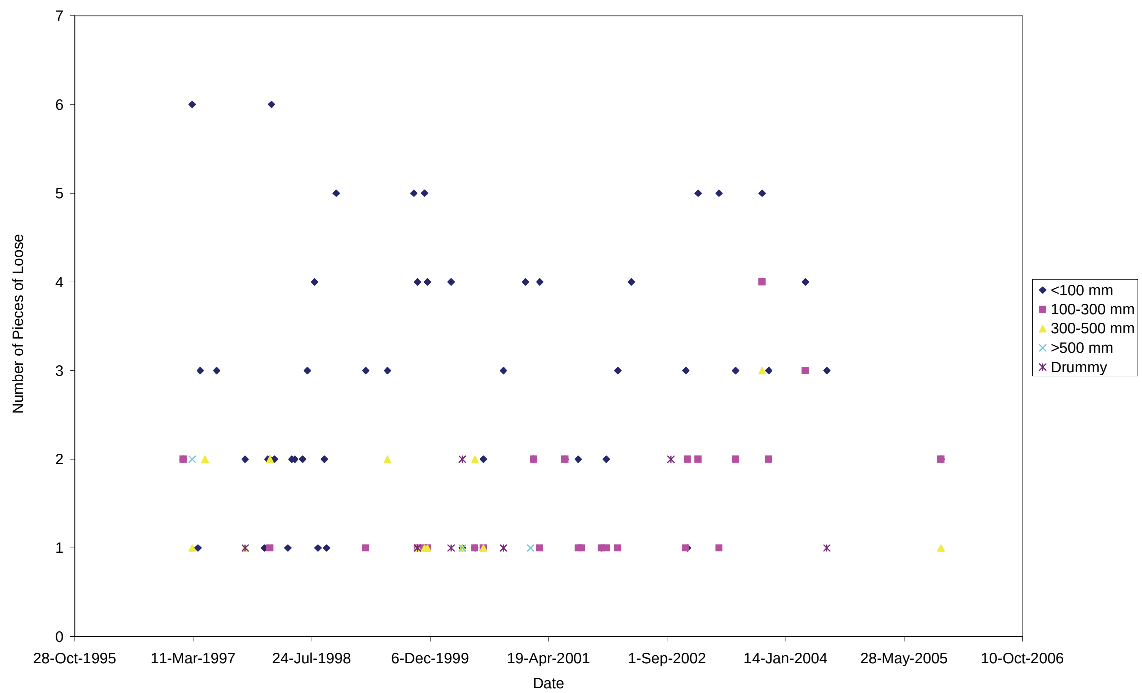


Figure B. 38: Amount of Loose Rock Scaled from Room 419 on the 420 Level of the URL

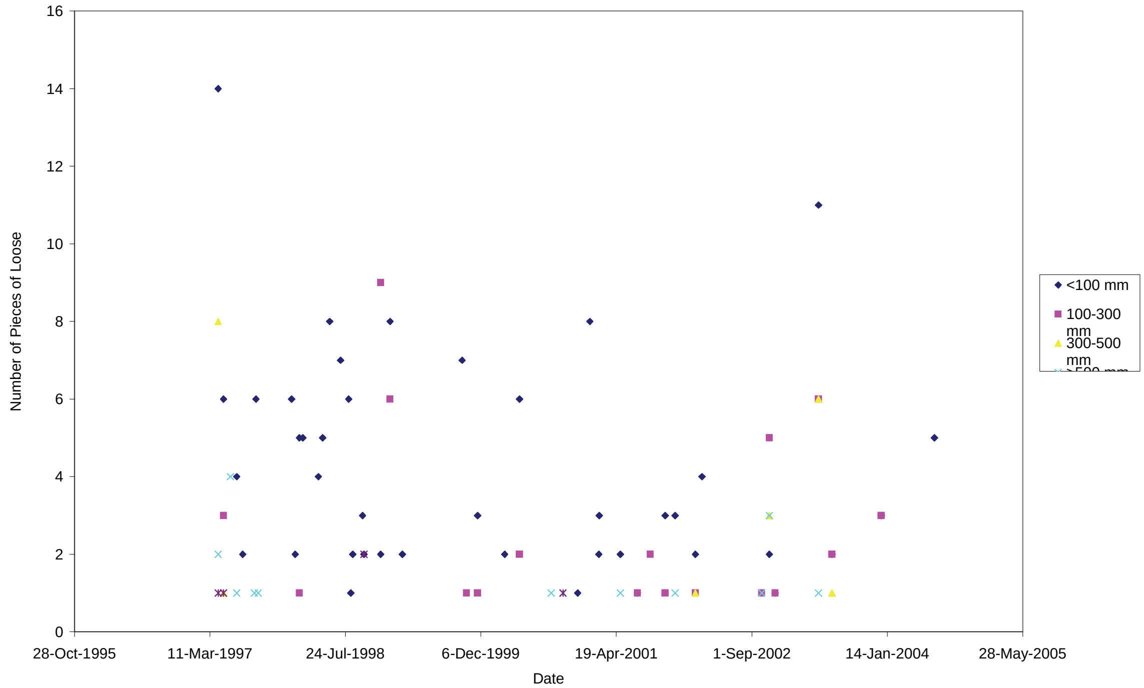


Figure B. 39: Amount of Loose Rock Scaled from Room 421 on the 420 Level of the URL

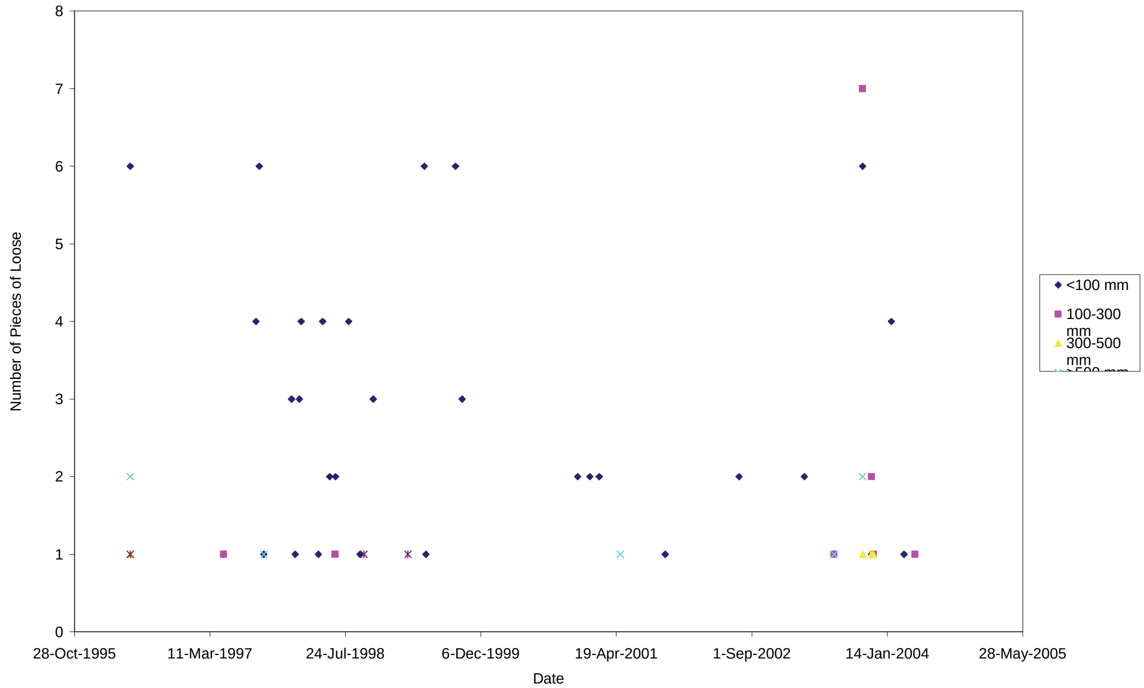


Figure B. 40: Amount of Loose Rock Scaled from Room 423 on the 420 Level of the URL

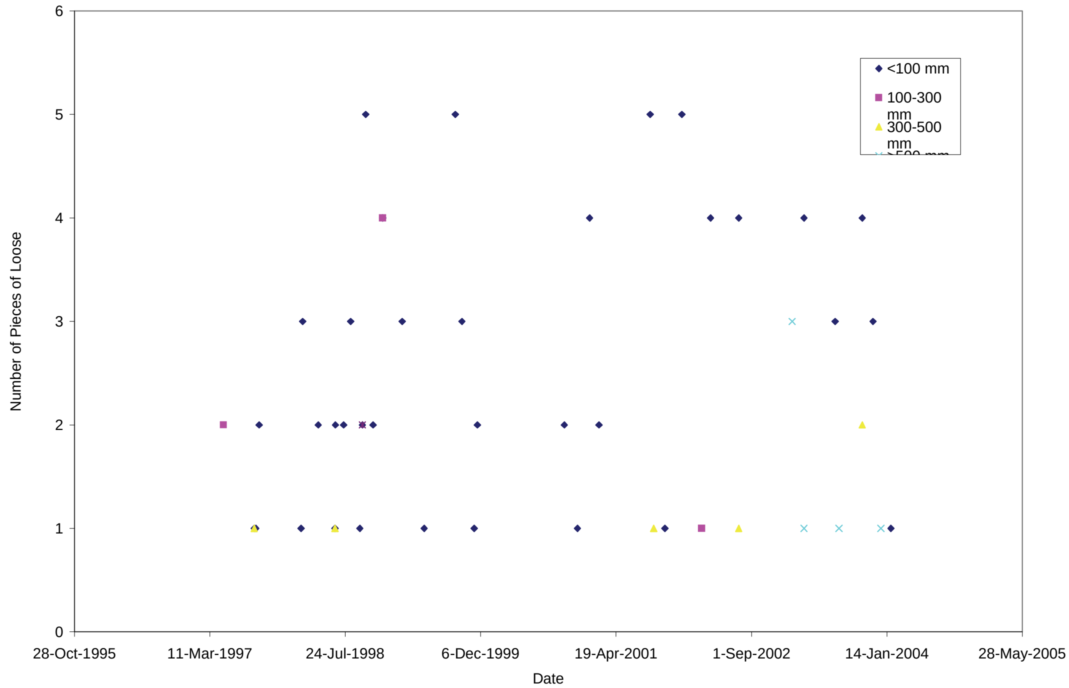


Figure B. 41: Amount of Loose Rock Scaled from Room 424 on the 420 Level of the URL

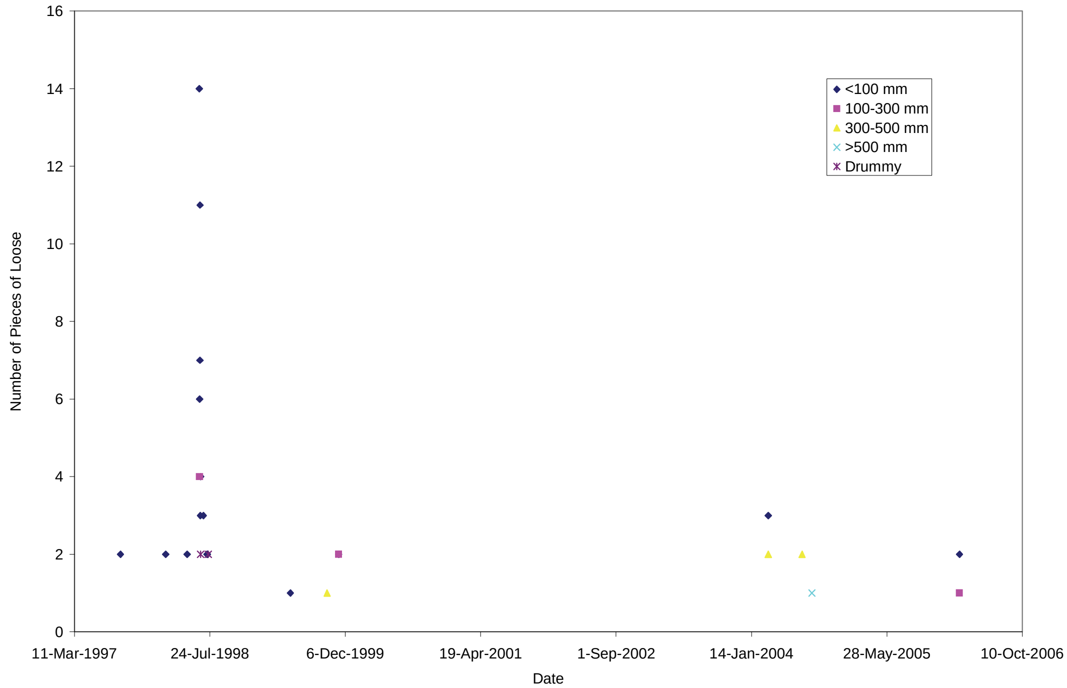


Figure B. 42: Amount of Loose Rock Scaled from Room 425 on the 420 Level of the URL