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Yucca Mountain Site Characterization Project

A Review of the Available Technologies for Sealing a Potential Underground Nuclear Waste Repository at Yucca Mountain, Nevada

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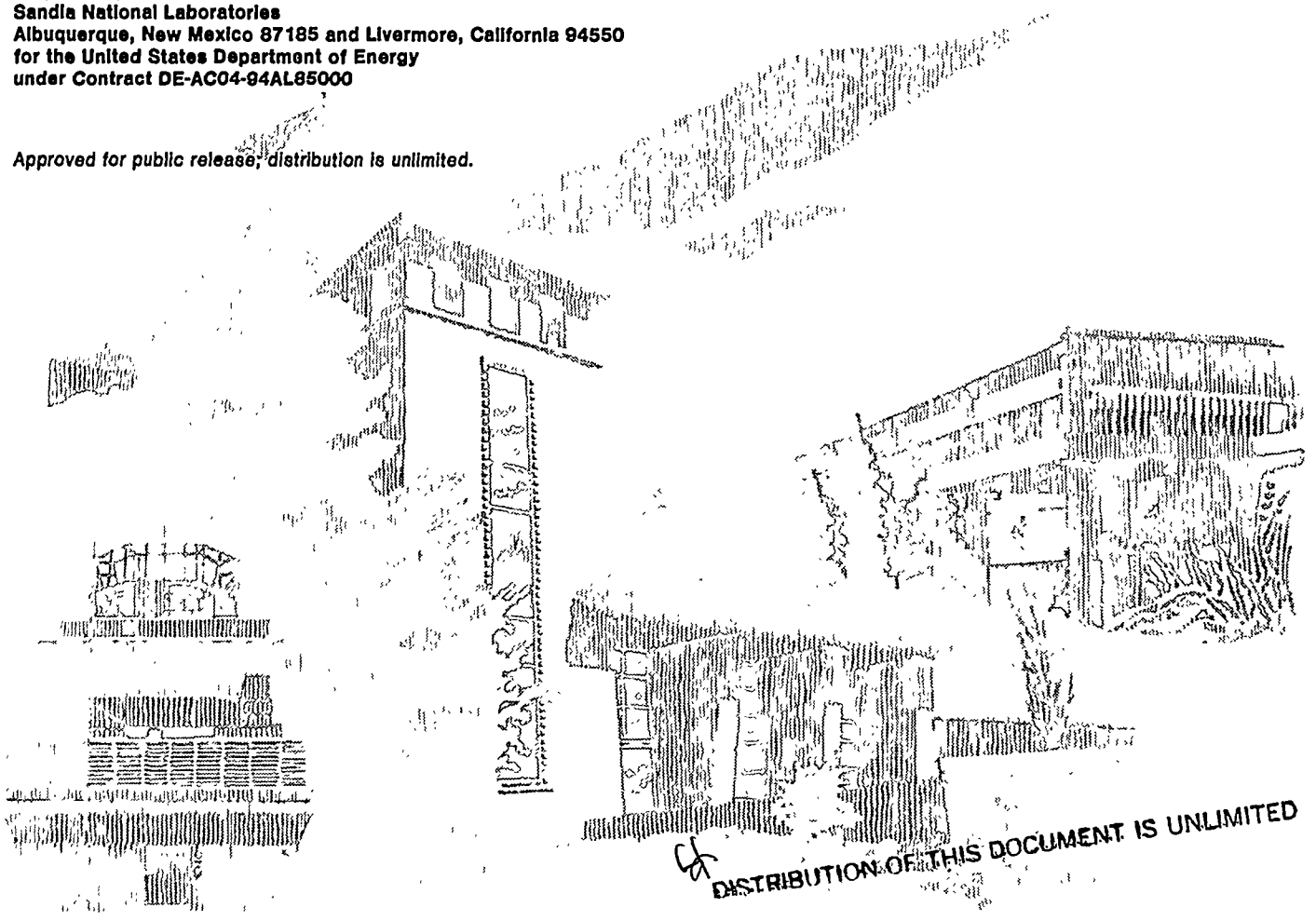

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A Review of the Available Technologies for Sealing a Potential Underground Nuclear Waste Repository at Yucca Mountain, Nevada

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Abstract

The purpose of this report is to assess the availability of technologies to seal underground openings. The technologies are needed to seal the potential high-level radioactive waste repository at Yucca Mountain. Technologies are evaluated for three basic categories of seal components: backfill (general fill and graded fill), bulkheads, and grout curtains. Not only is placement of seal components assessed, but also preconditioning of the placement area and seal component durability. The approach taken was: First, review selected sealing case histories (literature searches and site visits) from the mining, civil, and defense industries; second, determine whether reasonably available technologies to seal the potential repository exist; and finally, identify deficiencies in existing technologies. It is concluded that reasonably available technologies do exist to place backfill, bulkheads, and grout curtains. Technologies also exist to precondition areas where seal components are to be placed. However, if final performance requirements are stringent for these engineered structures, some existing technologies may need to be developed. Deficiencies currently do exist in technologies that demonstrate the long-term durability and performance of seal components. Case histories do not currently exist that demonstrate the placement of seal components in greatly elevated thermal and high-radiation environments and in areas where ground support (rock bolts and concrete liners) has been removed. The as-placed, in situ material properties for sealing materials appropriate to Yucca Mountain are not available. One example includes the impact of rock fill placement operations on the rock fill properties.

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Table of Contents

List of Figures x
List of Tables xiii

Executive Summary xv

1.0 Introduction 1-1

1.1 Purpose, Scope, and Organization of Report 1-1
1.2 Basis for Technology Needs 1-3

1.2.1 Repository Design Features 1-3
1.2.2 Site Description 1-6
1.2.3 Sealing Concepts 1-7
1.2.4 Integration of Backfill and Sealing with Repository Design and Decommissioning Plans 1-9

1.2.4.1 Construction and Operational Considerations—Removal of Foreign Materials 1-10
1.2.4.2 Construction and Operational Considerations—Rock Mass Permeability Modifications 1-11

1.3 Required Technologies 1-12

2.0 Available Technologies for Backfill Placement 2-1

2.1 Introduction 2-1
2.2 Fill Materials and Materials Handling 2-3

2.2.1 Introduction 2-3
2.2.2 Uncemented Fill 2-3
2.2.3 Cemented Fill 2-5
2.2.4 Materials Handling 2-7

2.2.4.1 Rock Fill 2-7
2.2.4.2 Sand Fill 2-10

2.3 Emplacement Methods 2-12

2.3.1 Mining Systems with Limited Vertical Extent 2-12

2.3.1.1 Mechanical Emplacement 2-12
2.3.1.2 Pneumatic Emplacement 2-13
2.3.1.3 Manual Emplacement 2-17
2.3.1.4 Hydraulic Emplacement 2-17

Table of Contents (Continued)

2.3.2 Mining Systems with Large Vertical Extent 2-18

2.3.2.1 Mechanical Emplacement 2-18
2.3.2.2 Gravity Dropping 2-18
2.3.2.3 Hydraulic Emplacement 2-20

2.4 Compaction Methods 2-22
2.5 Engineering Properties of Backfill 2-23

2.5.1 Particle-Size Gradation 2-23
2.5.2 Percolation Rate 2-28
2.5.3 Backfill Physical and Material Properties 2-29

2.5.3.1 Mining Applications 2-29
2.5.3.2 Civil Applications 2-32

3.0 Available Technology for Bulkhead Installation 3-1

3.1 Underground Mining and Mined-Space Storage Applications 3-1

3.1.1 Inundation Plugs 3-1
3.1.2 Ventilation Stoppings 3-3
3.1.3 Hydraulic Fill Bulkheads 3-3
3.1.4 Abandonment Bulkheads 3-3
3.1.5 Consolidation Bulkheads 3-5
3.1.6 Conversion Bulkheads 3-5

3.2 Civil Applications 3-5

3.2.1 Dam Diversion Plugs 3-11
3.2.2 Hydropower Tunnel Plugs 3-11
3.2.3 Lake Taps 3-11

3.3 Defense Applications 3-11
3.4 Design and Materials 3-12

3.4.1 General 3-12
3.4.2 Design Philosophy 3-12
3.4.3 Performance Criteria 3-13
3.4.4 Emplacement Environment 3-13
3.4.5 Bulkhead Geometry 3-14
3.4.6 Bulkhead Materials 3-21

Table of Contents (Continued)

3.5 Construction Method 3-27

3.5.1 General/Site Preparation 3-27
3.5.2 Block Bulkheads 3-27
3.5.3 Cast-in-Place Concrete Bulkheads 3-27
3.5.4 Grouted Concrete Bulkheads 3-28

4.0 Available Technology for Grouting 4-1

4.1 General 4-1
4.2 Materials 4-3

4.2.1 Chemical Mixtures 4-3
4.2.2 Cementitious Materials 4-4
4.2.3 Clay Materials 4-7

4.3 Grouting Methods 4-8

4.3.1 General 4-8
4.3.2 Equipment 4-9
4.3.3 Procedures 4-11

4.3.3.1 Fractured Rock Masses 4-11
4.3.3.2 Contact Grouting 4-14

5.0 Application of Available Technologies to Repository Sealing 5-1

5.1 Work Environment 5-1
5.2 Preconditioning 5-2

5.2.1 Concrete Liner Removal 5-2
5.2.2 Rockbolt Removal 5-4
5.2.3 Postclosure Drainage Area 5-11

5.3 Seal Component Placement 5-11

5.3.1 General Fill, Drifts and Ramps 5-11

5.3.1.1 Criteria 5-11
5.3.1.2 Screening 5-13
5.3.1.3 Suggested Sequence 5-16
5.3.1.4 Availability of Technology 5-18

Table of Contents *(Continued)*

5.3.2 Graded-Fill Structures, Drifts and Ramps 5-21

5.3.2.1 Criteria 5-21
5.3.2.2 Screening 5-21
5.3.2.3 Suggested Sequence 5-23
5.3.2.4 Availability of Technology 5-23

5.3.3 Bulkheads and Plugs, Drifts and Ramps 5-23

5.3.3.1 Criteria 5-23
5.3.3.2 Screening 5-26
5.3.3.3 Suggested Sequence 5-26
5.3.3.4 Availability of Technology 5-26

5.3.4 General Fill, Shafts 5-29

5.3.4.1 Criteria 5-29
5.3.4.1 Suggested Sequence 5-30
5.3.4.3 Availability of Technology 5-30

5.3.5 Bulkheads and Plugs, Shafts 5-30

5.3.5.1 Criteria 5-30
5.3.5.2 Suggested Sequence 5-33
5.3.5.3 Availability of Technology 5-33

5.4 Some Long-Term Performance Concerns 5-33

5.4.1 Backfill Size Control (Fines Creation) 5-33
5.4.2 Seal Durability 5-46

6.0 Conclusions 6-1

7.0 References 7-1

Appendix A.1 - Cannon Mine A.1-1
Appendix A.2 - Billie Mine A.2-1
Appendix A.3 - Apex Mine A.3-1
Appendix A.4 - Sullivan Mine A.4-1
Appendix A.5 - Castlegate Mine A.5-1
Appendix A.6 - Bullfrog Mine A.6-1
Appendix A.7 - American Girl Mine A.7-1

Table of Contents (Continued)

Appendix A.8 - Basalt Waste Isolation Program (BWIP) Near-Surface Test Facility (NSTF) A.8-1
Appendix A.9 - USBM Lake Lynn Laboratory A.9-1

Appendix B.1 - Helms Pumped Storage B.1-1
Appendix B.2 - New Waddell Dam B.2-1
Appendix B.3 - Vat Tunnel B.3-1

Appendix C.1 - Nevada Test Site (NTS) C.1-1
Appendix C.2 - Waste Isolation Pilot Plant C.2-1
Appendix C.3 - Eagle Mine C.3-1

Appendix D - Estimate of Materials in Repository D-1

List of Figures

Figure	Title	
1-1	Location Map for Sealing Case Histories Evaluated in the Western United States	1-4
1-2	Method of Protecting Floor of Postclosure Drainage Area During Construction and Operation Periods	1-13
2-1	Typical Fill-Size Distributions Used in Backfilling Operations at Mount Isa	2-4
2-2	Surface and Underground Rock Fill Handling System at Mount Isa	2-8
2-3	Schematic of Surface Cemented Rock Fill Batch Plant	2-9
2-4	Flowsheet of Materials Handling System to Produce Sand Fill from Mined Waste Rock	2-11
2-5	Truck-Mounted Belt Slinger Used to Fill the Meggan Mine	2-14
2-6	Typical Equipment for Pneumatic Stowing	2-15
2-7	Construction Drawing of the Pneumatically Stowed Muck Pile Stopping	2-16
2-8	Diagram Showing Effect of Fill Delivery Raise Inclination on Filling Process	2-19
2-9	Idealized View of Gravity Filling Open Stope Using Both Cemented and Uncemented Fills	2-20
2-10	Schematic of Delivery, Placement, and Drainage of Hydraulic Fill in a Cut-and-Fill Stope	2-21
2-11	Particle-Size Gradations for Various Waste Rock and Gravels Considered for Backfills	2-25
2-12	Variation in Particle-Size Gradation for Different Crushing Sizes and for Removal of Fine Materials	2-26
2-13	Attrition Rate of Rhyolite/Andesite Aggregate in Fill Raises at Kidd Creek	2-27
2-14	Plot of K_{min} Versus D_{10}	2-29
2-15	Variation of ϕ with Confining Pressure for Medium-Dense and Dense Crushed Basalts	2-33
3-1	Precautionary Plug Installation, Virginia Merriespruit Boundary, South Africa	3-2
3-2	Emergency Plug Installation, West Driefontein Mine, South Africa	3-2
3-3	Concrete-Block Bulkhead Used at Mount Isa for Retaining Hydraulic Fill	3-4
3-4	Abandonment Plug Concepts for Underground Coal Mines	3-6
3-5	Abandonment Plug Concepts for Mine Shafts	3-7
3-6	Seal Design for Two Coal Mine Shafts, Oneida Mine, PA	3-8

List of Figures (Continued)

Figure	Title	
3-7	Temporary Consolidation Bulkheads Used to Grout Unexpected Water Inflows	3-9
3-8	Complex Bituminous Shaft Seal Used to Convert Potash Mine for Natural Gas Storage	3-10
3-9	Common Shapes for Inundation Plugs	3-15
3-10	Conversion Plug Shapes	3-16
3-11	Dimensions of Experimental Plug at West Driefontein, South Africa	3-17
3-12	Nomograph for Determining Required Plug Length	3-22
3-13	Design Curves and Test Results for 0.45-m-Thick Concrete Block Bulkhead Used to Retain Hydraulic Fill at Mount Isa	3-23
3-14	Method of Sealing Abandoned Coal Mine Entries from Surface When Underground Access Is Not Available	3-29
3-15	Method of Constructing Bulkheads to Control Water Inflows in South African Mines	3-31
3-16	Use of Bulkheads to Handle Groundwater Inrushes in Hungarian Mines	3-32
4-1	Approximate Penetrability of Various Classes of Grouts	4-2
4-2	Particle Size of Type I and Type III Portland Cement Compared to Two Microfine Cements	4-5
4-3	Typical Grouting System	4-10
4-4	Helical Rotor Pump	4-11
4-5	Procedures for Grouting Fractured Rock Masses	4-12
4-6	Multiple Packer Sleeved (MPSP) Pipe in Place (left); Single Zone Being Grouted (right)	4-13
4-7	Grouting Pattern for Shaft Sinking	4-15
4-8	Allowable Pressures for Normal Grouting Conditions	4-16
5-1	Production Cycle for Breaking and Removing the Liner and Placing the Backfill	5-3
5-2	Removing Pieces of Concrete Using the Orange-Peel-Grab Unit	5-5
5-3	Typical Sequence for Concrete Liner and Roadway Removal in a Drift	5-6
5-4	Alternate Methods for Liner Removal (a, b) and Roadway Removal (c, d)	5-7
5-5	Existing Method of Removing Bolts with Preinstalled Pull Collar and a Crow's Foot	5-8
5-6	Concept for Removing Grouted Bolts Using Grapple Puller	5-8
5-7	Concept for Removing Grouted Rockbolts Using Cam Locks	5-9
5-8	Concept for Removing Grouted Bolts Using Reaming Bit	5-9
5-9	Concept for Removing Grouted Rockbolts Using Large-Diameter Reaming Bit	5-10
5-10	Concept for Removing Grouted Rockbolts by Overcoring	5-10

List of Figures (Continued)

Figure	Title	
5-11	Decommissioning of Postclosure Drainage Area	5-12
5-12	First Screen for General Fill Structures	5-14
5-13	Methods for Backfilling	5-17
5-14	Compaction Methods and Their Application	5-19
5-15	Methods for Compacting Fill in Tight Areas Using Nonavailable Equipment	5-20
5-16	First Screen for Graded-Fill Structures	5-22
5-17	Suggested Sequence for Placing Graded-Fill Structures	5-24
5-18	First Screen for Selection of Bulkhead Construction Methods	5-27
5-19	Suggested Sequence for Constructing a Tapered Cementitious Plug	5-28
5-20	Typical Sequence for Sealing Repository Shaft	5-31
5-21	Material Flow Diagram for Material Preparation and Handling of Backfill	5-35
5-22	Schematics of Colorado School of Mines Linear Cutting Machine	5-37
5-23	Size Distributions for Rock Broken by Various Methods	5-39
5-24	Material Flow Diagram—Tuff Handling	5-40
5-25	Special Concepts for Discharge Systems to Minimize Degradation	5-42
5-26	Schematic Diagram of Crushing and Screening of Reclaimed Material for Backfilling	5-44
5-27	Grain Size Distribution of Fill Material as Fed into the System and Initial Gradation Prior to Stowing	5-47
5-28	Effect of Laboratory and Field Compaction on Degree of Breakage of Lateritic Gravels	5-48
5-29	Initial Screen for Repository Sealing Materials	5-52
5-30	Settlement of Granular Materials	5-55
5-31	Separation Around Cement Plug Caused by Seismic Loading	5-58
5-32	Examples of Cementitious Grout Curtain Dissolution Under High Hydraulic Gradients	5-59

List of Tables

Tables	Title	
1-1	Selected Key Issues, Issues, and Information Needs Significant to the YMP Repository Sealing Program	1-2
2-1	Comparative Laboratory Properties of Cemented Hydraulic Fill and Cemented Rock Fill	2-6
2-2	Backfill Properties from Large-Scale Laboratory Testing	2-31
2-3	Experimental Stress-Strain Parameter for Coarse Oil Shale	2-32
3-1	Summary of Test Results on Experimental Plug	3-18
3-2	Rectangular Bulkhead Design Equations and Methods	3-19
3-3	Typical Plug Dimensions and Pressure	3-21
3-4	Mining Industry Stopping Construction Types and Materials from Survey of 19 Operating Mines in North America	3-24
3-5	Ultimate Compressive, Tensile, and Shear Strengths for 1:2:4 Concrete Mix	3-25
3-6	Cement Replacement Mixes Previously Used by Cementation Mining Ltd. for Underground Plugs	3-26
5-1	Deterioration of Sealing Components	5-50

Executive Summary

The Yucca Mountain Site Characterization Project (YMP), managed by the U.S. Department of Energy, is examining the suitability of disposing of high-level radioactive waste in a mined geologic repository at Yucca Mountain. Yucca Mountain is located on and adjacent to the Nevada Test Site (NTS), Nye County, Nevada. The potential repository would be located in an unsaturated tuff formation within Yucca Mountain. Current plans call for sealing a potential repository after waste is emplaced to contribute to the containment of radionuclides within the repository. Specifically, seals may be required to limit water flow into or out of the underground facility, and airflow out of the facility. The Site Characterization Plan (SCP) includes issues and information needs pertaining to the YMP Repository Sealing Program that must be resolved for licensing.

The purpose of this report is to address Key Issue 4 from the SCP, and more specifically Information Need 4.4.10, which calls for a determination that the seals for shafts, drifts, and boreholes can be placed with reasonably available technology. The scope includes: (1) reviewing selected sealing case histories through literature searches and site visits, (2) determining whether reasonably available technologies exist to seal a potential repository at Yucca Mountain, and (3) identifying deficiencies in existing sealing technologies, if any. The case histories evaluated are selected examples from civil, mining, and defense industries that have used sealing technologies, and are organized by basic seal component type.

Present plans (Fernandez et al., 1987) call for three basic categories of seal components: (1) backfill (general fill and graded fill), (2) bulkheads, and (3) grout curtains. These components will be appropriately located throughout the repository to meet the sealing objectives, with the final locations integrated with the repository design. At the time of this writing, the repository design of record is the SCP/CDR design, although that design will most likely be modified before construction. Key unsaturated-zone repository sealing objectives are to (1) prevent collection of possible groundwater in emplacement areas by diverting it using graded-fill structures into drainage areas, where it can drain into the fractured rock of the floor and (2) preserve the structural integrity of the host medium thereby reducing the potential for development of new pathways for radionuclide migration. Therefore, the graded backfill system becomes an underground water-control system with the general fill inhibiting collapse of any and all openings. General fill will be placed in all repository openings during decommissioning and will require placement of approximately

340 million cubic feet of in-place fill. Bulkheads and grout curtains will be used to seal access openings to prevent them from becoming preferential pathways for entry of surface water as well as escape of gases.

The required sealing technologies fall into three areas: (1) preconditioning, (2) seal component placement, and (3) seal durability. Preconditioning of the seal placement area would include potential removal of concrete floors, concrete liners, and rock bolts at required locations, keyway construction for bulkheads, and restoration or enhancement of rock mass permeability at selected drainage locations. Placement of seal components may involve processing, mixing and blending, transporting, and placing seal materials. Effective placement of seal components is a factor in seal component durability, an important consideration involving modification of seal permeability and other properties from the action of static, dynamic, or thermal loads or other environmental alterations. Specific concerns include settlement of the backfill due to dynamic loads; fines migration down the hydraulic gradient in general backfill; cracking or rock-seal interface debonding due to seismic, thermal, or geochemical forces; and dissolution of cementitious materials.

Available technologies for placing backfill include mechanical, pneumatic, manual, hydraulic, and gravity methods. To reduce the number of case histories reviewed, screens were applied to the various available technologies to eliminate those not suitable for present repository concepts, based on various criteria. "Must" criteria for placing general fill include (1) minimizing water use, (2) using existing penetrations for fill transport, and (3) having adequate backfilling capacity for the large volumes to be filled. These criteria were applied in a preliminary screening, eliminating manual, hydraulic, and gravity methods for the general fill. The remaining technologies (mechanical and pneumatic) were reviewed by studying nine case histories selected for their general applicability to some aspect of repository backfill placement, and documented in Appendix A of this report. These include (1) Cannon Mine, (2) Billie Mine, (3) Apex Mine, (4) Sullivan Mine, (5) Castlegate Mine, (6) Bullfrog Mine, (7) American Girl Mine, (8) BWIP Near-Surface Test Facility, and (9) USBM Lake Lynn Laboratory. The seven mining case histories represent a broad cross section of mechanical and pneumatic backfilling practice in western North American hard-rock mines, and illustrate the success of mechanical methods using rubber-tired equipment and the limitations of existing pneumatic stowing technology where high fill production rates, long horizontal transport distances, and flexibility of backfill sequencing are required. The BWIP case study illustrates the decommissioning of stable tunnels with low extraction ratios, and would be similar to backfilling nonemplacement areas at a potential repository. The

Lake Lynn Laboratory shows recent equipment innovations in two areas: (1) to overcome some of the wear problems with existing pneumatic stowing equipment, and (2) to place fill and monitor the progress and quality of filling remotely. The selected case histories were documented in detailed appendices because appropriate documentation does not exist in the literature. A number of case histories of backfilling for civil projects are available in the published literature and the technology adequately described in various textbooks, so further documentation was not considered necessary.

Bulkhead placement technology can be categorized by material type such as blocks, cast concrete, and grouted concrete, with the concrete plugs placed from the surface or underground. A preliminary screening eliminated the bulkheads placed from the surface based on a criterion to use existing surface penetrations. The remaining technologies were selectively illustrated with three case histories: (1) Nevada Test Site (NTS), (2) Waste Isolation Pilot Plant (WIPP), and (3) Eagle Mine. The NTS illustrates unique and complex sealing technology for stemming tunnel complexes and large-diameter boreholes with cementitious and earthen materials and was recommended by the Presidentially appointed Nuclear Waste Technical Review Board (NWTRB) as an important technology resource. WIPP demonstrates the technology developed to emplace both earthen and cementitious seals in a nuclear waste repository in salt, while the Eagle Mine documents the use of simple cementitious plugs to seal abandoned mine openings. These case histories are described in Appendix C.

Available technologies for grouting can be categorized by emplaced material type such as chemical, cement, and clay technologies. A preliminary screening eliminated chemical grouts from serious consideration based on possible toxicity, geochemical interactions, and durability concerns. Cementitious grouting technologies were illustrated by selecting three pertinent case histories from the many available: (1) the Helms Pumped Storage Project, which demonstrates the effective use of microfine cement to seal an isolated shear zone subject to high water pressure, (2) the New Waddell Dam, which highlights state of the art in cement grouting to seal a large dam foundation in fractured volcanic rock, and was also recommended by the NWTRB, and (3) the VAT Tunnel, which illustrates the technologies of sealing high water inflow and handling and transporting large volumes of concrete. Grouting case histories are presented in Appendix B.

Suitable case histories for preconditioning repository openings prior to sealing were not available. However, rockbolt removal and concrete liner and floor removal are occasionally practiced on a limited basis, and concepts are presented to modify existing equipment to

perform these tasks if they are required for repository sealing. Experience with dam foundation grouting suggests it is very difficult to clean fine material from fractures to enhance permeability for grouting. This implies that special drainage areas be designated and suitably protected during repository development/operations to minimize clogging of important fracture systems. Available technology consists of block floors for protection from traffic, and systems of sumps to capture fines before allowing water to enter the formation.

Some long-term performance concerns with the sealing components were addressed because they will directly affect the ultimate selection of sealing technologies. The sizing of backfill and control of fines is important because of the concern that fractures will be clogged with migrating fines, inhibiting drainage. Seal durability was considered because it affects both seal material selection and placement technology.

Conclusions were reached regarding the availability of technology in seven key areas. In four areas, available technology was either considered adequate or new technology could easily be developed. These include:

(1) Technology for placement of general backfill in underground openings. Under moderate temperature conditions ($<100°F$), this is available off-the-shelf and is being routinely used in many operations today. Backfill is also routinely placed and compacted on surface to very exacting specifications during civil construction. However, if very stringent specifications for fines content, degree of compaction, and tightfilling are developed for placing underground backfill, then new equipment and technology may require development.

(2) Technology exists for the placement of large-scale bulkheads in underground shafts and drifts at moderate temperatures.

(3) Technology exists to place grout in fractured rock masses at moderate temperatures.

(4) Technologies exist or could easily be developed to precondition areas where seal components are to be placed, although this is a nonroutine operation in civil and industrial practice.

Three areas were discovered where adequate technology does not exist and where development will be necessary:

(1) Technology does not currently exist to demonstrate the long-term durability and performance of seal components.

(2) Case histories are not available that adequately document sealing component placement or performance under greatly elevated temperatures, high-radiation environments, and potentially unstable underground openings.

(3) The as-placed, in situ properties of components are not available.

One example includes the impact of rock fill placement operations on the rock fill properties. Bench scale and field testing such as that described in the Sealing Field Test Definition Report (Fernandez et al., 1993), will be required to address these issues, especially if significance performance is ultimately allocated to the sealing system. This testing program must be integrated with and provide input to both the evolving repository design (e.g., gross thermal loading and borehole versus in-drift emplacement) and performance assessment efforts so that suitable technologies can be developed to completely resolve the issue of available technologies.

Acronyms and Abbreviations

ASTM	American Society for Testing and Materials
DOE	U.S. Department of Energy
ESF-ACS	Exploratory Shaft Facility, Alternative Configuration Study
LCM	linear cutting machine
LHD	load-haul dump
MPSP	multiple packer sleeved pipe
NTS	Nevada Test Site
NWTRB	Nuclear Waste Technical Review Board
RALF	rotary air-lock feeder
SCP-CDR	Site Characterization Plan, Conceptual Design Report
TBM	tunnel boring machine
USBM	U.S. Bureau of Mines
USBR	U.S. Bureau of Reclamation
YMP	Yucca Mountain Site Characterization Project
°C	degrees Celsius
cm	centimeter(s)
cm/sec	centimeters per second
ft	feet/foot
ft^2	square feet
ft^3	cubic feet
ft/sec	feet per second

in.	inch(es)
kg	kilograms
kg/m^3	kilograms per cubic meter
km	kilometer
kPa	kilopascal
m	meter(s)
m^3	cubic meter(s)
mm	millimeter(s)
MPa	megapascal(s)
μm	micrometer(s)
psi	pounds per square inch

1.0 Introduction

The Yucca Mountain Site Characterization Project (YMP), managed by the U.S. Department of Energy (DOE), is examining the suitability of disposing of high-level radioactive waste in a mined geologic repository at Yucca Mountain. Yucca Mountain is located on and adjacent to the Nevada Test Site (NTS), Nye County, Nevada. The potential repository would be located in an unsaturated tuff formation within Yucca Mountain.

The overall repository system is comprised of many subsystems that must be evaluated to determine the suitability (i.e., acceptable performance) of disposing of waste in Yucca Mountain. One subsystem is the sealing subsystem, which is comprised of sealing components. This report reviews the technologies that are currently available to place these sealing components.

1.1 Purpose, Scope, and Organization of Report

As indicated in the Nuclear Waste Policy Act of 1982 (NWPA, 1983) and in 10 CFR 60 criteria (NRC, 1986), a Site Characterization Plan (SCP) shall be prepared for candidate sites for a nuclear waste repository. The purpose of the SCP is to identify issues at a specific proposed site and the plans for resolving those issues. The site characterization efforts at the proposed sites are, therefore, centered on issues defined in the SCP.

The issues hierarchy is divided into three levels: key issues, issues (including characterization, performance, and design issues), and information needs. Key issues are broad questions of overall suitability, issues are more specific, and information needs are generally data or analyses about the natural or engineered systems required to address the issues. The key issues, issues, and information needs pertinent to the YMP Repository Sealing Program are given in Table 1-1.

As indicated in Table 1-1, there are two key issues pertinent to sealing—key issue 1 dealing with the ability of the mined geologic disposal system in isolating radioactive waste from the accessible environment, and key issue 4 dealing with the development of the repository using reasonably available technology.

The purpose of this report is to address key issue 4 and, more specifically, information need 4.4.10 in sealing shafts and drifts.

Table 1-1

Selected Key Issues, Issues, and Information Needs Significant to the YMP Repository Sealing Program (1-16-87 Version)

KEY ISSUE 1:	Will the mined geologic disposal system at Yucca Mountain isolate the radioactive waste from the accessible environment after closure in accordance with the requirements set forth in 40 CFR Part 191, 10 CFR Part 60, and 10 CFR Part 960?
DESIGN ISSUE 1.12:	Have the characteristics and configurations of the shaft and borehole seals been adequately established to (a) show compliance with the postclosure design criteria of 10 CFR 60.134 and (b) provide information to support resolution of the performance issues?
INFORMATION NEED 1.12.1:	Site, waste package, and underground facility information needed for design of seals and their placement methods.
INFORMATION NEED 1.12.2:	Materials and characteristics for seals for shafts, drifts, and boreholes.
INFORMATION NEED 1.12.3:	Placement methods for seals for shafts, drifts, and boreholes.
INFORMATION NEED 1.12.4:	Reference design of seals for shafts, drifts, and boreholes.
KEY ISSUE 4:	Will mined geologic disposal system construction, operation (including retrieval), closure, and decommissioning be feasible at Yucca Mountain on the basis of reasonably available technology, and will the associated costs be reasonable in accordance with the requirements set forth in 10 CFR Part 960?
DESIGN ISSUE 4.4:	Are the repository construction, operation, closure, and decommissioning technologies adequately established to support resolution of the performance issue?
INFORMATION NEED 4.4.10:	Determination that the seals for shafts, drifts, and boreholes can be placed with reasonably available technology.

The approach taken to address information need 4.4.10 involved three steps: (1) review selected sealing case histories through literature searches and site visits, (2) assess the availability of technologies to seal the potential repository at Yucca Mountain, and (3) identify deficiencies in existing sealing technologies, if any. This assessment is divided into three areas:

- backfill placement technologies (Chapter 2),
- bulkhead technologies (Chapter 3), and
- grouting technologies (Chapter 4).

In Chapter 5, the application of existing technologies to repository sealing is presented. Chapter 6 provides a summary of the conclusions reached in this report, including identification of deficiencies identified in technology availability.

The case histories evaluated are selected examples from civil, mining, and defense industries that have used sealing technologies. The locations of these case histories are illustrated in Figure 1-1. Descriptions of these case history evaluations are given in Appendices A, B, and C for backfilling, grouting, and bulkhead technologies, respectively. In Appendix C.1, a comprehensive assessment of the sealing technologies used at the NTS is given. This assessment of the NTS operations was specifically requested by the Congressionally created Nuclear Waste Technical Review Board (NWTRB) in their fifth report to Congress (NWTRB, 1992). It is intended that this report (through Appendix C.1) satisfy that request.

1.2 Basis for Technology Needs

The technology used to emplace seals depends, to a large degree, on the design of the repository, the site conditions, and the desired performance of the sealing component. Technologies are also needed to prepare the location where the seal components will be placed. These are referred to here as preconditioning or decommissioning technologies. The sections below provide a brief description of each of the areas.

1.2.1 Repository Design Features

The repository underground layout and design have developed from a preliminary conceptual design by Dravo (1984) through the conceptual design described in the Site Characterization Plan, Conceptual Design Report (SCP-CDR) (SNL 1987) and more recently by the adoption of the Exploratory Shaft Facility, Alternative Configuration Study (ESF-ACS) recommendations for repository configuration compatible with the tunnel boring

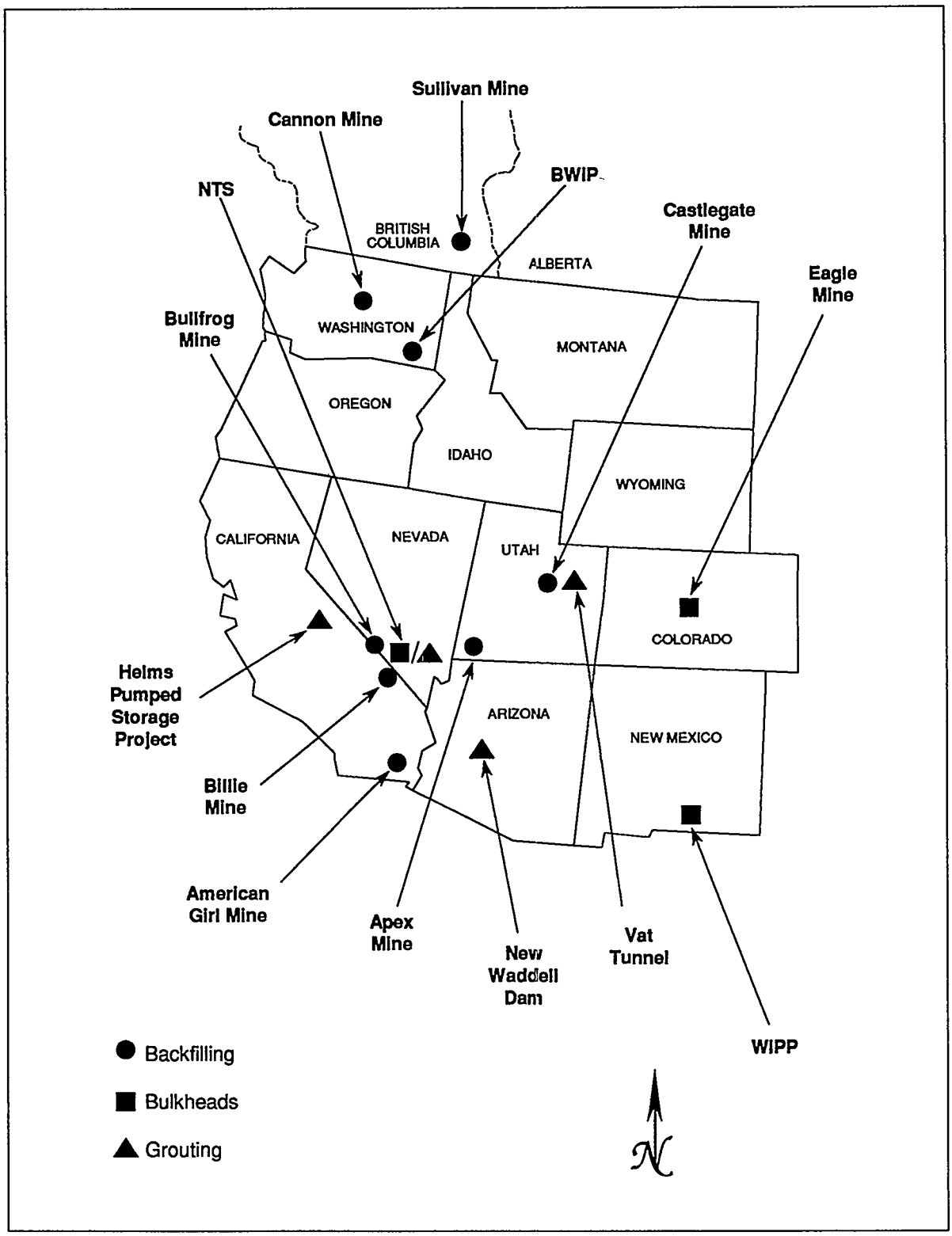


Figure 1-1

Location Map for Sealing Case Histories Evaluated in the Western United States

machine (TBM) drift excavation method (Option 30). The current design of record is the SCP-CDR design.

The repository designs proposed for the Yucca Mountain site have some basic similarities. They all propose a single-level repository located approximately 300 meters (m) below the surface and accessed by ramps and shafts. The waste containers would be transported to the waste storage areas through ramps. Shafts are used for ventilation and men and material access to the repository level. The waste containers could be stored in the repository either in short vertical boreholes in the floor or in long horizontal boreholes in the walls of the storage rooms. A recent consideration has been to permanently store the waste in the rooms inside transporter casks without placing the waste containers in boreholes in the rock. Four phases are identified in the life of the repository: construction, operations, caretaker, and decommissioning or closure. The caretaker period allows for up to 50 years of retrievable storage. Backfilling and sealing would be part of the decommissioning phase. Retrieval of part or all of the waste could occur during the caretaker period.

To assess the adequacy of available technology to backfill and seal the repository, the configuration of the repository and the environmental conditions under which the backfill is placed, and should perform, must be identified. Until the final design is developed, and quite possibly until the repository is actually built, the configuration and environment at the time of backfilling and sealing will not be known. Hence, at this stage in the development of the repository design, the range of possible repository layouts and placement environments that might emerge as the final repository design is considered in assessing the adequacy of the available technology for backfilling and sealing.

The SCP-CDR repository design features considered in this report include:

- access to repository by shafts and ramps;
- excavation of access and emplacement drifts by conventional (drill and blast) and mechanical means (tunnel boring machine);
- opening shapes from circular, rectangular, or square to inverted horseshoe shape;
- ground support including fully grouted rockbolts, shotcrete, and combination of both;

- size of openings, up to 25 feet (ft) in diameter to as low as 16 ft in diameter;
- roadways comprised of concrete or crushed tuff;
- emplacement of waste container in boreholes; and
- temperatures from ambient to 50 degrees Celsius (°C) in access drift in first 50 years to limit of 100°C in emplacement drift (Hertel and Ryder, 1991).

The repository design is in a state of evolution. Important design issues, such as allowable gross thermal loading, and borehole versus in-drift emplacement, are unresolved and may have major impacts on required sealing technologies. Hence, although this report by necessity emphasizes technology needs for the present design of record, it should be recognized that needs may change; specifically, the environmental conditions and geometry of seal placement.

1.2.2 Site Description

Volcanic rocks are the predominant rock types at Yucca Mountain and are primarily ash flow in origin, but ash fall, bedded, and reworked tuffs are also present. In general, the tuffs are quite variable in their degree of welding and alteration.

Following deposition, modifications to the tuffs depend on the temperature and pressure conditions, rate of cooling, original composition of the tuff, thickness of the tuffaceous deposit, and subsequent exposure to groundwater. As a result, the modifications that can result include the following:

- varying degrees of welding, ranging from nonwelded to densely welded tuff;
- petrographic differences in the tuff, which can be classified on the basis of three physical components—glass, crystals, and lithic fragments;
- lithophysae development;
- fracture development; and
- digenetic alterations of the tuff, including smectite and zeolite development.

Seal component placement can occur over a broad range of geologic and hydrologic conditions. Access to the repository will necessitate penetration through welded units (Tiva Canyon and Topopah Spring Members) and nonwelded to partially welded units (Pah Canyon, Yucca Mountain Members, and the tuffaceous beds of Calico Hills). The majority of the excavation, and consequently seal component placement, will be in the Topopah Spring Member. The total of all excavations for the SCP-CDR design is estimated to be 340,000,000 cubic feet (ft^3), of which 330,700,000 ft^3 is estimated to be in the TSw2.

1.2.3 Sealing Concepts

The primary objective of the sealing subsystem is to contribute to the isolation of radioactive waste. This objective is achieved through the following actions (Fernandez et al., 1987):

- reducing the amount of water entering the underground facility through vertical shafts, ramps, or other vertical or horizontal penetrations;
- controlling the flow of water away from the waste container and draining it into nonemplacement areas if water enters the vicinity of the waste container;
- controlling the release of radionuclides from the repository so that the shaft, ramp, and borehole seals does not become preferred pathways; and
- preserving the structural integrity of the host medium.

Sealing components have been proposed that achieve the primary objective through one of the four means described. In addition to the placement of manmade sealing components, an additional strategy proposed in the sealing program is to isolate and drain water encountered in the repository and at the base of the shafts. Drainage can be accomplished through the highly fractured, densely welded tuff of the Topopah Spring unit.

In the SCP (DOE, 1988), tentative sealing components were identified and placed into one of two subsystems: the shaft and borehole seals subsystem or the repository seals and backfill subsystem. Sealing components in shafts, ramps, and boreholes are intended to achieve the primary objective by accomplishing the first and third actions given above. Components associated with the shafts include the following:

- a capillary barrier at the surface to restrict surface water from entering the upper portion of the shaft;

- discrete seals to restrict flow downward into the shaft, to restrict airflow upward, and to reduce settlement effects that could propagate to the surface;

- shaft and borehole fill to restrict water flow downward and airflow upward; and

- the highly fractured, densely welded tuff at the base of the shaft to isolate and drain water in the shaft.

The primary component in the ramps will be the general rock fill placed in the majority of the openings. Periodically, structures, such as a dam with a lower hydraulic conductivity than the material upgradient from it, can be placed to restrict the lateral migration of water flow in the ramps. Drainage can be enhanced by such simple techniques as drilling multiple boreholes in floor areas with a high drainage potential. In exploratory boreholes, seals accomplishing a capillary barrier or fill restricting water flow downward and airflow upward are proposed. While the entire borehole will be sealed, there may be preferred locations to place seals that will better accomplish the objective. Preferred areas may be competent zones within the Paintbrush nonwelded tuff or the upper portions of the Topopah Spring Member, which are not highly fractured.

Within the underground facility sealing subsystem, multiple components are proposed to control various quantities of water inflow. The proposed components would control expected small inflows. More elaborate sealing components, such as the double bulkhead, are proposed in the unlikely event that extremely large inflows occur. These bulkheads are intended to preserve the structural integrity of the host rock. The more probable sealing components that would be emplaced in the repository are simple: a backfilled channel, the drift fill, and a simple dam constructed by placing low-conductivity material adjacent to higher-conductivity material.

From the preceding discussion the sealing components can be placed into one of four categories:

- general fill,
- graded fill or specialty fill,

- bulkheads, and
- grout curtains.

1.2.4 Integration of Backfill and Sealing with Repository Design and Decommissioning Plans

At the time of this writing, the repository design is conceptual, and many aspects of the design that will ultimately define the repository sealing program strategy are evolving. For example, the allowable gross thermal loading for the repository, which defines the environment for seal emplacement, has not yet been finalized. The waste emplacement concept (boreholes vs. in-drift emplacement) is in a state of flux, and the extent to which the backfill near the waste containers is required to be an engineered barrier has not been determined. It is not within the scope of this report to address repository design and backfill strategies, which will evolve and mature throughout the site characterization period.

The hypothetical repository used in this section is loosely based on the design of record, the SCP-CDR design (involving above-boiling rock temperatures), and waste emplacement in boreholes. The general fill ensures long-term ground stability, but has no special requirements as an engineered barrier, although it may perform as a capillary barrier (Fernandez and Freshley, 1984). The strategy of backfilling and sealing the repository after an extended operations and retrievability period is also consistent with the SCP-CDR design.

Deviations from the SCP-CDR design include circular, machine-bored drifts rather than conventionally mined openings; also the possibility of in-drift waste storage and other thermal environments are briefly considered. These concepts are consistent with contemporary repository design philosophy.

The postclosure repository performance can be enhanced by designing the repository openings to maximize postclosure performance and by backfilling and sealing the repository openings. The repository openings can be designed so that water flowing into the repository openings is not focused toward the waste containers and to minimize changes, both structural and chemical, to the material host rock. Materials introduced during mining to stabilize the rock; provide working platforms; or to deliver power, ventilation, air, or water may need to be removed prior to or as part of decommissioning. Such foreign materials might be removed to minimize potentially unfavorable chemical interactions.

Regarding the postclosure performance of the repository, the primary concerns in decommissioning are to remove all foreign materials that might cause deleterious changes to the geochemical environment for which the waste container is designed and to minimize disturbance of the natural permeability of the rock mass in designated drainage areas. The extent of alterations to the postclosure repository environment is largely determined by the layout, the thermal loading, the construction and operations plan, and the decommissioning plan.

1.2.4.1 Construction and Operational Considerations—Removal of Foreign Materials

During decommissioning it may be necessary to remove some or all foreign materials. There are seven categories of foreign materials that will be introduced into the repository during construction and operation: (1) utility systems will be installed in the repository to supply air, water, and electric power to the working areas; (2) mobile equipment and conveyors will be used for transportation of men and materials to and from the active working areas; (3) tangible materials, such as TBM cutters, drill bits, drill steel, etc., will be used but not consumed during construction; (4) consumable materials, such as fuel and lubricants, will be used by men and equipment; (5) ground support will be installed in repository openings for safety; (6) underground structures, such as bulkheads, ventilation stoppings, temporary sumps, temporary floors, rails, and other underground structures, will be installed; and (7) waste storage components, such as the waste container, borehole liners, plugs, and caps will be designed as part of the engineered barrier system.

Utility systems include ventilation tubing, compressed air line, service water pipes, dewatering pipes, electric power cables of various capacities, underground pumping equipment, and electrical equipment. Both construction and operation utilities will be required. Although utilities represent large volumes of installed material, they are expected to have very little importance to the postclosure repository for the simple reason that they are readily removable prior to closure using routine procedures.

Mobile equipment, including TBMs, drills, personnel carriers, etc., will also be removed prior to repository operation, as will any muck-handling systems such as conveyors. Special mobile equipment used during repository operations, such as waste transporters, will also be removed prior to closure as part of normal operations.

Tangible materials will be completely removed from the repository on a routine basis after their service life is over. On the other hand, small amounts of spillage of fuels, lubricants, and other consumables may occur during normal operations. Additionally, minor amounts of rubber from equipment tires and diesel exhaust residue will remain on the floor and walls. These are not routinely removed in civil and mining operations, and if they are considered chemically significant, special cleanup procedures may be required as part of decommissioning.

The ground support category represents the largest amount of foreign material introduced into the repository that is not routinely removed in similar underground operations. Cementitious materials (concrete, grout, shotcrete) and steel (rockbolts, roof trusses, etc.) will be the primary materials introduced. If these materials must be removed because of geochemical or other considerations, special procedures and equipment may be required to enable efficient operations without safety hazards. These technologies are discussed in Chapter 5. Appendix D defines and provides an estimate of the amount of material that could be left in the repository assuming the SCP-CDR design. To minimize material removal, the repository final design should attempt to restrict use of materials in critical areas if they are shown to be deleterious to the chemical environment.

From a sealing perspective, removal of foreign materials may be required at specific locations, such as in the immediate vicinity of a seal, to improve seal quality and eliminate possible sources of leakage. For example, it will probably be necessary to remove all existing ground support in the process of constructing a keyway for a cementitious plug.

1.2.4.2 Construction and Operational Considerations—Rock Mass Permeability Modifications

An additional potential impact on the postclosure repository environment directly resulting from repository operations and backfilling involves changes in rock mass permeability caused by fines migration into natural fracture systems. The SCP identifies the Topopah Springs Formation as a "sealing component," because part of the overall sealing strategy is to use the natural permeability of the rock mass to drain water that enters the repository and, hence, to maintain the waste containers in a dry environment above the water table. This is the only way that water will be able to exit the system during the extended postclosure period; as pumping will not be practical. During the early postclosure period when the local temperature will be above the boiling point of water, water vapor may exit the repository area.

Water entering the underground facility will be drained in the floor of the drifts. Assuming locally saturated conditions, the hydraulic conductivity of the highly fractured, welded tuff portion of the Topopah Springs Member is dominated by fractures. Clogging of the fractures with fine materials could occur. This in turn could reduce the hydraulic conductivity and drainage of the rock. The source of these fines can be formation fines that were deposited during geologic times or fines introduced during placement of the backfill. These will be discussed in later sections. However, there is also considerable potential for fines creation and subsequent permeability reduction during the construction and operations period on the floor or drifts and ramps. Degradation mechanisms include:

- drilling and boring operations,
- blasting,
- mucking and cleanup activities,
- muck spillage, and
- roadway wear from tire action.

In many areas of the repository, drainage into the formation will not be required, and clogging will not be important. However, in postclosure drainage areas, special precautions may be required to protect the floor rock during construction and operation. Figure 1-2 shows a drainage area being temporarily protected from wear caused by traffic using a layer of coarse pea gravel interlocking concrete blocks. Possible water accumulations in the gravel underlying the blocks can be drained as required.

1.3 Required Technologies

The technologies needed in the YMP Repository Sealing Program fall into three areas: preconditioning of the placement area, seal component placement, and seal component durability. As mentioned earlier, there are four basic categories of seal component: general fill, graded fill, bulkheads, and grout curtains. These four basic types would be located in different areas throughout the repository and thereby subjected to various environmental conditions.

Preconditioning of the seal emplacement area would include:

- potential removal of concrete floors and liners,
- potential removal of rockbolts,

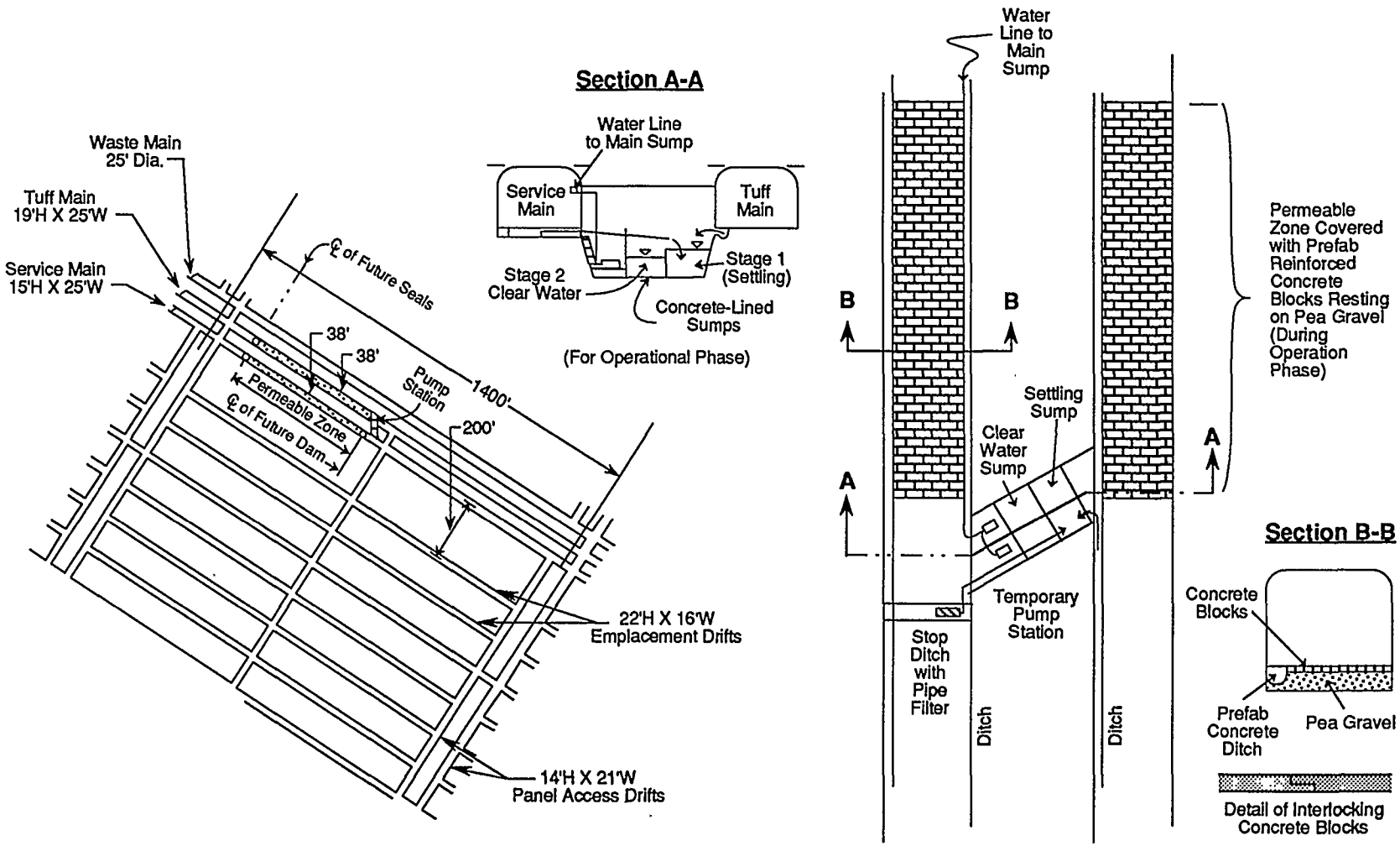


Figure 1-2

Method of Protecting Floor of Postclosure Drainage Area During Construction and Operation Periods

- keyway construction for bulkheads, and
- restoration/enhancement of rock mass permeability of fractured welded tuff.

In general, seal emplacement may involve:

- processing seal materials,
- mixing and blending seal materials,
- transporting seal materials, and
- placement of material.

Assuming that the seal component was effectively placed, an additional consideration would be the seal component durability. This would involve modification of the properties of the seal component due to static, dynamic, or thermal loads, or other environmental alterations. Examples include:

- settlement of the backfill due to dynamic loads;

- fines migration in general backfill due to hydrologic forces;

- cracking or rock-seal interface debonding due to seismic, thermal, or geochemical forces; and

- dissolution of cementitious materials.

Because the desired performance of the seal component is under continued evaluation, a conclusive statement on available technologies and ultimate performance cannot be made in the report; however, general evaluations are provided in Chapter 5.

2.0 Available Technologies for Backfill Placement

2.1 Introduction

This chapter reviews available technologies with potential application to backfilling the excavations in the mined geologic repository. Historically, underground backfilling systems have been developed to maximize resource recovery in active mining operations. Recently, techniques to fill abandoned mines to reduce environmental problems or to prevent surface subsidence have been investigated. Other technologies for placement and treatment of fill in civil structures (for example, earth-filled dams) may also have application in repository backfilling. Civil applications of backfilling are generally well-documented in the technical literature, so this discussion will emphasize mining applications.

Available technologies for underground backfilling are discussed in this section with respect to critical elements of the process. These elements include:

- backfill materials,
- backfill materials handling,
- backfill placement,
- potential postemplacement treatments, and
- engineering properties.

Backfilling in underground mining operations is conducted for various purposes, including:

- control of rock instability,
- assurance of high resource recovery,
- prevention of surface disturbance,
- control of ventilation airflow/groundwater inflow, and
- disposal of surface mill waste or mined waste rock.

Backfill materials and systems are engineered to achieve the specific performance required. This engineering is conducted within constraints associated with the type of mining method, available backfill material resource, backfill functional requirement, and imposed cost criteria.

The geometry of the mining system has a substantial impact on the type of backfilling system that can be employed. Mining methods can be subdivided into two general categories according to whether the excavations have significant vertical extent. Methods that address ore zones with large vertical extent allow much greater feasibility in the backfill system. Emplacement schemes can take advantage of gravity dropping, which allows a relatively large volume to be filled from an isolated access. Methods that address thin ore zones with large horizontal extent or extract ore in slices with limited vertical height impose much greater restriction in the placement schemes. The repository layout is similar to a thin-seam mining method with very large horizontal extent.

Material resources available for backfilling are an important consideration. Most backfilling systems use mill waste by-products as fill material to reduce surface environmental impacts, cost of materials, and materials handling. Backfilling also provides an opportunity for in-mine disposal of mined waste rock. Backfill functional requirements may preclude use of mill tailings and require that fill materials be mined and processed to provide special-size gradations. In this case, fill materials may be mined on the surface or in adjacent, non-ore underground workings. Tuff waste, mined during repository construction, would form a fill material similar to waste rock used in filling some underground mines.

Backfill functional requirements have a primary impact on the type of backfill system employed. Some mining methods use backfill as the primary support of overburden during pillar extraction. These fills generally have requirements on the strength and deformability of the backfill that eliminate mill tailings as a potential material source. Materials may be well-graded coarse and fine aggregates with cement contents between 5 and 10%. In other cases, the fill must stand intact in exposed vertical walls as high as 600 ft (180 m) during pillar recovery. These fills must only be self-supporting and can be mixtures of portions of the mill tailings and cement. The primary backfill function may simply be to reduce the extent of open void, thereby minimizing the potential for surface subsidence or propagation of rock disturbance for long distances into the overburden. In this case, unconsolidated fills are adequate and can range from waste rock to the sand fraction of mill tailings.

In all fill systems, integration of the backfilling operation with the production (excavation) operation is critical to success. Materials handling underground is doubled. Backfilling systems that use mill tailings are strongly tied to the mill operating schedule. It is common for the backfilling system to have significantly higher capacity than the nominal mining rates.

Backfill costs are a major constraint in underground mining and dictate the type of approach applied. Costs have to be carefully weighted against the functional benefits. Backfilling costs can account for up to one-third of the total cost per ton of rock mined.

2.2 Fill Materials and Materials Handling

2.2.1 Introduction

This section discusses backfill materials in two general categories: uncemented and cemented fills. Within these two general categories, each type of fill can be further grouped into sand or rock fills. Uncemented fills are generally used where fill is not required to be self-supporting or to carry overburden loads. The primary purpose of the fill is to reduce the unfilled void volume to limit deleterious movement of rock. Specifically, the fill must prevent the vertical propagation of collapsed strata into other zones of ore, areas of mine facilities, or overlying water-bearing strata. Cementing agents are added to fill when there are requirements for strength and stiffness. This may be required for fill that will be freestanding and self-supporting during ore pillar mining. In other applications, cemented fill may be required to support overburden loads as pillars are removed or to form the mine roof in mining methods that proceed underhand (sequentially downward).

Fill materials may include both cement and aggregates, and the cement fraction is an important design consideration. Other material characteristics used to design backfill are:

- particle-size gradation,
- average particle size,
- proportion of fine materials,
- moisture content,
- mineral content, and
- abrasivity.

2.2.2 Uncemented Fill

Uncemented fills used in mine backfilling can be classified into rock and sand fills on the basis of particle size. Figure 2-1 illustrates these general categories and shows particle-size gradations for the range of fills used at the Mount Isa Mine in Australia. Uncemented rock fills are similar to the tuff waste rock being considered to backfill repository openings. In

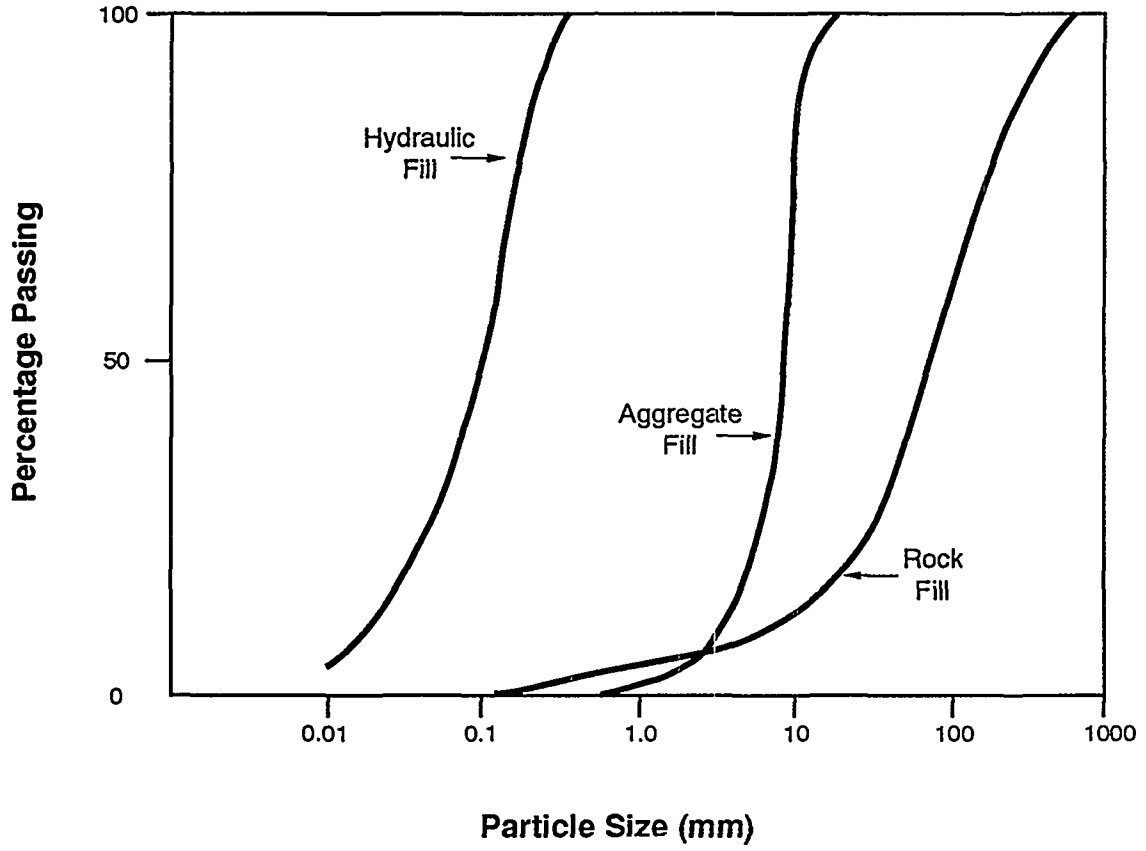


Figure 2-1
Typical Fill-Size Distributions Used in Backfilling Operations at Mount Isa (after Grice, 1989)

mining operations, this fill can be composed of development muck, waste rock, smelter slag, mined aggregate, overburden, and heavy media rejects or mixtures. Primary selection criteria generally include availability and emplacement cost.

Uncemented rock fill is generally a mixture of coarse and fine particles placed in a relatively dry state. Water content is typically less than 5% to assure free flow during transport and placement and to prevent material from sticking to surfaces. Because it is nearly cohesionless, it is generally used where walls of the excavation will not subsequently be removed, so the fill material will be permanently contained. Otherwise, removal of the containing walls (for example, by pillar recovery) may allow the rock fill to flow into adjacent excavations, interfering with operations and causing dilution of the ore grades.

Particle-size gradation must be controlled if the material is to be transported by gravity dropping in boreholes. It is likely that fine materials content will be limited to prevent plugging of the boreholes. Placement of the fill in a dry state can also cause excess dust in

the ventilation air and may be a reason to control fine content. Water inflows can result in fill liquefaction, especially in fills with an appreciable clay fraction, allowing fill materials to flow out of small openings.

Sand fill is generally made from mill tailings, the waste product remaining after the ore is crushed and ground and the economic minerals are extracted. The crushing and grinding process typically produces a large percentage of fine material, which is undesirable in fill because it reduces the settling rate and permeability of the fill, thereby inhibiting drainage. Tailings are, therefore, usually classified or deslimed by removing fine material in a hydrocyclone classifier prior to placement as fill. Material finer than 325 mesh is commonly removed during classification. Although imported natural sand is sometimes placed as fill; deslimed tailings have the consistency of sand and are generally termed sand fill.

Uncemented sand fill has similar constraints to rock fill, but can be less expensive because it is typically transported and placed hydraulically at high water/solids ratios. Drainage considerations are therefore critical, and the material percolation rate (permeability) generally governs the particle-size gradation requirements.

2.2.3 Cemented Fill

Cementing agents are added to both rock and sand fills when applications require the fill to have specified strength. Cement contents typically range from 3 to 15% by weight, with portland cement being the most common additive. Portland cements are often supplemented or replaced by pozzolanic additives in an effort to reduce backfill cost. Pozzolans are siliceous or aluminous/siliceous materials that react in the presence of lime and water to produce a stable, cementitious product similar to those produced by the portland cements. They are typically available as waste by-products of coal-fired power plants (fly ash), cement plants (kiln dust), and smelters (slag). Ground smelter slag was added to hydraulic fill at the Mount Isa Mine, as reported by Thomas and Cowling (1978). Studies of the use of fly ash, kiln dust, and slag as partial replacements for portland cement for use with coal waste backfills were reported by Brechtel et al. (1985). The range of strengths typical for cemented backfills used in mining is listed in Table 2-1.

Cemented rock fills are used when high strength is required in the fill. A recent example of high-strength fill was reported by Brechtel et al. (1989), where backfill pillars were used to support overburden loads while high-grade gold ores in rock pillars were recovered (see Appendix A.1). Design strengths were reported to be 8.3 megapascals (MPa). In other

Table 2-1

Comparative Laboratory Properties of Cemented Hydraulic Fill and Cemented Rock Fill

Property	Cemented Hydraulic Fill Value[a]	Cemented Rock Fill Value[b]
In situ density (kg/m^3)[c]	2,100	2,100
Cohesion (MPa)[d]	0.29	6.7
Unconfined compressive strength (MPa)	0.95	10.2
Friction angle (degrees)	31	37
Elastic modulus (MPa)	700	6,200

[a]Dight and Cowling (1979)
[b]Berry (1981)
[c]kg/m^3 = kilograms per cubic meter
[d]MPa = megapascals

applications (Dight and Cowling, 1979), rock fill is added to sand fill derived from tailings to produce an intermediate-strength cemented fill. In many mines, up to 50% of the tailings material may have ultrafine particle sizes (slimes) that are adverse to strength and must be separated by hydrocyclone. Rock fill is added to the mixture to make up the volume deficit. Cemented rock fills produce strength with higher cement efficiency because of the strength of the coarse fraction. Brechtel et al. (1989) reported cemented rock fill with mean laboratory compressive strengths of 8.5 MPa when portland cement contents were between 5 and 6% and it had low water/cement ratios. Similar cemented rock fill-type fills may be used for construction of special seal structures in the repository.

Cemented sand fills with solid contents of 40 to 50% are typically transported and placed hydraulically. The high water content and low volume of cement relative to the surface area of the sand particles result in low strength. Drainage of the excess water after placement transports a portion of the cement out of the fill. If placed with low water content, cemented sand fills can have higher strengths.

Recently, sand fill systems that use the entire tailings fraction have been developed. These fills, called "paste" backfill, are hydraulically transported at higher solid densities (60 to 80%). The low water content results in better cement efficiency. Thibodeau (1989) reported that in situ strengths of 3.4 MPa were obtained with portland cement content of 6.4% for a

high-density "paste" fill. In other tests, Hunt (1989) reported that average strengths of 3.8 MPa were obtained at 28 days for a total tailings fill with 12.5% portland cement placed at 78% solid density.

2.2.4 Materials Handling

Materials handling systems vary depending on the backfill resource, type of backfill, and mining method. The most fundamental subdivision is based on the rock fill/sand fill distinction. Rock fills are generally transported to and into the mine by some form of mechanical or pneumatic system, and sand fills are usually transported hydraulically. Materials handling, distinct from emplacement, is discussed under these general subdivisions in the following sections.

2.2.4.1 Rock Fill

When the backfill resource is on the surface, rock fill is generally subjected to the following sequences of operations:

- crushing/screening,
- surface transport,
- gravity dropping into the mine, and
- underground transport.

A schematic of such a system used at Mount Isa is shown in Figure 2-2. Neindorf (1983) reported that siltstone mined in an open pit was transported to a crusher by haul truck. The crushed rock was screened and then transported by conveyor belt a distance of 2.5 kilometers (km) to holding pits. Rock fill passes (boreholes), bored to the mining levels from the base of these holding pits, choke-fed with chutes, and installed at the base of the passes control the discharge of the rock fill into the mines. On the mining levels, rock fill was transported horizontally to the emplacement points by conveyor. Cemented hydraulic fill was then concurrently placed in the stope to form the cemented rock fill.

A materials handling system for producing high-strength fill, described by Barrett et al. (1983), was based on the surface batch plant shown in Figure 2-3. Gravel was quarried at a pit 20 km away from the mine and transported by tractor trailer to a mine stockpile. The gravel was rehandled into the mixing plant, where it was combined with cement and water in a batch mixer. The backfill was then transported to 1.2-m-diameter vertical boreholes by

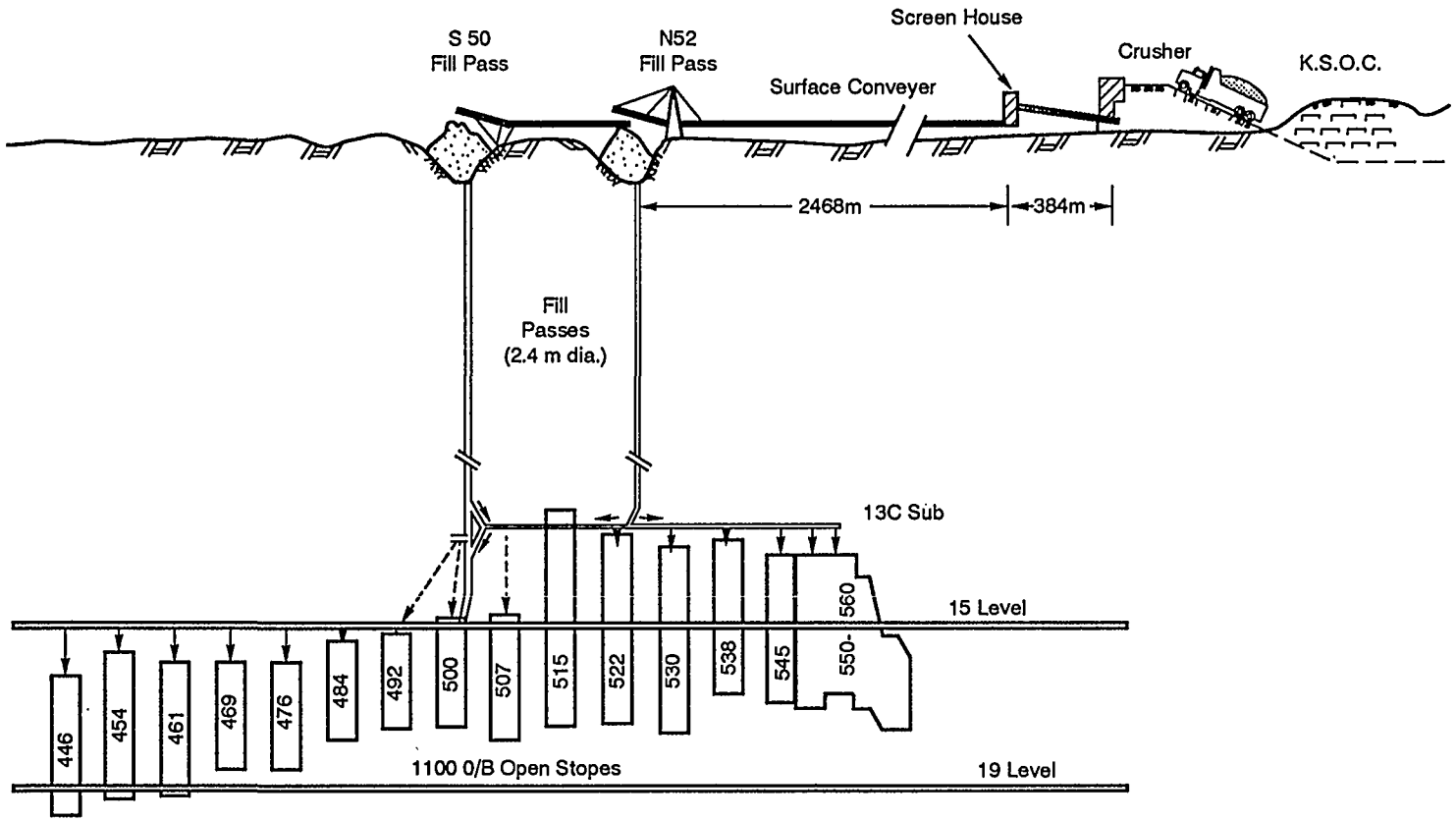


Figure 2-2

Surface and Underground Rock Fill Handling System at Mount Isa (Neindorf, 1983)

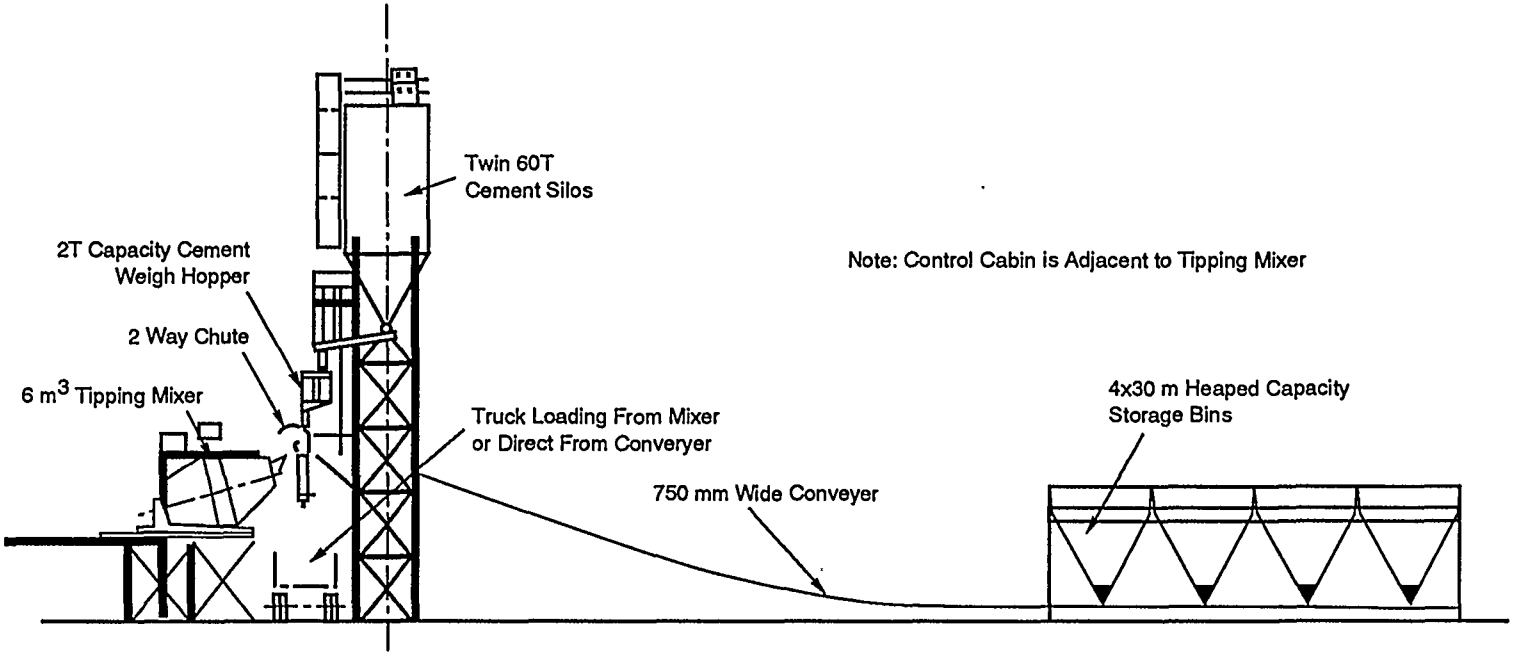


Figure 2-3

Schematic of Surface Cemented Rock Fill Batch Plant (Barrett et al., 1983)

truck and dropped directly into the stopes. High-strength fill used at the Cannon Mine was mixed in an underground batch plant and transported underground by trucks using telescoping beds. This operation is described in detail in Appendix A.1.

Transport of rock fill into mines is generally done by drop holes (boreholes). Voss (1983) reported that vertical dropping of soft coal waste caused excessive degradation of the fill particle size. Drop pipes were manufactured with internal spiral guides to reduce wear, reduce fill degradation, and develop smoother flow. Within the mine, rock fill was transported horizontally by trucks, conveyors, and rail cars.

Pneumatic transport of rock fill from the surface was reported at the Billie Mine in southern California by Garrett (1985). This system is discussed in detail in Appendix A.2. Fill was screened at a surface plant and then blown underground in a vertical pipe. Pipe connections were used to pull fill out of the vertical pipe at various levels and then place it in pipes that transported it horizontally. The pneumatic blower was surface-mounted.

2.2.4.2 Sand Fill

Sand fill materials handling is almost always associated with a mill tailings handling system. This type of system is exemplified by the system for hydraulic backfilling at the East Driefontein Mine in South Africa. Tailings are passed through hydrocyclones to remove the ultrafine fraction, then passed into surge tanks connected to vertical pipes that transport the fill into the mine. In this example (Pothas, 1988), the ultrafine slimes are routed to a secondary circuit where diluted flocculate is added to agglomerate the slimes into large particles, which are then recombined with the sand fill and sent underground.

Tailings may not supply the entire need for fill and must be supplemented with waste rock. Figure 2-4 (DeJongh and Morris, 1988) shows a schematic of a system that reduces waste rock to sand size for use as hydraulic fill, with porosities of from 25 to 40% and placement densities from 1,600 to 2,100 kilograms per cubic meter (kg/m^3).

Hydraulic fill is transported within mines through pipe networks. To select pumping requirements, the pipe network design considers gravity head from the surface, flow characteristics, and distance to emplacement point.

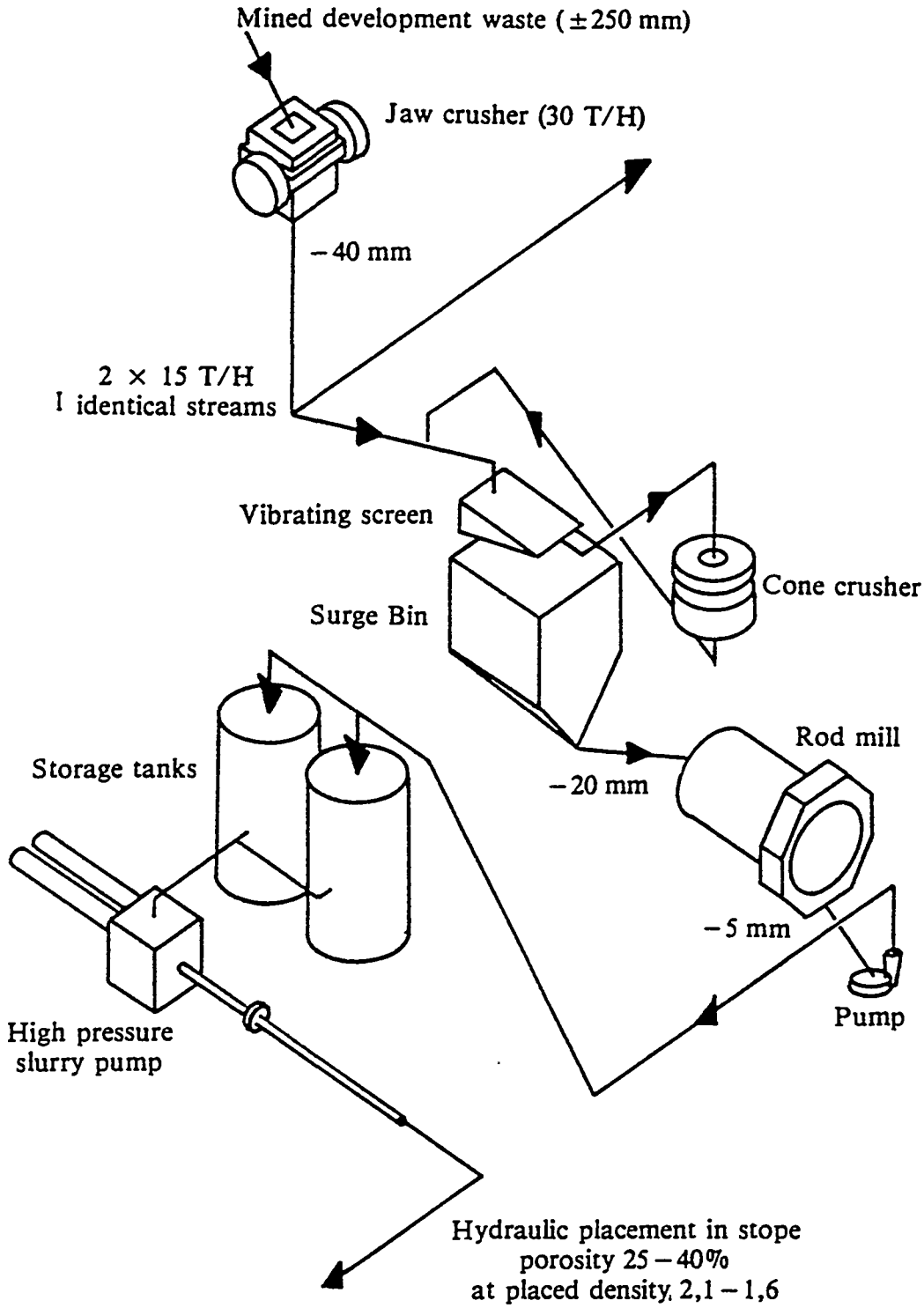


Figure 2-4

Flowsheet of Materials Handling System to Produce Sand Fill from Mined Waste Rock (from DeJongh and Morris, 1988)

2.3 Emplacement Methods

This section describes the various methods of emplacement, which is the action of placing the backfill locally in the mined excavation. The primary emplacement methods used in underground mining are mechanical, pneumatic, and hydraulic. Selection of the appropriate scheme is governed by the vertical extent of the excavation, access to the excavation, and desired proximity of the fill to the roof of the excavation. The different emplacement schemes used in underground mining are described under the general categories of excavations with both limited and large vertical extent.

2.3.1 Mining Systems with Limited Vertical Extent

Backfill placement in excavations with limited vertical extent (for example, coal seams) is energy-intensive and requires both transport to the immediate location of the fill face and spreading at the face. This type of emplacement scheme can also occur where excavations have large vertical extent, and rock fill must be placed tightly against the roof. Emplacement methods employed are based on the full spectrum of options: mechanical, slinger belt, pneumatic, manual, and hydraulic systems.

2.3.1.1 Mechanical Emplacement

Mechanical methods of fill emplacement offer great flexibility because they are usually based on mobile, rubber-tired equipment. Any type of fill can be mechanically placed in underground rooms of nearly any shape and size. However, mechanical emplacement can be costly and labor-intensive, so it is generally employed to place fills that cannot be hydraulically placed (i.e., coarse rock fills) or in situations where supplying and handling large quantities of water underground, associated with hydraulic fill, presents difficulty.

Truck and LHD—A flexible method of delivering backfill to the face involves using rubber-tired equipment such as trucks, load-haul-dump (LHD) units, or front-end loaders. While this equipment can be used to handle nearly any type of fill material, it is typically employed with cemented or uncemented rock fills that are too coarse to be handled otherwise.

At the Cannon Mine, very stringent requirements exist for fill strength and stiffness to allow safe total extraction of a small, high-grade gold orebody in the ground of varying quality, without surface subsidence (see Appendix A.1). At this mine, an underground pug mill is used to mix batches of high-strength cemented rock fill that is transported to the stopes via rubber-tired trucks. At the top of the ore zone, this fill is emplaced in 15-ft-high drifts by

dumping the fill at the face and then using an LHD unit (Baz-Dresch, 1987) outfitted with a special push plate attached to the bucket. In secondary stopes where high-strength fill is not required, only coarse aggregate is hauled to the stopes in trucks. Tight packing produces a fairly uniform void at the roof that varies between 2 to 6 inches (in.).

Slinger Belt—The truck-mounted belt slinger (shown in Figure 2-5) is designed to throw granular materials and has been successfully used at the Meggan Iron Mine in Germany (Rohlfing, 1983) for backfilling. It consists of a feeder/proportioning unit that feeds fill material onto a high-speed slinger belt. Belt speeds of approximately 60 feet per second (ft/sec) can throw the material up to 25 ft high and 50 ft horizontally. Attempts to use a skid-mounted slinger belt system at the Cannon Mine were reported to be unsuccessful due to its lack of mobility.

Slingers work well with materials of less than 2-in. particle size; wet materials with high fines content do not generally sling well. The force of the impact causes some compaction of the fill and provides good strength characteristics. The Meggan Iron Mine in Germany (Rohlfing, 1983) is an iron mine that covers a considerable extent and uses several different mining methods, including sublevel shrinkage stoping, room-and-pillar, and crosscut stoping. Due to the extent of the mine, need for flexibility, and difficulties with draining water, dry fill was selected. Both slingers and pneumatic stowing systems were considered, with the slingers selected for economic reasons. Truck-mounted slinger belts are also used for tight filling headings at the Kerretti Mine in Finland.

2.3.1.2 Pneumatic Emplacement

Pneumatic backfilling or stowing involves transporting and placing fill via pressurized pipes. The technology was originally developed in the European coal mines in the early 1930s for stowing in longwall panels and has since found application in both overhand cut-and-fill and filling of abandoned coal mines. Pneumatic stowing equipment is commercially available in the United States. It consists of a rotary positive-displacement blower that provides a source of pressurized air, a rotary air-lock feeder (RALF) permitting introduction of fill into the pipeline, and a pipe and nozzle system as shown in Figure 2-6. The fill is transported in a high-velocity airstream and blown forward to the face. Pneumatic stowing systems have been successfully used to transport and place uncemented rock fill at the Billie Mine (described in Appendix A.2) and to place uncemented rock fill on the cave side of longwall shield systems in German coal mines (Voss, 1983). Voss (1983) reports that the pneumatic stower equipment is located near the face and that backfill is transported to the stower by

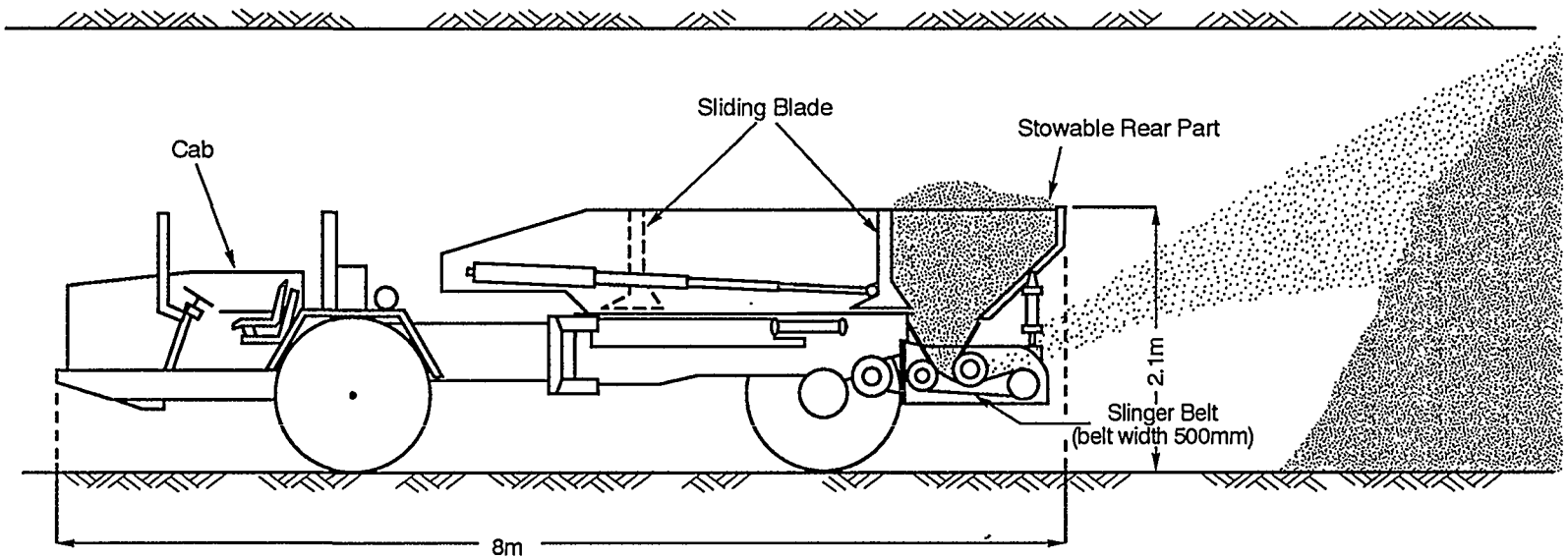


Figure 2-5

Truck-Mounted Belt Slinger Used to Fill the Meggan Mine (after Rohlfing, 1983)

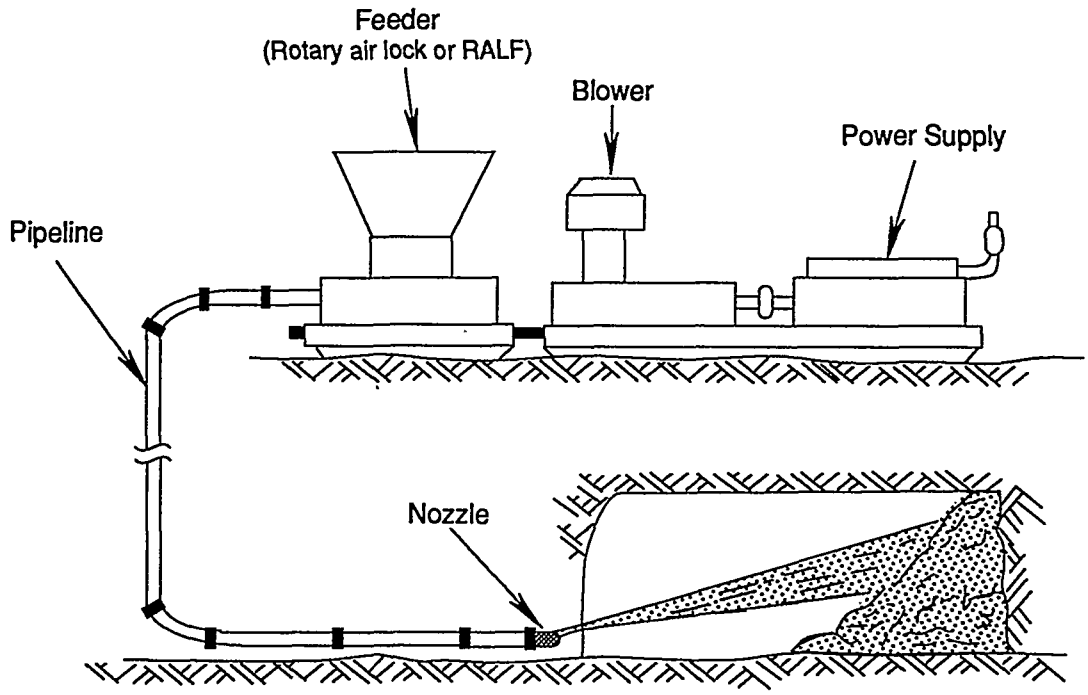


Figure 2-6
Typical Equipment for Pneumatic Stowing (after Djahanguiri and Mahtab, 1983)

train or conveyor. Adam et al. (1986) report the use of a pneumatic stower to place uncemented rock fill close to the roof to construct a muck pile ventilation slab. A schematic of the seal is shown in Figure 2-7. Backfill was transported to the location by front-end loader and was spread using a bulldozer to within 6 ft of the 30-ft-high roof. Roadheader cuttings were screened to minus 3 in. and placed to close the 6-ft gap. Settlement of the rock fill produced an appropriate 1-in. gap at the roof.

Cemented rock fill has been pneumatically placed in small excavations at the Apex (Appendix A.3), the American Girl Mine (Appendix A.7), and Mina Maria (Reuss and Olivier, 1992) mines. In these examples, fill was pneumatically transported from the surface to the placement location. Reuss and Olivier (1992) report fill placement rates between 30 to 50 tons per hour.

The most severe problems encountered in pneumatic emplacement are pipe wear and RALF wear. These are especially severe problems for hard abrasive fill materials, especially those confining silicate minerals. Voss (1983) reports pipe life of 300,000 to 400,000 m^3 of stowed material for coal waste materials. Other problems with pneumatic filling in small

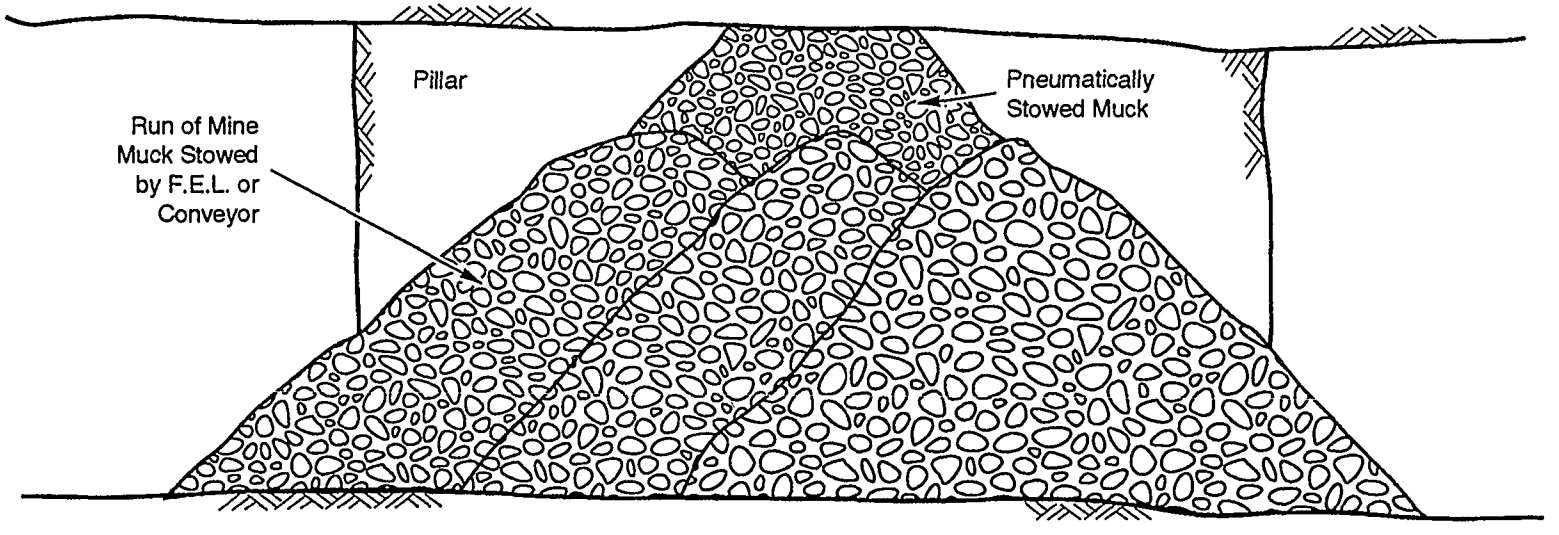


Figure 2-7

Construction Drawing of the Pneumatically Stowed Muck Pile Stopping (after Adam et al., 1986)

excavations are dust and noise because of the limited working space. Some degradation of fill materials also results, depending on the strength of the transported fill particles. The fill pipes take up space underground, sometimes interfering with vehicular traffic. The filling process lacks the flexibility of rubber-tired mechanical fill systems and can only transport fill a limited horizontal distance.

For these reasons, pneumatic stowing is seldom used in underground mining today, and the Billie, American Girl, and Mina Maria mines have reverted to mechanical methods. However, where remote stowing is required, such as in blind backfilling of abandoned mines, pneumatic methods may offer the best solution.

2.3.1.3 Manual Emplacement

Manual handling of backfill has been used for many years to place fill in all types of underground mines. All types of fill can be placed using wheelbarrows, shovels, and similar equipment. Bagged fill can be manually placed as packs; this practice is still used in European coal mines. Except for very localized fill situations, manual methods have been largely replaced by other methods in modern mines, but are still employed in mining regions with inexpensive labor.

2.3.1.4 Hydraulic Emplacement

Hydraulic filling of small excavations is practiced extensively in the gold mines of South Africa. Generally, these mines extract gold ores from thin, tabular (1 to 2 m) orebodies of great extent. These seams typically dip in the range of 15 to 20 degrees and are mined by advancing the longwall downdip. Longwall panels are mined with drill-and-blast techniques using panels in the range of 16 by 40 m. The panels are barricaded with poles, and porous geotextile fabric is used to seal the panel before filling. The geotextiles are used to allow water to drain from the fill. Fill is generally required to be in contact with the roof over much of the filled area to reduce movement of the overburden, which results in large stress concentrations at the advancing face. This is easily achieved in dipping seams but is more difficult for flatter seams. Faure (1988) describes hydraulic backfilling with tight roof contact at the Harmony Mine in South Africa, where the seam has slight dips. Tight fill is achieved by careful staging of the fill from the downdip side by slowly withdrawing the fill pipes along the roof as roof contact is achieved locally. Pothas (1988) reports the use of premanufactured "sausage bags" of geotextile to reduce the width of the panel undercut before backfilling. Drilling of the face is conducted at the same time as backfill emplacement.

2.3.2 Mining Systems with Large Vertical Extent

Backfill emplacement in excavations with large vertical extent allows much greater flexibility, because backfill can be introduced from one centralized location, and gravity feed can be used. Methods include mechanical, gravity dropping, and hydraulic emplacement. Examples of each of these systems are described in the following sections.

2.3.2.1 Mechanical Emplacement

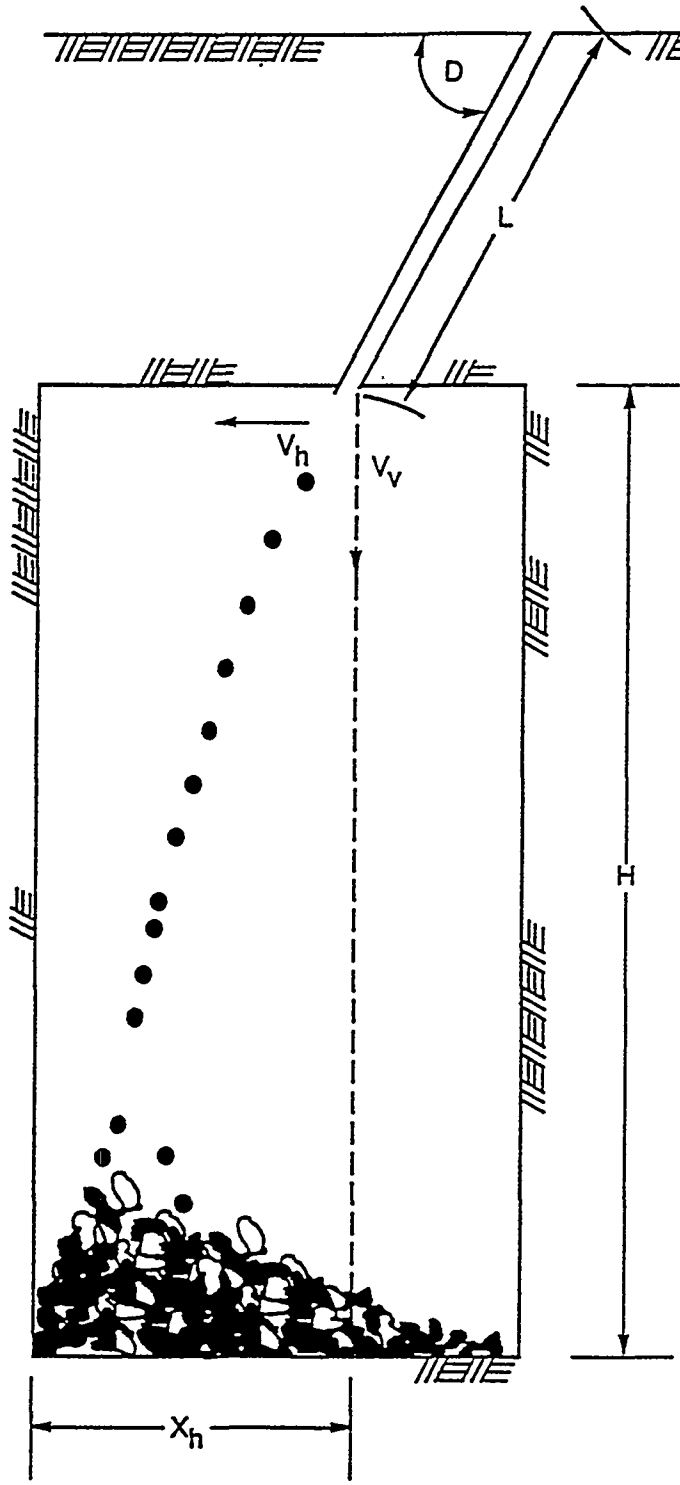
Backfill is hauled by rubber-tired equipment and dumped into large open stopes by trucks and LHDs. In the overhand cut-and-fill system used at the Cannon Mine (Appendix A.1), tall (up to 30 ft high) excavations are backfilled from a single access at the top of the stope. Trucks with telescoping beds dump the fill into the excavation and advance the fill pile to the end of the stope. The American Girl Mine (Appendix A.7) and the Bullfrog Mine (Appendix A.6) use similar equipment for emplacement and tightfilling.

2.3.2.2 Gravity Dropping

If the emplacement area geometry is appropriate, raises and chutes can be used to gravity-feed backfill. This section discusses only dropped dry fills or rock fills; most hydraulic fill systems also employ gravity, but are addressed separately in Section 2.3.2.3.

In metal mining practice, dropped fills are used for the open and shrinkage stoping methods, because the space to be filled has appreciable vertical extent and can be filled from a limited number of holes without secondary mechanical spreading within the stope. Rock fills and dry aggregates will form a conical pile under the fill entry point, with the dimensions limited by the angle of repose, typically 30 to 60 degrees. The delivery raise must be designed considering the effect of its inclination on the filling process as illustrated in Figure 2-8.

On the 1100 level of Mount Isa, as reported by McKinstry and Laukkanen (1989), uncemented rock fill and cemented hydraulic fill are independently transported to a fill pass at the top of each stope and run simultaneously through the pass to mix while falling, resulting in a cemented rock fill. The angle of repose of the rock fill is reduced; additionally, some segregation occurs with the hydraulic fill spreading to the outside of the stopes, as shown in Figure 2-9. This method allows for tighter filling and reduces the consumption of cemented fill, but can lead to loose areas on the exposed walls if irregular distribution of cement occurs.



LEGEND

$X_h = V_h \times T$

$V_h = V_v/\tan D$

$= (2g \times L \sin D \times K)^{0.5}/\tan D$

V_h and V_v are horizontal and vertical velocity components respectively when discharging fill into stope, m/sec

L = length of the fill raise, m

D = dip angle of the fill raise, degree

H = height of the free fall of aggregate in stope, m

Figure 2-8

Diagram Showing Effect of Fill Delivery Raise Inclination on Filling Process (after Yu, 1989)

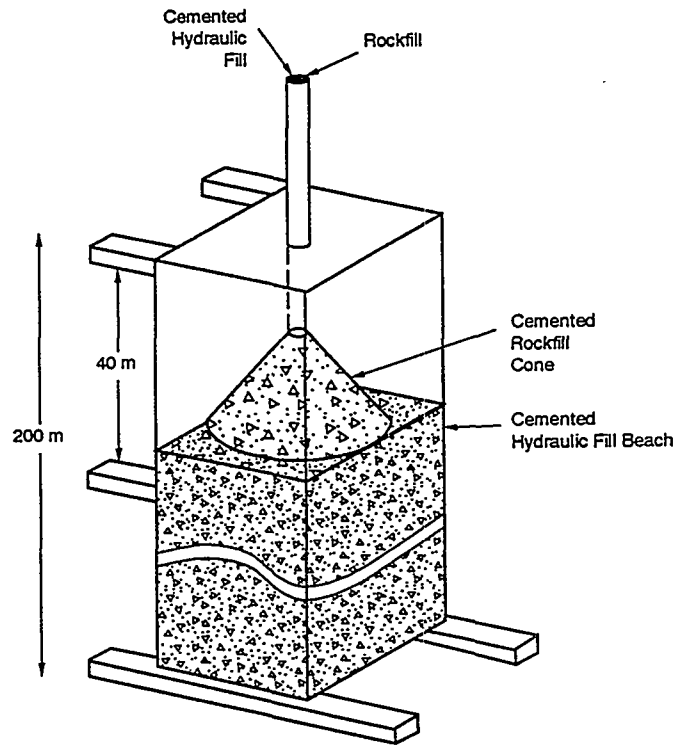


Figure 2-9
Idealized View of Gravity Filling Open Stope Using Both Cemented and Uncemented Fills (after Cowling et al., 1989)

2.3.2.3 Hydraulic Emplacement

Hydraulic filling using classified mine tailings or alluvial sand accounts for a large proportion of the backfill placed in underground mines. Hydraulic fill comprises 76% of all Canadian mine fill, according to a survey by Thomas (1979).

Figure 2-10 is a schematic of delivery, placement, and drainage of hydraulically placed tailings in a cut-and-fill stope. Prepared fill is fed by gravity down a pipe or borehole and transmitted by a system of pipes to the area to be filled. The fill is placed above existing fill in a space prepared by bulkheads. Provisions are made for drainage of water from the fill, which is typically 70% solids and 30% water. A system of ditches and boreholes leads to settling; clear water sumps beneath the stopes, where the water is collected, clarified, and pumped to the surface water treatment plant. This example demonstrates the level of design complexity that current operating mines routinely incorporate in their backfill systems, which involve fines removal, replacement of mined material, and complex drainage systems.

Dewatering of the fill is crucial and requires that the fill have adequate permeability. Settlement and consolidation cause excess water to pond on the surface of the newly placed

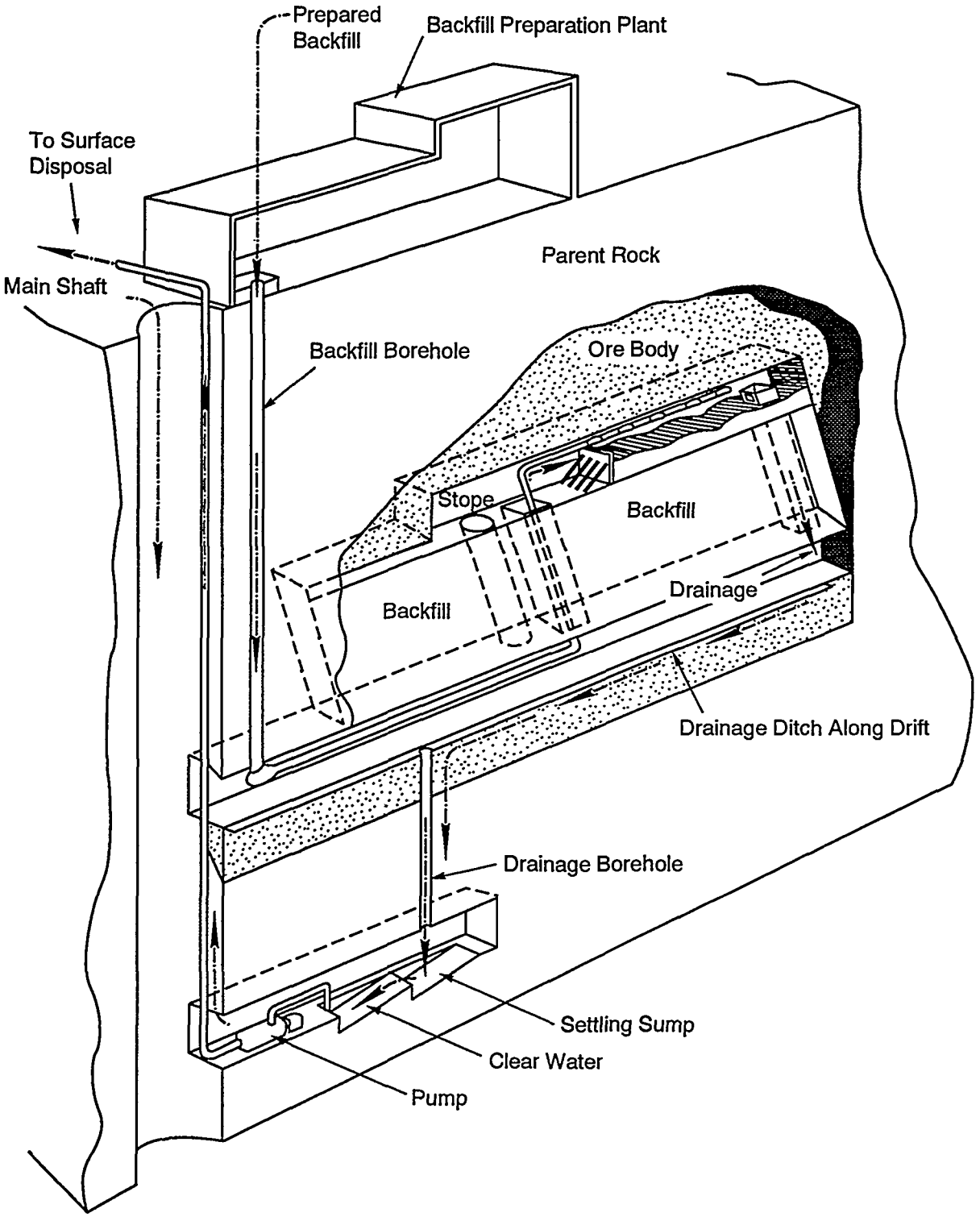


Figure 2-10

Schematic of Delivery, Placement, and Drainage of Hydraulic Fill in a Cut-and-Fill Stope (after Cummins and Given, 1979)

fill, where it must be decanted and removed through perforated pipes and raises that transact the stope. Water may also flow through the fill itself and be collected at the base of the stope. Drainage is inhibited by any fine materials or cements added to the fill and can be responsible for transporting much of the cement out of the fill after emplacement. A poorly drained fill is susceptible to liquefaction and catastrophic failure under loading conditions that would lead to high pore pressure.

2.4 Compaction Methods

Postemplacement treatment of backfill is not often practiced in underground mining because of restricted access and economic constraints. Also, more uncertainty in the characteristics of in situ backfill are acceptable during most mining operations. These types of treatments do occur in earthwork associated with civil engineering applications, and those with potential repository applications are discussed in the following section.

Special compaction of fills is common to earthwork associated with civil engineering applications, generally to limit settlement of surface structures. Compaction is the process of increasing soil density by mechanical action. The term compaction is sometimes applied to the entire operation of constructing an embankment, including spreading-mixing, wetting (or drying), and compacting. In mining situations, backfill is rarely subjected to special compaction after emplacement because of cost reasons and lack of access. The fill is usually well-confined by the surrounding rock, and closure of the room walls (if it occurs) will compact the fill until equilibrium is reached. Some compaction is associated with methods of emplacing the fill, especially dropping, slinging, and pneumatic stowing.

Four primary mechanisms are used to compact soil: static loading, impact, kneading, and vibration. A compactive effort must be applied to the soil. For a given soil type and compactive effort, there is a moisture content that results in the greatest degree of compaction, or "maximum density." This number can be measured in the laboratory; field compaction is specified as a percentage of maximum density obtained in the laboratory. Moisture content is less critical in granular fills than in clay fills.

Some of the compactors in common use today for civil applications include sheepsfoot, tamping foot, and vibratory compactors. Any of these compactors could be used underground, space permitting, although approval for underground use may be required from appropriate regulatory agencies. Descriptions of this equipment are available in any of several civil textbooks (e.g., Church, 1981).

In underground excavations to be tight-filled, there is no room for compaction equipment between the fill and the back in the final lift. Compaction, therefore, must occur on the angled face of a fill embankment.

Sheepsfoot rollers are used for compaction of clay, but are not useful for granular materials, such as alluvial fills, which can be compacted using rubber-tired rollers or vibratory compactors. Rubber-tired rollers may be light or up to 200 tons, depending on the depth of lift that must be compacted. Vibratory compactors come in a variety of sizes and may be towed, self-propelled, or hand-held. A typical hand-held, heavy-duty rammer would deliver approximately 600 blows per minute and can be used for small compaction jobs and those in close quarters. Regardless of the compaction method, frequent field tests should be made as compaction proceeds to ensure that specifications are being met.

In addition to premixing fill materials with portland cement and pozzolans, fills may be treated by grouting after placement to increase bearing capacity and reduce permeability.

2.5 Engineering Properties of Backfill

Engineering properties of interest in underground backfill are:

- particle-size gradation,
- percolation rate,
- density,
- moisture content,
- void volume,
- stiffness, and
- strength.

The relative importance of each parameter depends on backfill function, backfill type, transport methods, and emplacement. In the following sections, published data on these properties are presented to quantify the range of properties that may be applicable to repository design.

2.5.1 Particle-Size Gradation

Particle-size gradations are usually presented as the percent by weight of material passing a screen with certain mesh dimensions. Several other parameters are derived from the size gradation:

- D_{10} (effective size)—The particle size at which 10% of the material is fine and 90% of the material is coarse.

- D_{50} (mean particle size)—The particle size at which 50% of the material is fine and 50% of the material is coarse.

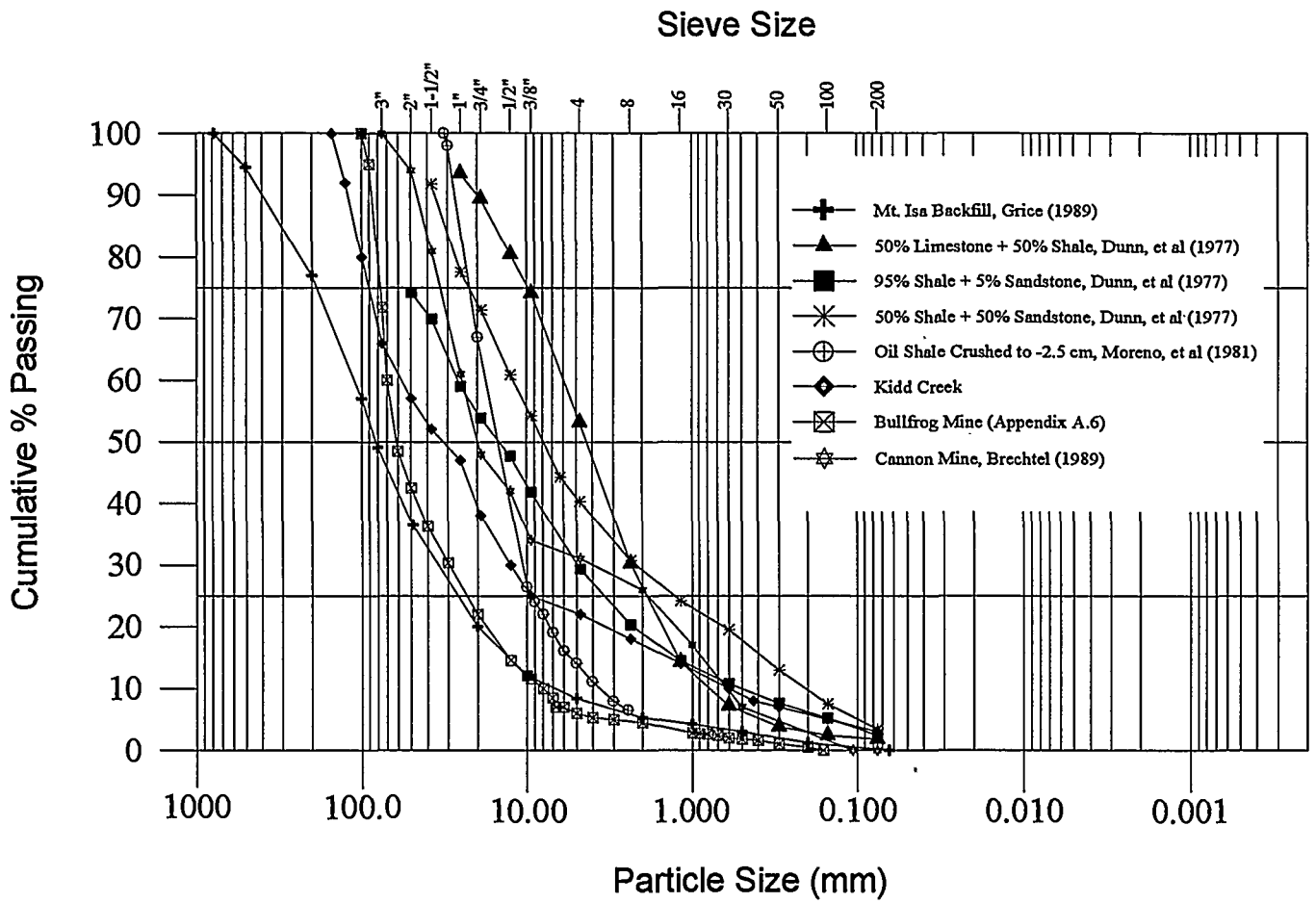
- Fineness modulus—A numerical index used to indicate the relative proportions of coarse to fine aggregate.

These parameters are useful in assuring that the character of the fill falls within acceptable limits for different applications. For example, the D_{10} parameter is a measure of the amount of ultrafine particles (slimes, clays) in the fill and is correlated with percolation rate. Fineness modulus is a measure of the grading of the fill material that is used to assure that good strength properties can be attained with the addition of cement. The American Society for Testing and Materials (ASTM) Standard C33 for concrete specifies that the aggregate have a fineness modulus between 2.13 and 3.37.

Particle sizes for different waste rock/aggregates described in backfill literature are compared in Figure 2-11. These range from the very coarse rock fill used at Mount Isa (Grice, 1989) to local quarry wastes being considered for filling abandoned coal mines in the Pittsburgh area (Dunn et al., 1977). Hydraulic fill particle sizes are in the 0.4- to 0.01-millimeter (mm) range. For these relatively coarse rock fills, the D_{10} parameter ranges from 6.0 to 0.2 mm, indicating relatively little ultrafine material. Hydraulic fills would typically have D_{10} parameters below 0.1 mm. The D_{50} parameters for the rock fills range from 60 to 4.5 mm, at least an order of magnitude above hydraulic fills that have D_{50} parameters in the range of 0.1 mm.

Sizing of materials is accomplished by crushing, screening, and washing to control the gradation. Figure 2-12 illustrates the change in size gradation of basalt caused by crushing to 12.7 mm versus 6.4 mm and later by washing each to remove the ultrafine material below 0.16 mm.

Crushing the material to the different sizes produces substantially different effective sizes (D_{10}). After washing the ultrafine materials from the mixture, the two gradations have nearly equivalent effective sizes.



Cobbles	Gravel		Sand			Silt or Clay
	Coarse	Fine	Coarse	Medium	Fine	

Figure 2-11

Particle-Size Gradations for Various Waste Rock and Gravels Considered for Backfills (Preliminary)

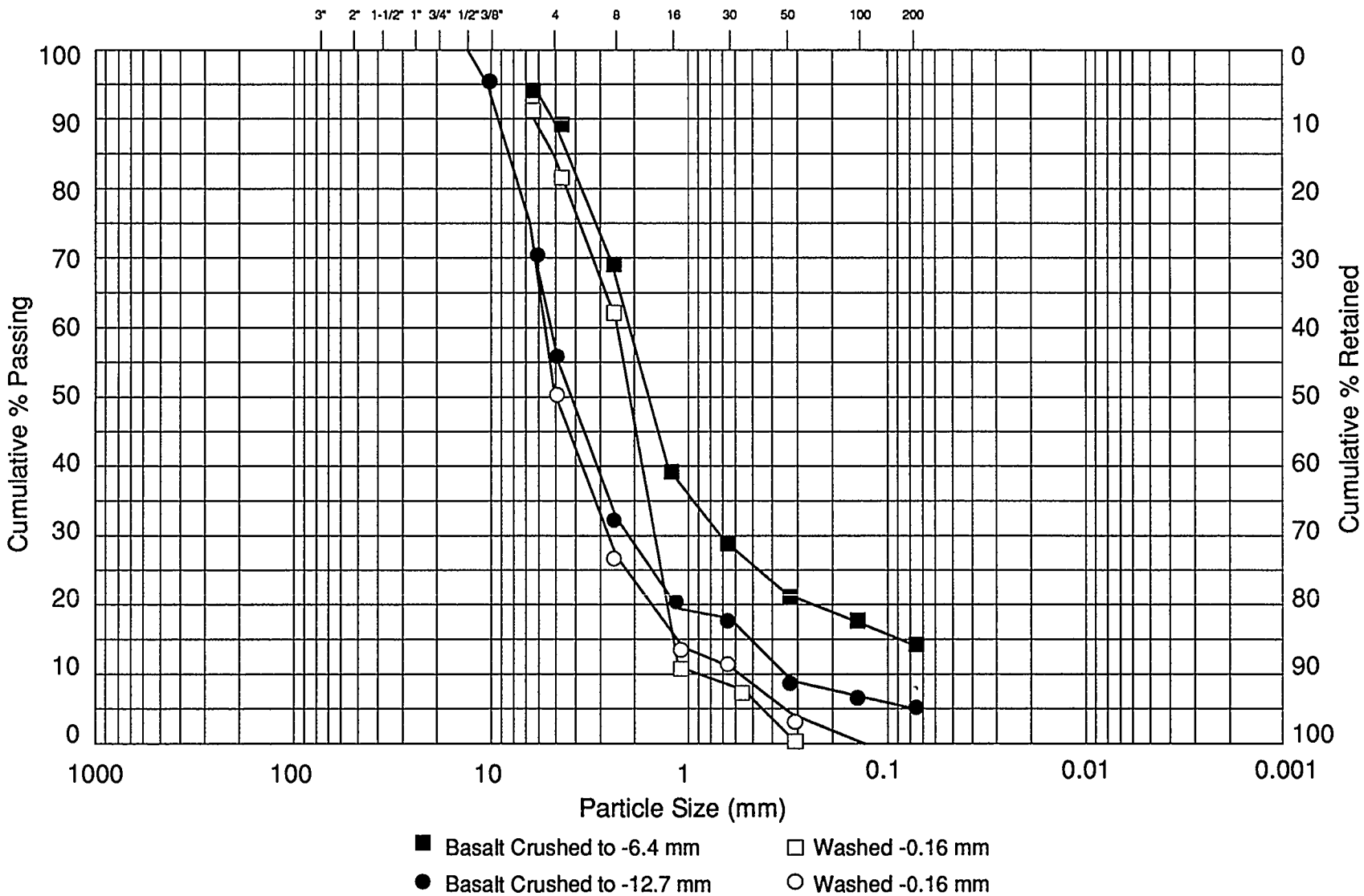


Figure 2-12

Variation in Particle-Size Gradation for Different Crushing Sizes and for Removal of Fine Materials

Changes in the size gradation can occur during transport and placement of the backfill. Measurements of rock fill degradation due to dropping fill down boreholes is reported for the Kidd Creek Mine by Yu (1989). A relationship was developed based on the observations shown in Figure 2-13 that relate the attrition ratio of mean particle size to the vertical travel distance.

$$\frac{D_{50} - Surface}{D_{50} - Depth} = 1 + (h/1100) \ , \qquad (2\text{-}1)$$

where h = vertical travel distance in meters. This particular rock fill is comprised of hard rhyolite and andesite rocks and undergoes roughly 9% reduction in mean particle size per 100 m of vertical travel in the boreholes.

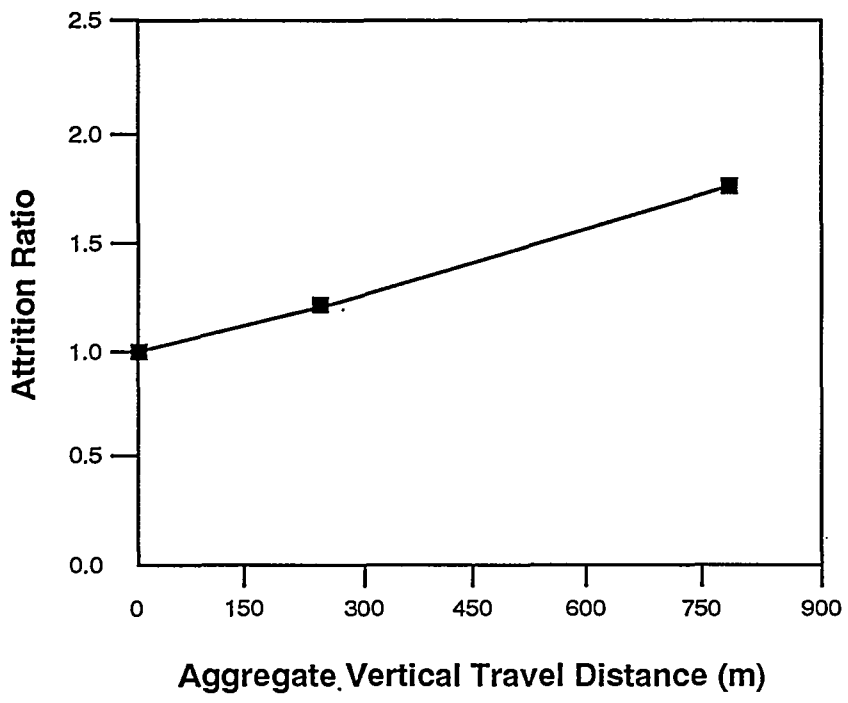


Figure 2-13
Attrition Rate of Rhyolite/Andesite Aggregate in Fill Raises at Kidd Creek
(Yu, 1989)

Degradation of particle size has also been reported by Masullo (1990) in crushed aggregate used in the aboveground testing of pneumatic stowing for coal mine filling. Segregation occurred during pneumatic stowing, with the coarser material being thrown farther than the fine material.

2.5.2 Percolation Rate

Percolation rate is a measure of the capability of the backfill to drain. Percolation rates are related to the size of void spaces through which the fluids must flow; however, since particle size is much easier to determine, various relationships have been proposed to relate percolation rate to particle size. Crude predictions can be made using the Hazan relationship (Hough, 1969, p. 75):

$$k = 100D_{10}^2 , \tag{2-2}$$

where k = coefficient of permeability [centimeters per second (cm/sec)] and D_{10} = effective size (cm) of grains in which 10% are finer.

Granular materials with a percolation rate of 0.10 cm/hr can be considered impervious to water.

Although the Hazan approximation was developed for graded filter sand, it has been found to apply to hydraulic fills based on mill tailings in experiments by Thomas and Cowling (1978). Figure 2-14 shows a scatter diagram of effective size versus minimum percolation rate for the experiments. Thomas (1978) concludes that fill permeability is controlled by the smallest particles present, with larger particles having less effect. Addition of portland cement greatly reduces fill permeability because of its small particle size. Application of the Hazan relationship to rock fills suggest percolation rates would range between 36 and 0.04 cm/sec or many orders of magnitude greater than the value where fill would be considered impervious to water.

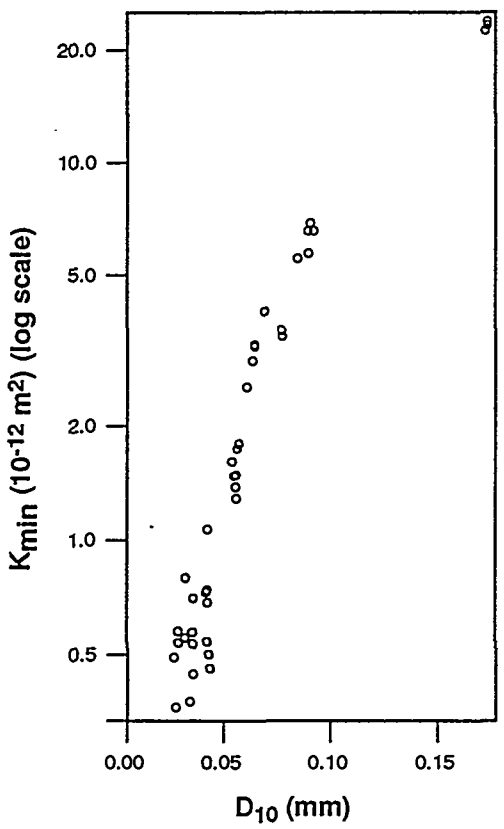


Figure 2-14
Plot of K_{min} Versus D_{10} (Thomas and Cowling, 1978)

2.5.3 Backfill Physical and Material Properties

2.5.3.1 Mining Applications

Backfill material properties are highly variable depending on the aggregate materials, moisture contents, and placement techniques. Standard tests to characterize properties are not generally available, and since backfills are custom-designed for each application, there is no standardization in reported results. Hydraulic fill studies generally concentrate on the flow and drainage properties, since handling of the fill is critical. High-strength fill studies concentrate on aggregate source, cement, and water contents. Laboratory properties are easily determined and therefore more available. In situ properties are relatively scarce in the published literature.

Stout and Friel (1982) present a series of experimental results on large-scale laboratory tests on backfill ranging from uncemented to cemented rock fills that illustrate the range of

properties of relatively low-strength backfill. The tests used four different backfill materials, two mill tailings, and two coarse aggregates. The materials are described as:

A. mostly quartz tailings (Comet-Gray Eagle Mine, Idaho),
B. river gravel, washed and crushed to -3.8 cm,
C. mill tailings from the Cooperative Mine, and
D. development waste rock from the Cooperative Mine argillites and quartzites.

Tests were performed on various combinations of the fill materials A-B and C-D with different cement contents and different simulations of placement in 1.22-m-long by 0.61-m-wide by 1.22-m-high forms. The results are listed in Table 2-2. The tests were performed as part of an investigation to produce backfill with stiffer properties and to examine the change in stiffness for different materials, cement ratios, and special treatments. Treatments included "double placing," which consists of filling the sample form with the coarse aggregate, then flooding the coarse voids with mill tailings from the bottom of the test form, and vibrating the fill within the form.

The stiffness of the fill is characterized by the shear stress-strain displacement curve in a penetration test with the penetration modulus chosen at a strain of 0.05. The stress-strain curves for the penetration measurement are typical of soils-type behavior with large plastic deformation.

Other work to characterize the stiffness of coarse rock fill was reported by Moreno et al. (1981) for the oil shale material shown in the particle-size curves in Figure 2-11. One-dimensional compression tests were conducted with lateral deformation presented by a steel enclosure. The results show a nonlinear hardening behavior typical of constrained backfill. An equation of the form

$$\varepsilon = E \left\{ 1 - e^{\left(\frac{P}{P_o}\right)^N} \right\} \tag{2-3}$$

was used to describe the stress-strain behavior, where ε = strain, E = initial void volume, P = applied stress, and P_0 and N = constants.

Table 2-2

Backfill Properties from Large-Scale Laboratory Testing (Stout and Friel, 1982)

Material	Placement Pressure Head (ft)	Cement Content in Tailings (%)	Special Treatment	Dry Density (kg/m^3)	Surface Balloon Density (kg/m^3)	Void Ratio	Porosity (%)	Penetration (Shear) Modulus (Mpa)
A	4	0	None	1475.3	1475.3	0.89	47	4.13
B	0	0	None	1295.9	——	1.14	53	5.52
A	4	0	Vibrated	1550.6	1741.1	0.82	45	10.21
A and B	4	0	Double placed	1920.6	——	0.45	31	23.45
A	4	10	None	1563.4	——	0.77	44	34.48
A	4	10	Vibrated	1637.1	——	0.69	41	91.03
A and B	4	10	Doubled placed and vibrated	2063.2	——	0.34	25	314.50
C	4	0	None	1565.2	1723.8	0.91	48	2.76
D	0	0	None	1693.5	——	0.93	39	2.76
C	4	0	Vibrated	1654.3	1584.4	0.82	45	9.38
C	4	5	Double placed	1629.2	——	0.83	45	12.41
C	4	5	None	1645.3	——	0.82	45	45.52
C and D	4	0	Vibrated	2020.1	1784.6	0.48	32	21.38
C	4	10	Vibrated and double placed	1645.1	1821.5	0.82	45	48.28
C	4	10	Vibrated and double placed	1956.0	1957.6	0.53	35	92.41
C and D	4	5		——	2053.8	0.52	34	30.34
C and D	4	10		1927.2	2356.7	0.55	35	58.62

Average values of these parameters for the oil shale material are listed in Table 2-3. This type of constrained compression test is performed to characterize the stress-strain curve of particulate materials while simulating the effect of constraint provided by the excavation.

Triaxial strength tests performed on crushed basalt with different particle sizes were reported by Al-Hussaini (1981) and are shown in Figure 2-15. The tests indicate increasing angles of internal friction with increasing maximum particle size between 6.3 and 76.2 mm. Higher density was also found to produce increases in the internal friction angle.

Table 2-3
Experimental Stress-Strain Parameter for Coarse Oil Shale (Moreno et al., 1981)

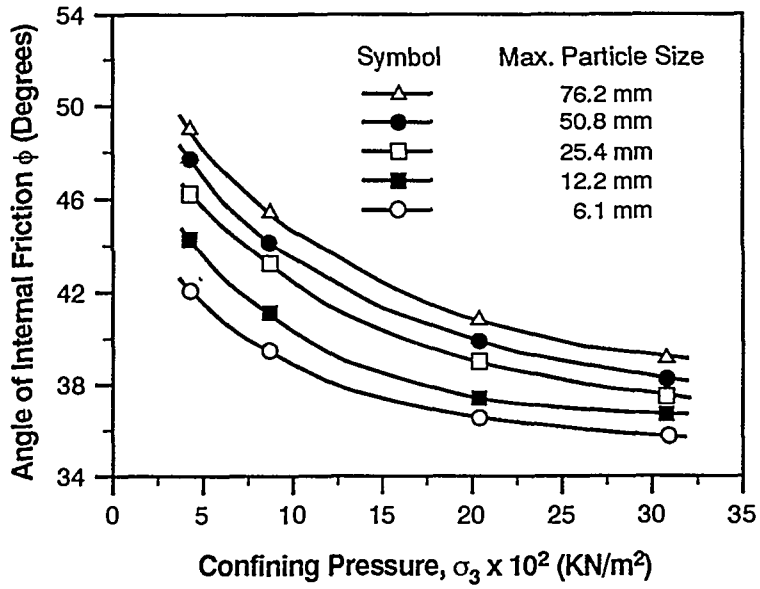
Height/Width Ratio	E_0	Average N	Average P_0 (Mpa)
1	0.52	0.27	17.58
0.8	0.53	0.34	11.9
0.6	0.51	0.35	14.42
0.4	0.5	0.39	13.96

2.5.3.2 Civil Applications

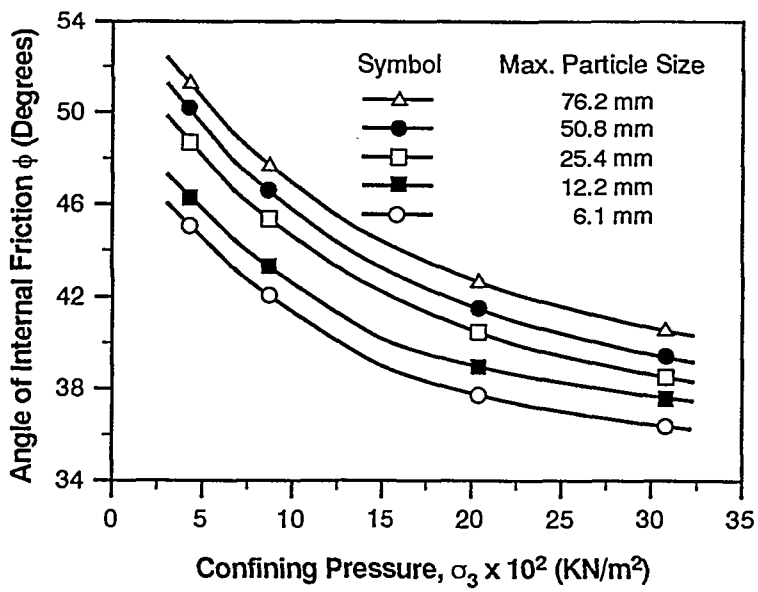
Shear strength and compressibility are the two most important factors in designing most civil rock fill structures (Marachi et al., 1972). Most investigations of rock fill consider variables, such as relative density, rock fill gradation, dominant size of rock particles, particle shape, degree of saturation, and confining pressure, that can affect these factors. Results of some of these investigations are presented below.

Through laboratory studies (Leps, 1970; Marachi et al., 1972; Marsal, 1973; Donaghe and Cohen, 1978; Charles and Watts, 1980; and Charles and Soares, 1984), fundamental property relationships were established for numerous rock types. These studies provide direction for some of the more important variables to be considered in evaluating rock strength. Some of the results are summarized below:

- In general, compacting rock fill increases shear strength and decreases compressibility and permeability.



a) Medium-Dense Crushed Basalt



b) Dense Crushed Basalt

Figure 2-15

Variation of ϕ with Confining Pressure for Medium-Dense and Dense Crushed Basalts (Al-Hussaini, 1981)

- The angle of internal friction provides a measure of shear strength. The angle of internal friction shows a decrease at a decreasing rate with increasing confining pressure. Shear strength increases with increasing coefficient of uniformity, increasing relative density, and increasing compressive strength of the intact rock.

- For large hard rock (similar to welded tuff), particle breakage occurs by splitting and breaking off edges and corners. Softer rock is reduced mainly by crushing.

- Clean, hard, angular, quarried basalt possesses higher strength than well-rounded gravel (Marsal, 1973).

- Saturation of the rock fill generally results in greater consolidation and greater strains.

- At very low confining pressures, dilation of rock fill occurs; at higher confining pressures, compression increases to some limiting value.

The results above show some of the relationships established in the past for rock fill. Additional investigations are needed to determine the properties of crushed tuff to support analyses evaluating the tuff response in the confined environment of the potential repository.

3.0 Available Technology for Bulkhead Installation

3.1 Underground Mining and Mined-Space Storage Applications

Bulkheads are used in underground mining to control airflow and water flow and to prevent human intrusion. There are a variety of specific applications resulting in different types of bulkheads. Major categories of mining bulkheads include: (1) inundation plugs, (2) ventilation stoppings, (3) hydraulic fill containment bulkheads, (4) abandonment bulkheads, and (5) consolidation bulkheads. Abandoned and sealed mines are sometimes reopened and converted to underground storage applications, particularly pressurized natural gas and petroleum products; such conversion requires improved sealing of openings using (6) "conversion" bulkheads.

Bulkheads used to completely seal underground openings are often called "plugs," particularly if they are constructed from concrete. Inundation plugs are also referred to as "dams" in some mining districts. In this section, the various types of bulkheads are reviewed, and examples of each type are presented.

3.1.1 Inundation Plugs

Inundation plugs may be installed in shafts, ramps, or drifts to protect a mine from inundation by sudden inflows of water, a potential hazard when mining below the water table in rocks such as carbonates that have discrete, highly conductive water channels.

Inundation plugs have been further categorized by Auld (1983a) as: (1) precautionary plugs, which are installed at strategic locations in the mine and contain watertight doors; (2) control plugs, to seal off abandoned portions of the mine that will no longer be pumped; and (3) emergency plugs, to seal off unexpected inrushes of water. Inundation plugs must often withstand very large pressures and are usually designed for full hydrostatic head to the water table. In special cases, open dam walls may be used to impound water without completely sealing the mine opening (Chekan, 1984).

Inundation plugs are important in the gold mining region of South Africa (Garrett and Campbell-Pitt, 1961; Cousens and Garrett, 1969). Figure 3-1 shows a precautionary plug installation at the Virginia Merriespruit Mine boundary. The 36-ft-long permanent plug contains a watertight steel door. A 12-ft-long temporary plug was installed to permit pressure testing of the permanent plug; it was removed before the plug was commissioned.

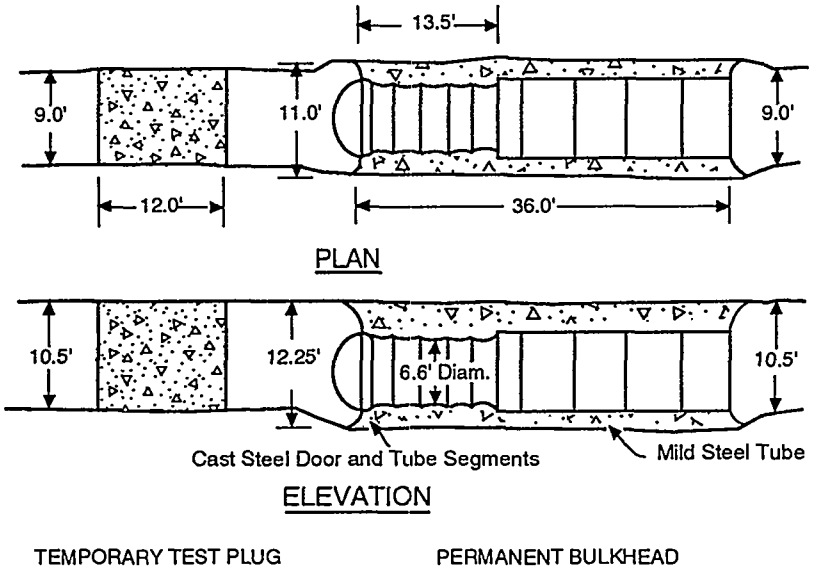


Figure 3-1

Precautionary Plug Installation, Virginia Merriespruit Boundary, South Africa (from Garrett and Campbell-Pitt, 1961, p. 1286)

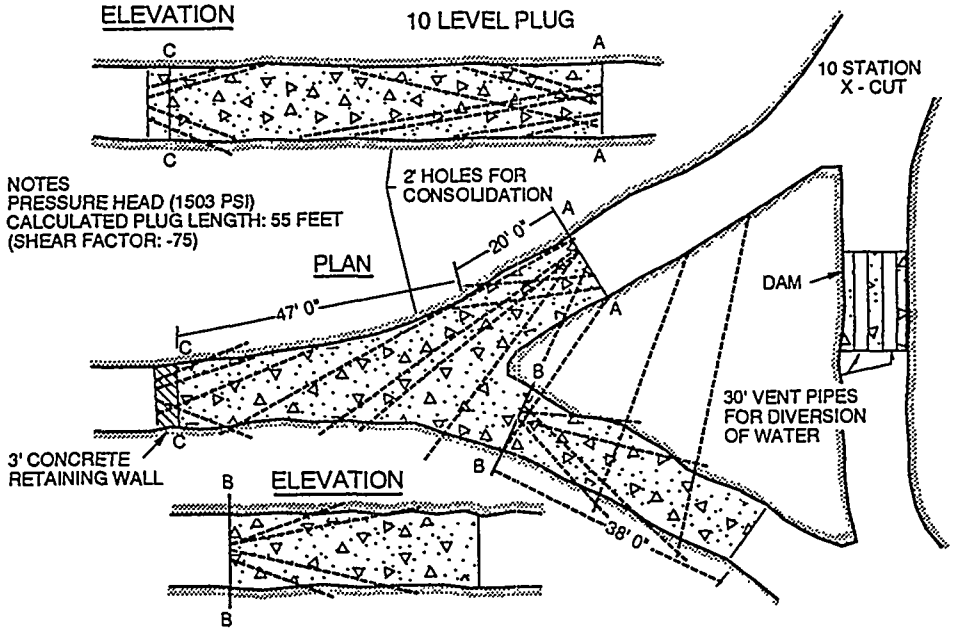


Figure 3-2

Emergency Plug Installation, West Driefontein Mine, South Africa (from Cousens and Garrett, 1969, p. 454)

In operation, the steel door would normally remain open. If a water inrush occurs, the miners can retreat behind the plug and seal the doors. Figure 3-2 shows an emergency plug installed after an unexpected inrush exceeded the pumping capacity of the West Driefontein Mine in the Witswatersrand. The plug was installed under adverse conditions, with water flowing 3 to 4 ft deep in the mine drifts during sealing operations. The installation was completed in time to prevent complete flooding of the mine (Cousens and Garrett, 1969).

3.1.2 Ventilation Stoppings

Ventilation stoppings are required in nearly all underground mines to control the flow of air. Both permanent and temporary stoppings are used and have been constructed of many different materials; among them, concrete blocks, wood, canvas, sheet metal, and muck (mined rock). Ventilation stoppings are designed for relatively low pressure differentials, rarely exceeding 1 psi. Low-cost construction, fire resistance, and minimum leakage are desirable attributes. Ventilation stoppings will be required during repository construction and operation; however, their purpose is to control ventilation airflow rather than to seal, therefore, they will not be discussed further.

3.1.3 Hydraulic Fill Bulkheads

Mines using hydraulic fill use bulkheads to retain the fill during drainage and consolidation. The bulkhead must impound the wet fill and permit it to drain until it has enough bearing capacity to support mining operations, typically several days. Figure 3-3 shows a typical application used at the Mount Isa Mine in Australia (Grice, 1989). Although these bulkheads are normally only designed for a limited pressure from the fill, several bulkhead failures at Mount Isa have been attributed to development of small sinkholes in the fill by erosion. These sinkholes became connected to water ponded on the fill surface, resulting in high hydrostatic pressures on the bulkheads.

3.1.4 Abandonment Bulkheads

Abandonment bulkheads are used for two purposes: (1) sealing off abandoned areas of active mines and (2) sealing abandoned mines. In the first case, bulkheads are installed to seal off abandoned areas of active mines to minimize pumping and ventilation requirements. These bulkheads must be designed to withstand hydrostatic pressure if they are used to seal water. If the mine is gassy, such abandonment bulkheads may be required to be explosion-proof (Code of Federal Regulations 30 CFR 329-2) to preclude the possibility of an explosion in the unventilated abandoned area propagating throughout the mine. The U.S. Bureau of Mines (USBM) has conducted considerable research on explosion-proof seals

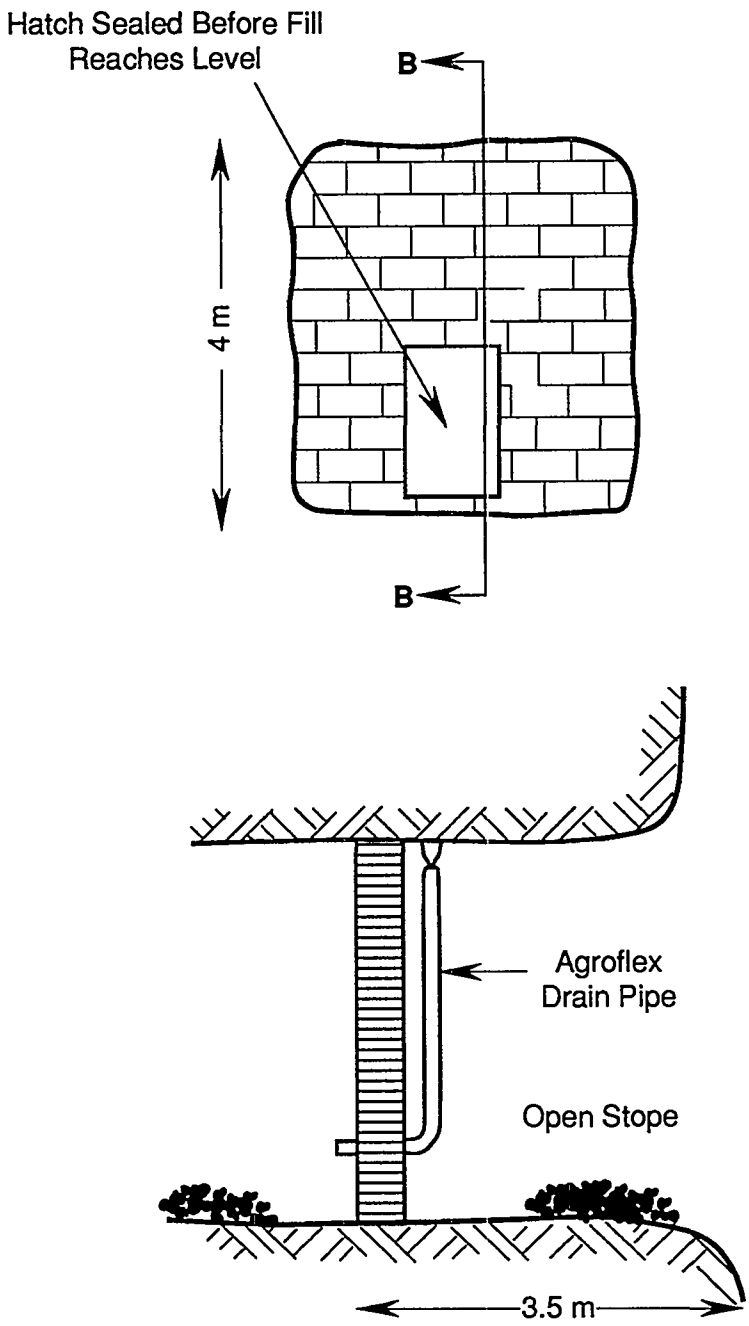


Figure 3-3
Concrete-Block Bulkhead Used at Mount Isa for Retaining Hydraulic Fill
(from Grice, 1989)

(Mitchell, 1971; Chekan et al., 1983; Foster-Miller Associates, Inc., 1974; Triebsch and Sapko, 1990; Stephan, 1990a and b). When the entire mine is being abandoned, bulkheads may be installed in shafts, access drifts, or portals to retain mine drainage water and prevent human intrusion (see Appendix C.3 *Eagle Mine case history*). Figure 3-4 shows typical abandonment plug concepts for coal mines (Chekan, 1984; Loy, 1974), while Figures 3-5 and 3-6 show concepts for sealing abandoned mine shafts (National Coal Board, 1982; Dressel and Volosin, 1985; Beck, 1992). Lutzens (1992) discusses low-cost foam concrete seals for abandoned mine shafts and adits.

3.1.5 Consolidation Bulkheads

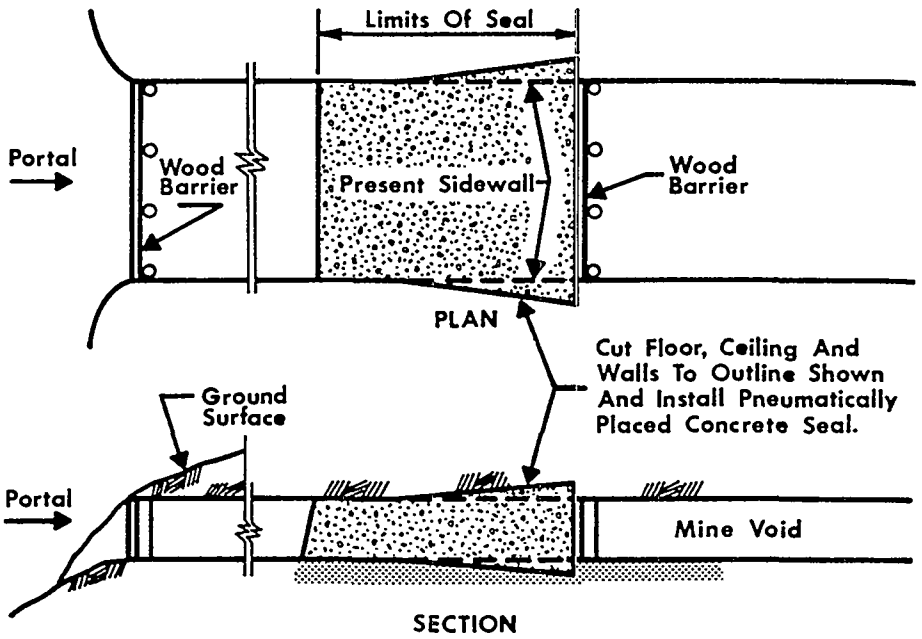
Consolidation bulkheads are commonly used with grouting operations during shaft sinking in water-bearing ground. Figure 3-7a shows a temporary consolidation bulkhead installed during construction of a coal mine shaft (Nash 1984). The plug was used to control water inflows and provide a platform for grouting a major water-bearing zone below the shaft bottom. After approximately two months of grouting from the plug, it was removed and sinking progressed through the grouted annulus. Figure 3-7b shows a temporary plug used to control groundwater inflows during drift development in South African mines. After the water-bearing fracture is grouted, the plug is removed by blasting and drifting continues.

3.1.6 Conversion Bulkheads

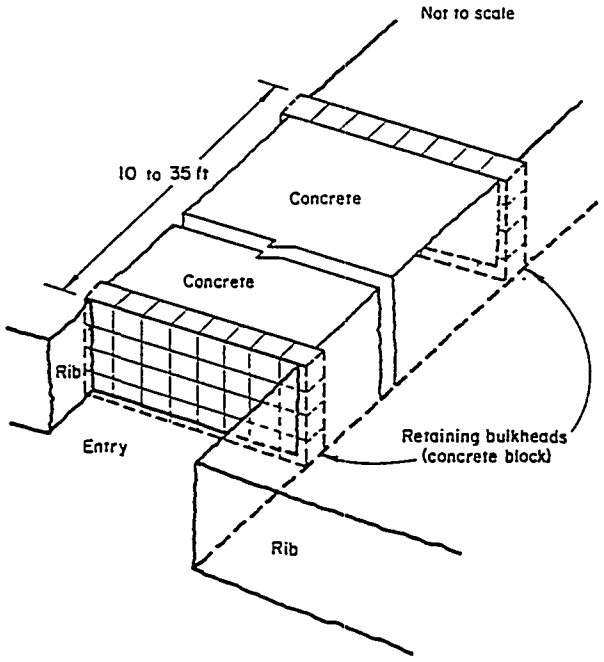
It is frequently cost-effective to use abandoned mine openings for underground storage, provided they can be sealed to minimize loss of the stored product by leakage. Sealing can be especially difficult when mines are converted to store pressurized gas products (Buttiens, 1977). Accordingly, conversion bulkheads are among the most complex to design and construct. Figure 3-8 shows a complex seal installed in a mine shaft during conversion of an abandoned potash mine for gas storage (Sitz and Forster, 1989). This seal relies on a pressurized, oil-filled annulus to prevent leakage and is reportedly very tight.

3.2 Civil Applications

Bulkheads are used in major civil applications for sealing temporary tunnels and dividing permanent tunnels into compartments. Two common civil applications are (1) dam diversion plugs and (2) hydropower tunnel plugs.

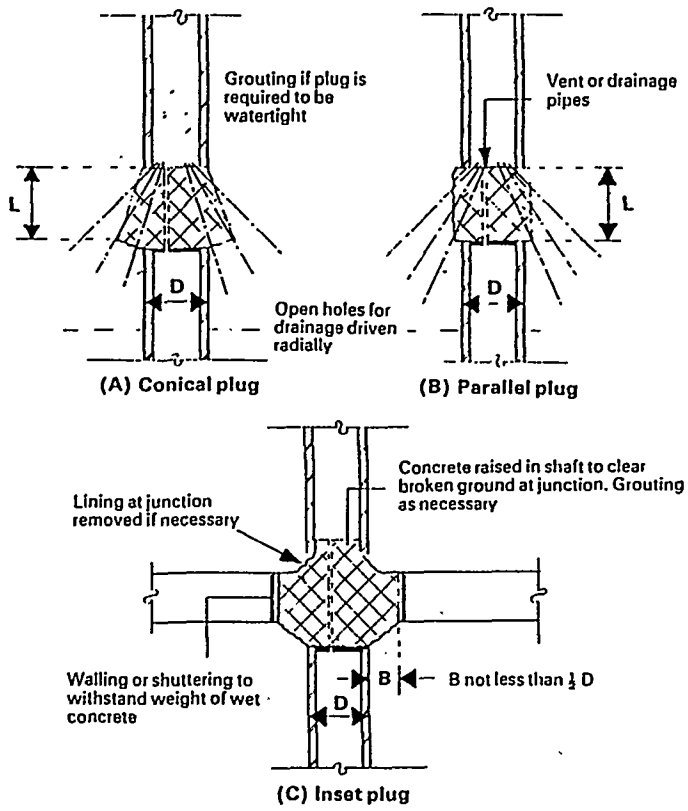


a) Pneumatically Placed Tapered Plug (from Loy, 1974, p. 158)

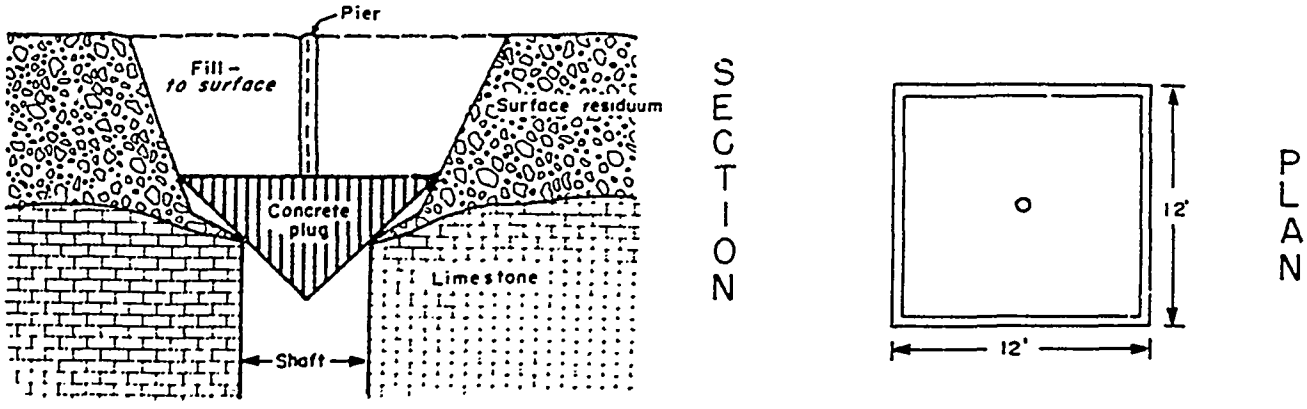


b) Double-Bulkhead Seal (from Chekan, 1984, p. 12)

Figure 3-4
Abandonment Plug Concepts for Underground Coal Mines



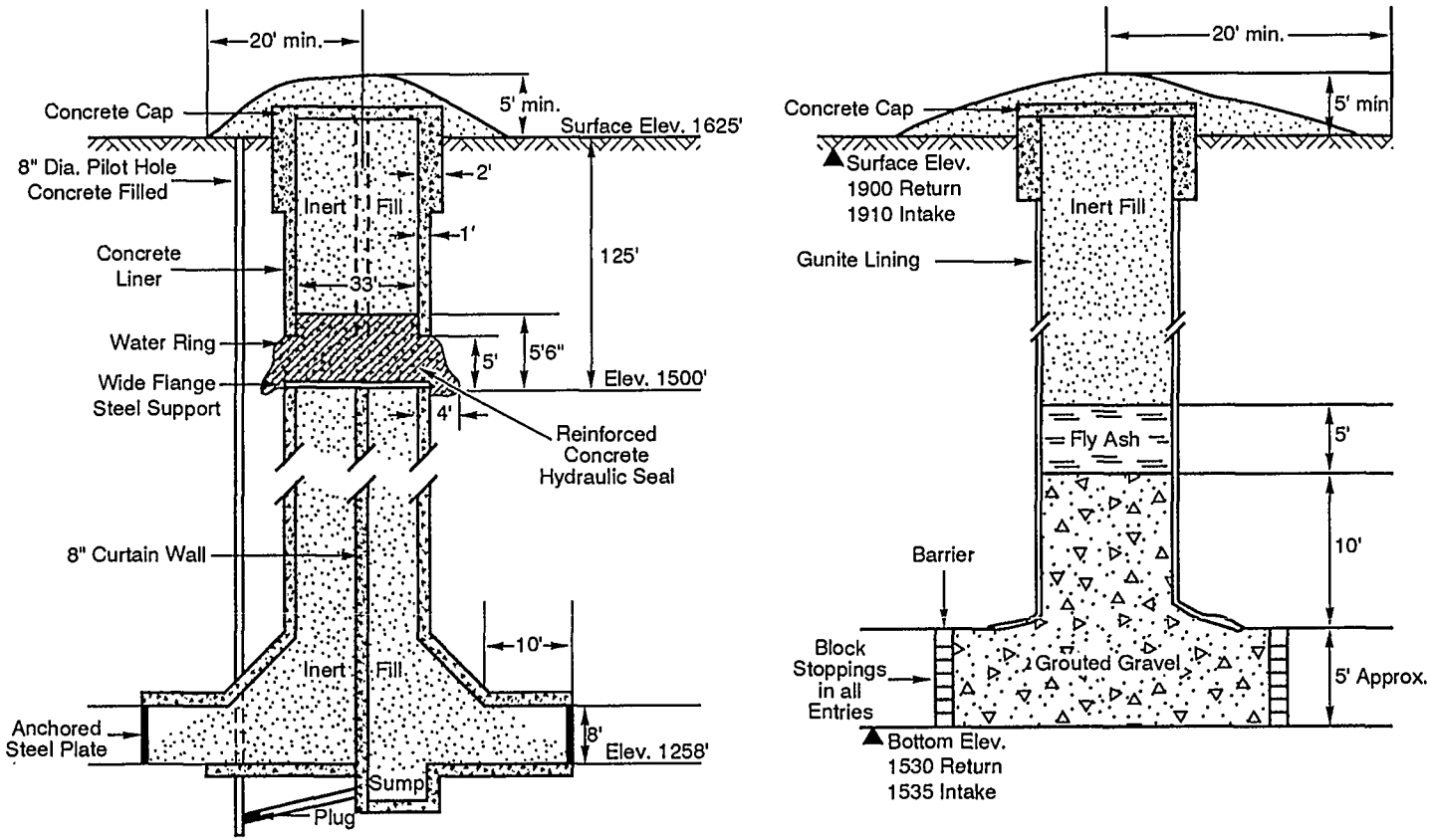
a) National Coal Board Recommended Practice (National Coal Board, 1982, p. 48)



b) Concept Tested by USBM for Sealing Shafts in Zinc-Lead Belt of Kansas (Dressel and Volosin, 1985, p. 6)

Figure 3-5

Abandonment Plug Concepts for Mine Shafts

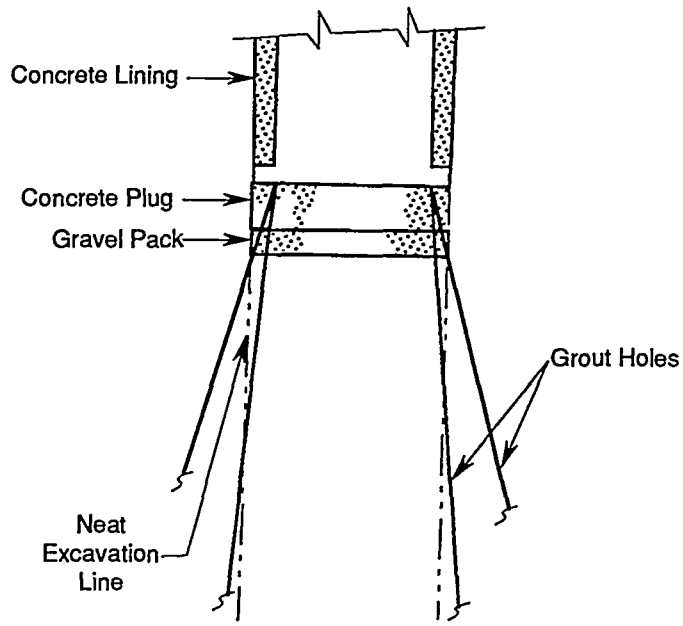


a) Hydraulic Seal Required to Resist Upward Pressure

b) No Hydraulic Seal Required

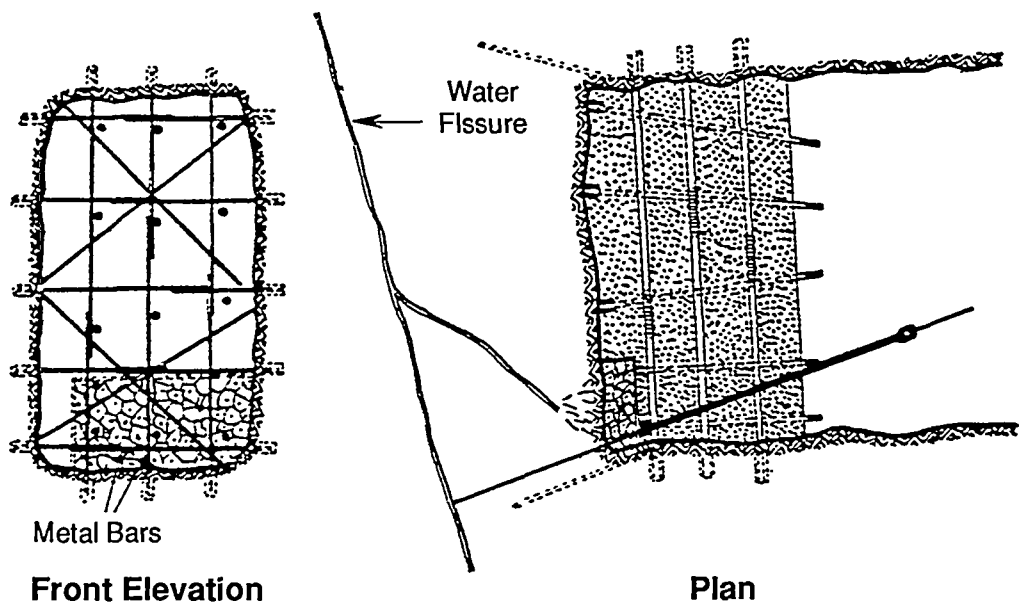
Figure 3-6

Seal Design for Two Coal Mine Shafts, Oneida Mine, PA (Beck, 1981, p. 38)



Typical Shaft Section

a) Temporary Plug for Grouting During Shaft Sinking for a Coal Mine (from Nash, 1984, p.14)



b) Temporary Plug for Grouting Water-Bearing Fracture Encountered During Drift Development, Underground Gold Mine in Witswatersrand, South Africa (from Jeppe, 1946, p. 748)

Figure 3-7

Temporary Consolidation Bulkheads Used to Grout Unexpected Water Inflows

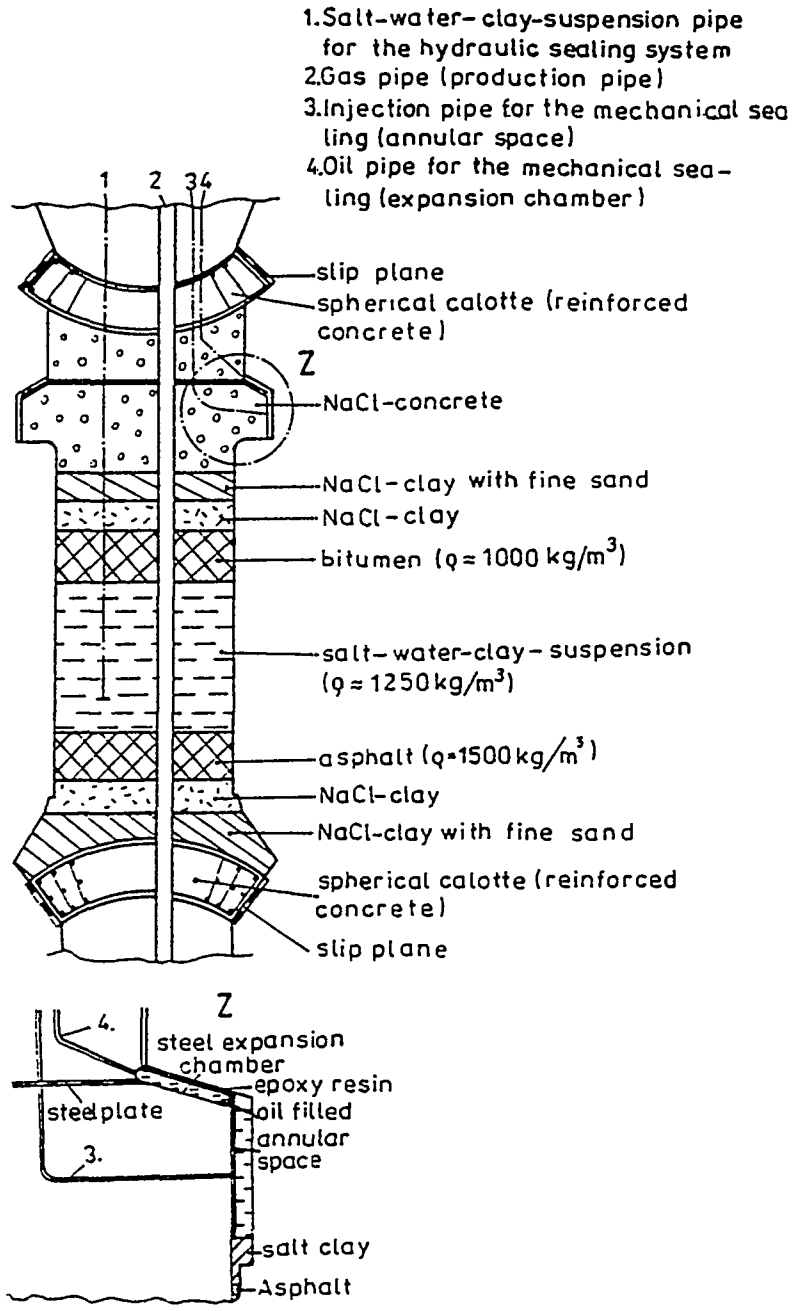


Figure 3-8

Complex Bituminous Shaft Seal Used to Convert Potash Mine for Natural Gas Storage (from Sitz and Forster, 1989, p. 199)

3.2.1 Dam Diversion Plugs

Tunnels are used to temporarily divert river flow around dam sites during the construction period. These tunnels must be sealed when the dam is completed. An example of such a diversion tunnel was used in the construction of the Foster Reservoir Dam in Oregon (Department of the Army, 1978). A tapered concrete plug was installed under the dam axis to permanently seal the tunnel prior to filling the reservoir.

3.2.2 Hydropower Tunnel Plugs

Tunnel complexes for hydroelectric projects are connected during construction for expediency, but they are later subdivided into pressurized and dry areas using plugs. Because large heads are required to generate electricity efficiently, the pressure gradients across the plugs can be quite high. Watertight doors are required in the plugs to permit access to penstocks for inspection and repair. A tunnel plug was used in the Helms Pumped Storage Project excavations near Fresno, California (see Appendix B.1).

3.2.3 Lake Taps

"Lake taps," an unusual use of concrete bulkheads in most parts of the world, are fairly commonplace in Norway. This procedure involves tunneling into an existing lake (Berdal et al., 1985). The plugs are used to protect gates and other underground workings during blasting of the final rock "plug," which causes a sudden inrush of water and debris.

3.3 Defense Applications

Massive plugs are used to contain nuclear explosions during underground weapons tests at the NTS. These plugs are typically made of cementitious and/or earthen materials and must withstand temperatures of 1,000°F and pressures of 1,000 pounds per square inch (psi) for very short periods of time. Due to the severe design criteria and the fact that any leakage may constitute a violation of the Atmospheric Test Ban Treaty, the plugs have evolved into sophisticated seal structures. The sealing problem is compounded by the numerous instrument cables passing through the plugs. Details of these special seals are covered in Appendix C.1.

3.4 Design and Materials

3.4.1 General

As discussed in Chapter 1, current plans for repository sealing call for installing bulkheads at specific locations in shafts, ramps, and repository drifts to control water inflows and gaseous releases. Bulkheads will also serve to limit human intrusion. The load environment may include various combinations of water pressure, ground pressure, and thermal and seismic loads transmitted through the rock to the bulkhead sides. Additionally, although the drainage capacity of the fractured rock mass at Yucca Mountain is expected to be large relative to the minor quantities of water likely, the possibility of water pressure acting on the bulkhead face cannot be discounted. The required service life of these bulkheads is long (circa 500 years). With these constraints under consideration, the following discussion will emphasize permanent plugs constructed of cementitious materials. Bulkheads or dams of earthen materials will be considered backfill. Temporary bulkheads, and those made of noncementitious materials, such as steel and wood, will not be discussed in detail.

3.4.2 Design Philosophy

Auld (1983a) lists eight factors that must be considered in inundation plug design, (paraphrased below):

- purpose of the plug,
- type of opening (tunnel or shaft),
- location (geologic and working conditions),
- shape,
- head of water,
- surrounding rock quality and stress,
- plug strength, and
- method of construction.

With minor modifications, Auld's list can be generalized to cover factors important to all types of underground bulkhead design:

- performance criteria,
- emplacement environment,
- bulkhead geometry,

- materials, and
- construction method.

The remainder of this chapter will address these five considerations.

3.4.3 Performance Criteria

Performance criteria must be established for a bulkhead prior to its design. These include: (1) the purpose for the bulkhead, (2) the acceptable amount of leakage of the gas or water being contained, and (3) the design life of the structure. The various purposes for bulkheads were covered in an earlier section.

The amount of leakage to be tolerated presents a more difficult issue. Most bulkheads are required to be as tight as reasonably possible, yet it is rare in the mining and civil industries to establish quantitative criteria for allowable leakage prior to construction. If leakage is considered unacceptably high during operation, the plug may be subjected to additional grouting cycles at higher pressures with more penetrating grouts, and/or it may be lengthened. Some bulkheads, especially those used to retain hydraulic fill, have drains permitting leakage of one phase (water) while retaining solids.

The design life of bulkheads has not received much attention in the literature, yet it is an important issue in nuclear waste storage. Most concrete structures have a design life of 100 years, yet typically last much longer than that. Most mines have an operating life of less than 100 years, but abandonment plugs may be expected to provide longer service.

3.4.4 Emplacement Environment

There are several aspects of the emplacement environment important to plug design: (1) the quality of the rock mass surrounding the plug, (2) any significant geologic features that might permit leakage past the plug, (3) the method used to excavate the opening, and (4) the effect of underground activities on stresses at the plug location.

Like a dam on the surface, an underground bulkhead optimally should be located to take advantage of the natural geologic features; however, sometimes the location is dictated by other considerations. There will be more potential for leakage around plugs located in poor quality ground, requiring consideration in the design of grouting procedures and selection of plug shape and length. Leakage pathways around the plug can occur through shear zones and other discrete features that cannot always be readily sealed.

The method used to excavate the opening will impact the decision whether or not a keyway is required. Smooth-bored openings and openings with poor rock or excessive blast damage may require keying to ensure a structurally secure, leak-free installation.

Other underground activities may affect, or be affected by, the bulkhead installation. Care should be taken to avoid locations where future ground movements, due to mining or time-dependent ground movements, may cause stresses and displacements at the plug location and possibly affect its long-term integrity.

3.4.5 Bulkhead Geometry

Two aspects of the bulkhead geometry have a major impact on plug performance: shape and thickness (length). Figure 3-9 shows common shapes for inundation plugs, while Figure 3-10 shows shapes used for conversion plugs. Basic plug shapes are parallel-sided, arched, or tapered. Parallel plugs may be optionally keyed (hitched) into the surrounding rock, while arched and tapered plugs are inherently keyed. Preparation of a keyway requires extra site preparation, and keyed plugs, being larger, consume additional materials than plugs that are not keyed.

Garrett and Campbell-Pitt (1961) present experimental evidence from tests at West Driefontein in South Africa that keyed plugs are unnecessary for inundation service. In these tests, an experimental parallel concrete plug (Figure 3-11) was tested at pressures up to 6,800 psi; considerable leakage occurred through the rock surrounding the plug but without structural failure (Table 3-1). Other tests performed on the previously mentioned Virginia/Merriespruit boundary plug prior to commissioning (see Figure 3-1) were conducted at pressures up to 1,340 psi. These pressures caused little leakage and no distress to the main plug but resulted in considerable leakage past a 12-ft-long temporary plug used during the test. This work was conducted in hard quartzite excavated by drill-and-blast mining. The resulting rough walls evidently provided a good bearing surface for parallel plugs, but caution should be exercised in extrapolating these results to other situations involving smooth-walled excavations and variable wall-rock quality.

Table 3-2 is a summary of some design equations and methods used to determine the required thickness of rectangular plugs based on structural considerations. Early equations are based on beam flexure, punching shear, or bearing on inclined contact planes (Auld, 1983a). More recently, numerical methods have been used to analyze the stress distributions in and around plugs (Sitz et al., 1989). However, Garrett and Campbell-Pitt (1961)

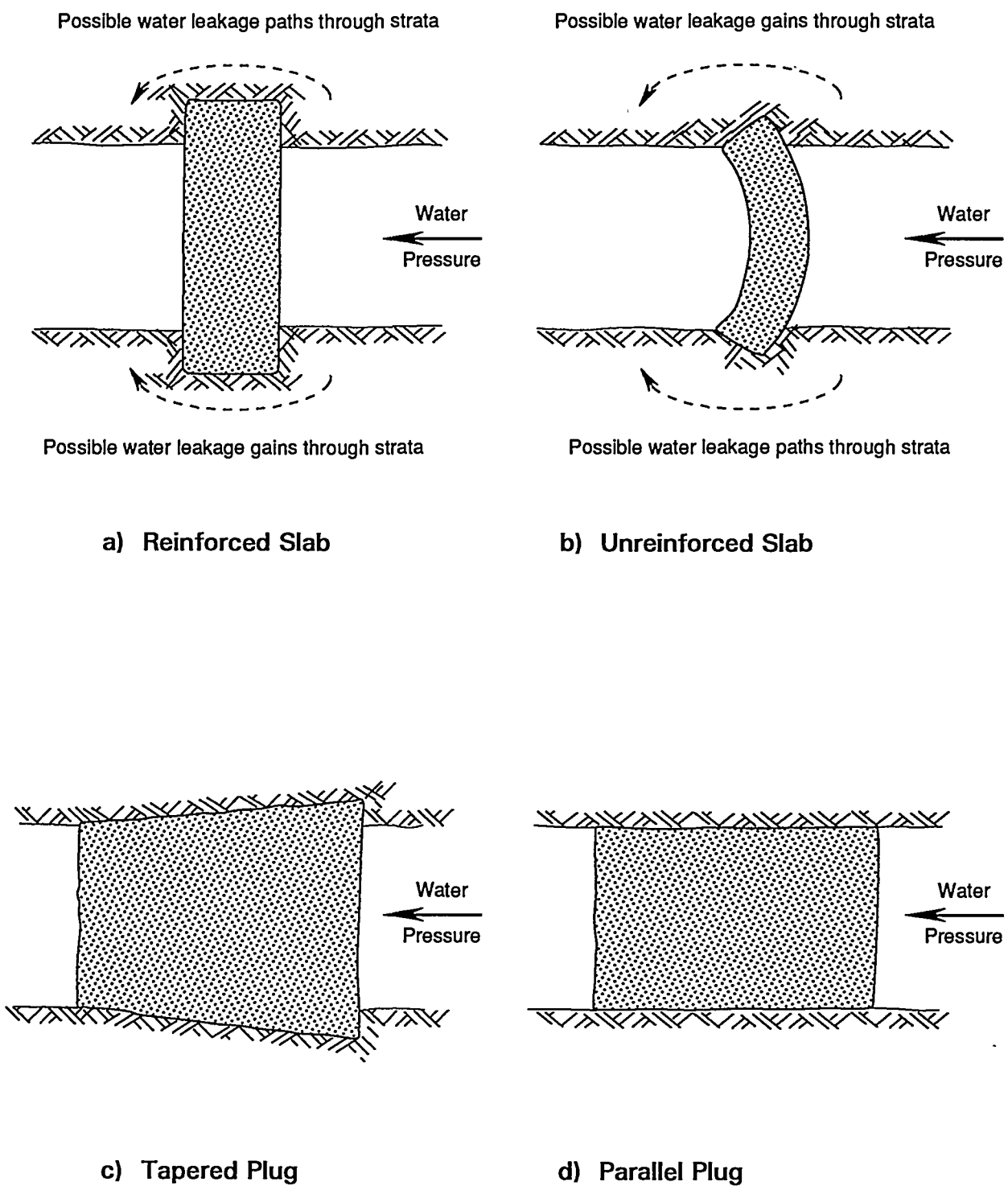
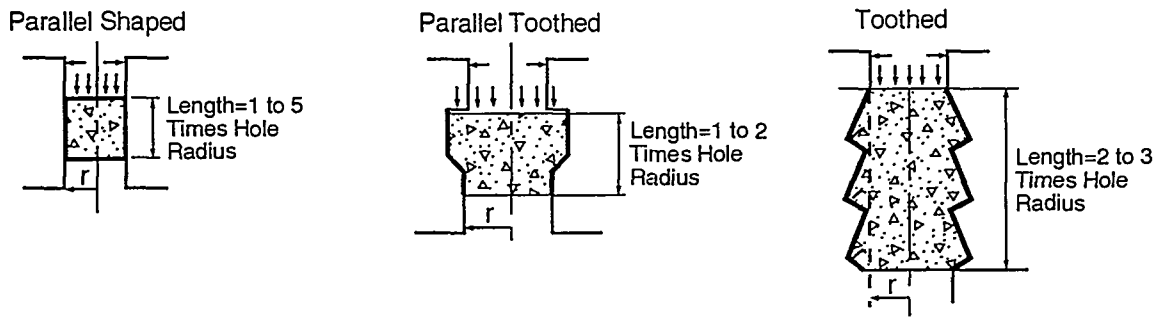
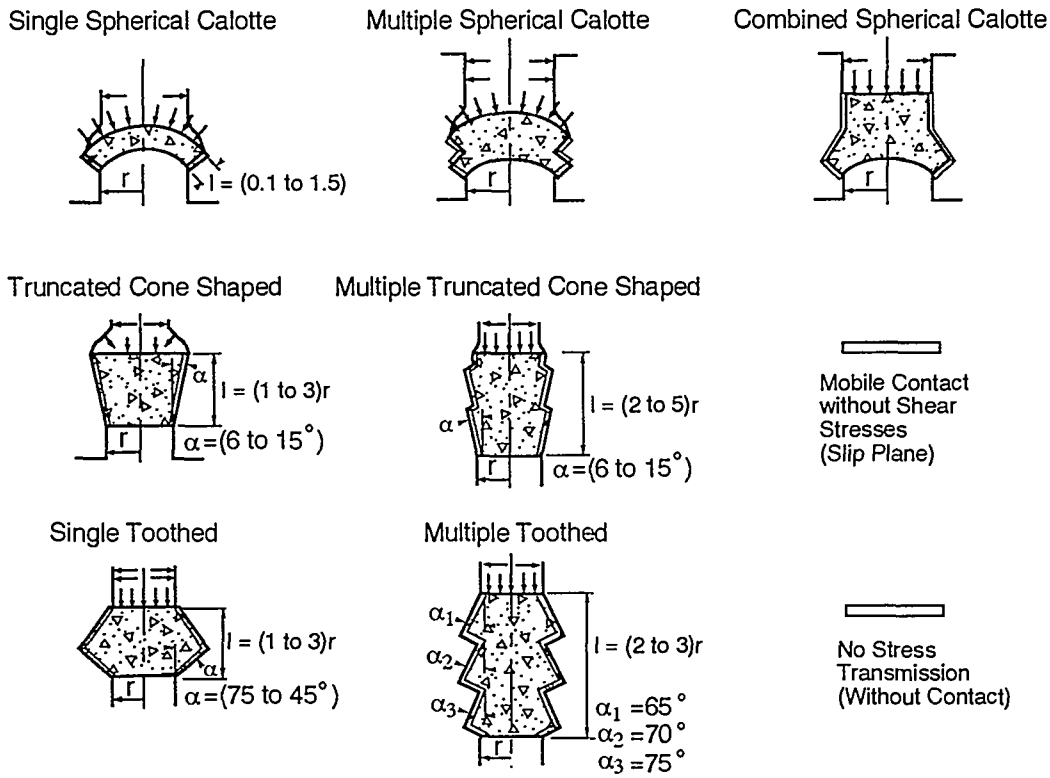


Figure 3-9
Common Shapes for Inundation Plugs (after Auld, 1983a, p. 193)



a) Rock-Connected Constructions



b) Nonrock-Connected Constructions

Figure 3-10

Conversion Plug Shapes (after Sitz and Forster, 1989, p. 196)

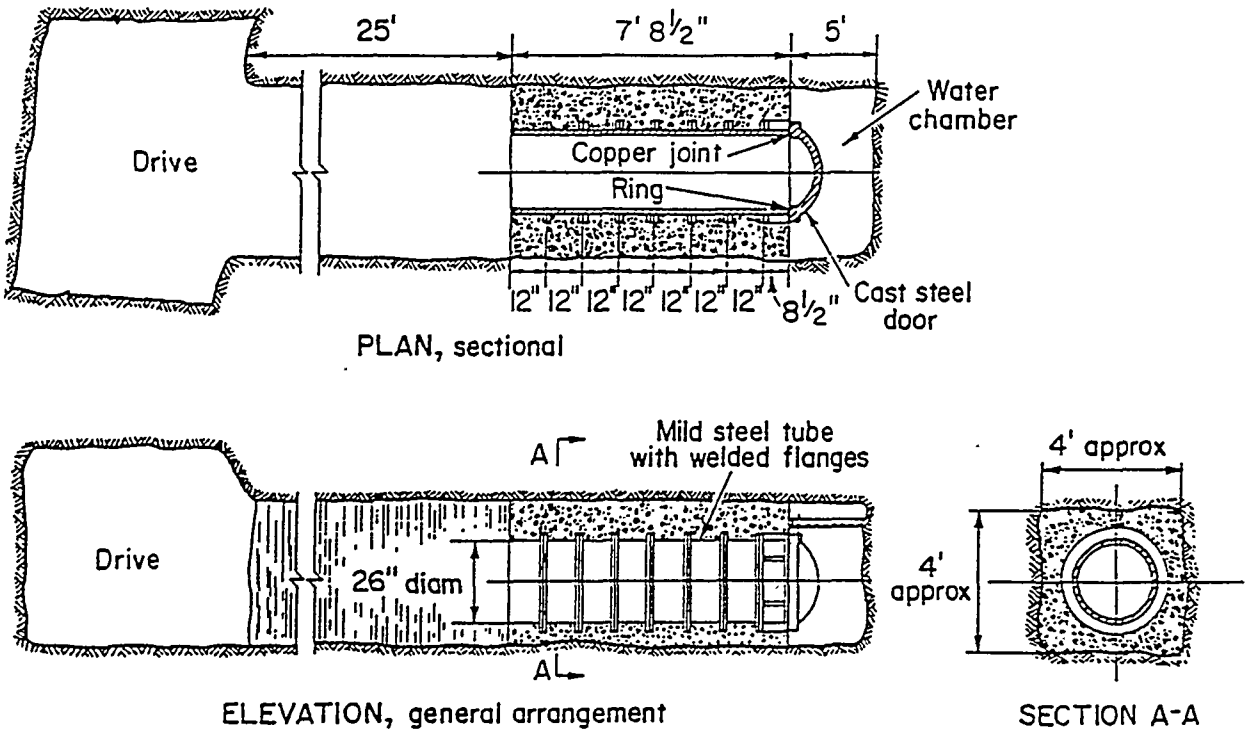


Figure 3-11

Dimensions of Experimental Plug at West Driefontein, South Africa (this plug withstood pressures of up to 6,800 psi without structural failure, despite lack of a keyway) (adapted from Garrett and Campbell-Pitt, 1958, p. 125)

discovered that strength was a secondary consideration in calculating plug thickness, the controlling criteria being leakage through the rock surrounding the plug. Table 3-3 shows estimated leakage values for plugs of various lengths. Garrett and Campbell-Pitt calculated the lengths of plugs required to control leakage, assuming normal grouting; this is shown plotted with recommended lengths based on bearing strength in Figure 3-12. Auld (1983a, p. 214) points out that "normal" grouting in South African practice is twice hydrostatic pressure, which is not normal in many other mining areas. Although they are specific to rock type and only apply to concrete plugs, the South African results have been widely used to design various types of bulkheads in different locations, due to a lack of other data.

Grice (1989) reports tests used to determine strength of 0.45-m-thick concrete block bulkheads used to retain hydraulically emplaced backfill at Mount Isa (see Figure 3-3b). Figure 3-13 shows design curves prepared using four methods as well as pressure test results. Only one of the three bulkheads was loaded to failure because of excessive leakage and an insufficient water supply in the first two tests. The third bulkhead failed at 750 kilopascal (kPa), equivalent to a head of 75 m. Inspection of Figure 3-13a shows that

Table 3-1

Summary of Test Results on Experimental Plug
(after Garrett and Campbell-Pitt, 1958, p. 128)

Test	Date	Cementation	Pressure (psi)	Remarks
1	7/31/57	No cementation.	75, 200, 310.	Heavy leakage on rock and concrete contacts particularly at hanging; tappings on hanging contact closed off to build up pressure >75 psi.
2	8/15/57 to 9/5/57	Rock and concrete contacts cementated at 3,000 psi.	650 to 750	Leakage on the rock and concrete contacts reduced; leakage past plug 50 gal/h at 1,750 psi.
3	9/12/57	Rock surrounding the plug cementated at 6,000 psi.	1800, 2500	Total leakage past plug 146 gal/h at 1,800 psi; 300 gal/h at 2,500 psi.
4	10/3/57	Leaks sealed by cementation.	4,300	An old diamond drill hole began to leak at 3,000 psi; leakage 128 gal/h at 4,300 psi; leakage stopped when pressure was reduced to 2,000 psi.
5	10/8/57	Further cementation to seal leaks.	5,700	Leakage not measured; pipe sleeve corrugated with crests of corrugations 15, 27, and 37 in. from the door face.
	10/15/57	——	6,200	Leakage in footwall of the drive 400 gal/h; no further distortion of the pipe sleeve was apparent.
	10/17/57	Footwall leak cementated.	6,800	Leakages in footwall and hanging of main drive; pressure could not be raised further.

the Peele (1941) equation (see Table 3-2), based on simple beam bending, was fairly accurate. The bearing equation of Garrett and Campbell-Pitt (see Table 3-2) (which is not really appropriate for thin brick bulkheads) led to an insufficient plug thickness, while the other two methods were overly conservative. Large safety factors are recommended to cover the uncertainty in design of critical bulkheads; fortunately, this is readily achieved by increasing bulkhead thickness in most cases. The practice of pressure-testing critical bulkheads prior to commissioning is also recommended.

Table 3-2

Rectangular Bulkhead Design Equations and Methods

Description	Reference(s)	Equation	Recommended Allowables
Thin plate equation	Chekan (1984)	$T = 0.865a \frac{p}{F_T}$	$F_T = 150$ psi
"Flat dam" equation (uniformly loaded, simply supported beam flexure)	Peele (1941)	$T = 0.707a \frac{p}{F_T}$	$F_T = 50$ psi
Parallel plug equation—smooth walls (punching shear)	Jeppe (1946) Lancaster (1964) (as presented in Auld, 1983a)	$T = \frac{pab}{2(a+b)F_s}$	$F_s = 50$ (concrete on smooth walls) $F_s = 100$—150 (reinforced boundary) $F_s = 85$ (concrete, ungrouted) $F_s = 120$ (concrete, grouted)
Parallel plug equation—rough walls (bearing strength)	Garrett and Campbell-Pitt (1961)	$T = \frac{pab}{(a+b)F_c}$	$F_c = 600$ psi
Tapered plug equation—rough walls (bearing strength)	Auld (1983a)	$T = \frac{pa_{max}b_{max}}{(a_{avg} + b_{avg})\ F_c\ (1+\tan\alpha)}$	Use compressive strength appropriate for direct bearing (concrete or rock)

Table 3-2
Rectangular Bulkhead Design Equations and Methods (Continued)

Description	References(s)	Equation	Recommended Allowables
Tapered plug equation—smooth wedge	Auld (1983a)	$T = \dfrac{p a_{max} b_{max}}{2(a_{avg} + b_{avg})\, F_c \tan\alpha}$	Use compressive strength appropriate for direct bearing (concrete or rock)
Flexural stress (plate fixed on three sides	Chekan (1984)	$T = b \dfrac{\beta p}{F_T}$	F_T = 40 psi (nonreinforced masonry)
Complex plugs	Sitz et al. (1989)	Finite-element method	

where a = bulkhead width (in.)
b = bulkhead height (in.)
T = bulkhead thickness (in.)
p = pressure (psi)
F_T = tensile stress/strength
F_c = compressive stress/strength
F_s = shear stress/strength
β = water height factor (see Young, 1989, p. 469)
α = plug taper angle (tapered plugs only)
avg = average

Table 3-3

Typical Plug Dimensions and Pressure
(after Garrett and Campbell-Pitt, 1961, pp. 1290 and 1292)

Mine	Plug Length (ft)	Pressure (psi)	Pressure Gradient (psi/ft)	Leakage (gph)
C.M.R. No. 6	17	360	21.2	Estimated 400
Free State Geduld	27	828	30.7	——
Free State Geduld	100	2250	22.5	——
East Daggafontein	28	650	23.2	Estimated 800
West Driefontein	8	6800	887.0	300/400 heavy grouting
West Driefontein	42	1827	43.5	Estimated 200
Virginia	63	1650	26.2	——
Virginia	36	1340	39.2	——
Virginia	12	1340	111.7	Estimated 1000
City Deep	12	100	8.3	——
Crown	16	111	6.9	——
Sub Nigel	11	199	18.0	——
Government GM Areas	5	100	20.0	——

Precautionary plugs with doors have additional design considerations. The concrete is cast around a cylindrical steel tube containing watertight doors. In addition to the hydrostatic loads on the end of the plug, the concrete cylinder must withstand external loads from strata and grouting pressures. Thick cylinder equations can be used to calculate stresses in the concrete. Also, the doors themselves must be adequately designed.

3.4.6 Bulkhead Materials

The most common material used for underground bulkheads is cast-in-place concrete with portland cement binder. However, a wide variety of materials have been used, such as concrete blocks, bricks, and wood. Table 3-4 shows the results of an industry survey on materials used in ventilation stopping construction (Adam et al., 1986). Because

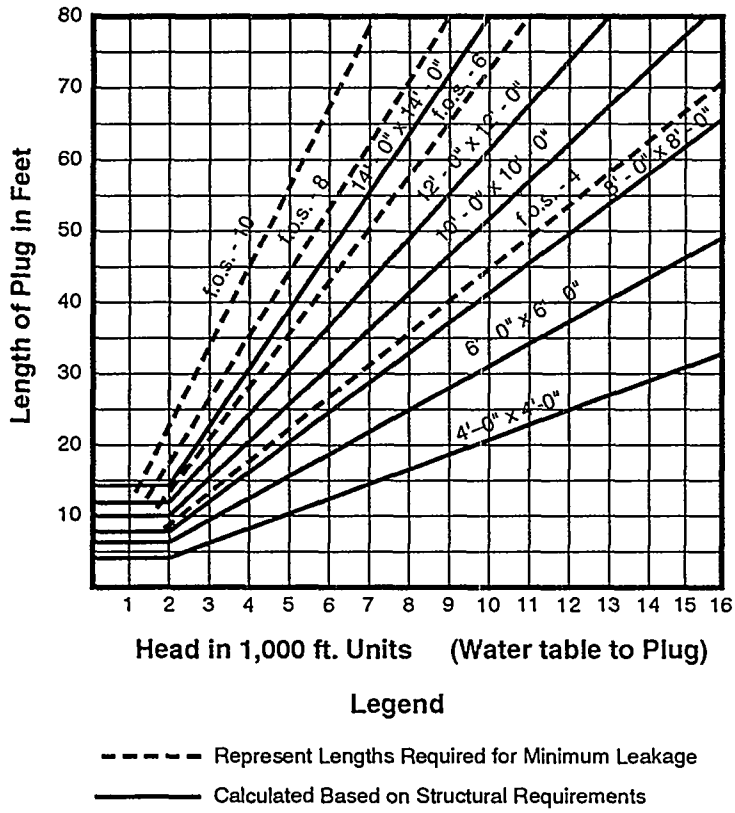
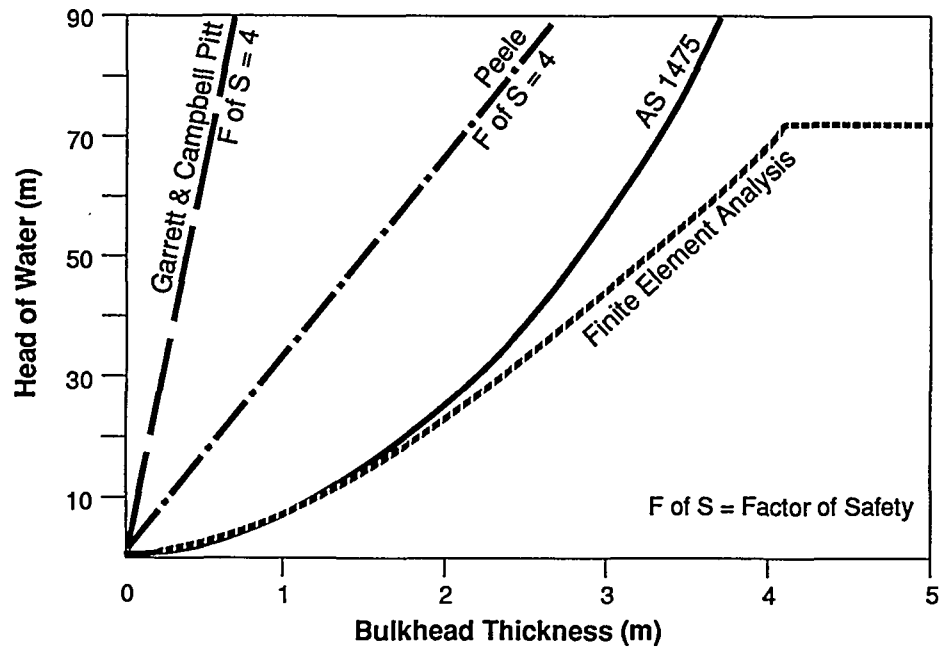


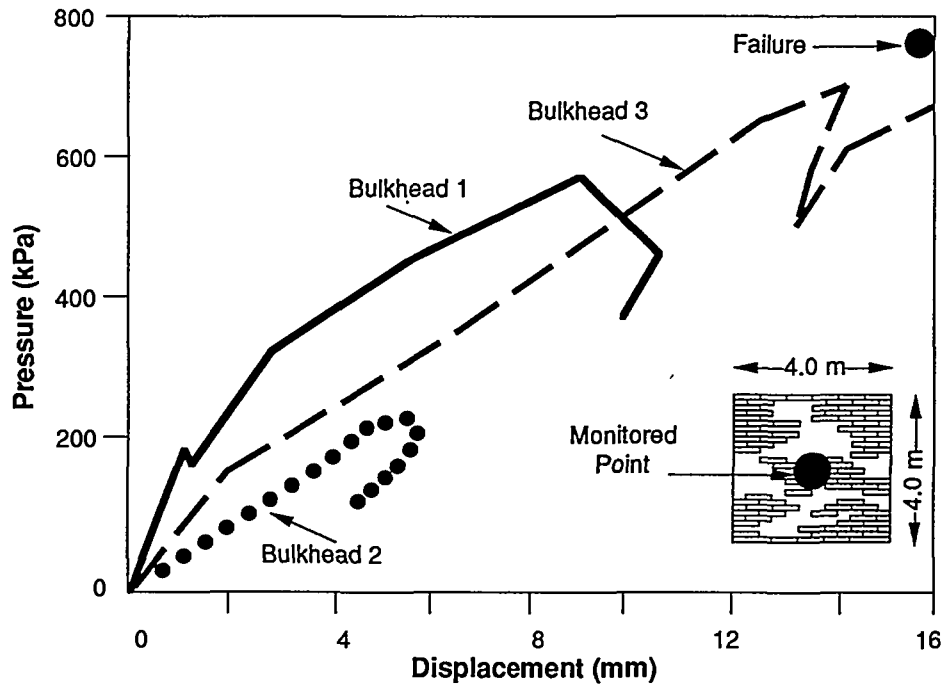
Figure 3-12
Nomograph for Determining Required Plug Length
(after Garrett and Campbell-Pitt, 1958, p. 142)

materials are predominate in plug and pressure applications, some additional discussion is warranted.

The USBM has recommended a mix of one part Type I portland cement, two parts clean sand, and four parts clean gravel for explosion-proof bulkhead construction in underground coal mines. They recommend a relatively stiff mix with a minimum of concrete. Tests by an independent contractor show that this mix results in an average strength of about 3,000 psi, as shown in Table 3-5 (Chekan, 1984). However, concrete placed in large masses will generate considerable heat from hydration, and cracking can occur. Shrinkage is a function of water content and will be minimized if the concrete is placed with a minimum of water; pressure grouting will generally be required to seal shrinkage cracks at the plug periphery after curing. Type IV concrete gives low heat of hydration, and Type V has high sulfate resistance. Pozzolan additives can improve workability, reduce cracking and shrinkage, and increase resistance to sulphates if properly applied, but should be carefully tested for possible adverse effects. Recently, other materials have been tested and approved



a) Design Curves



b) Test Results

Figure 3-13

Design Curves and Test Results for 0.45-m-Thick Concrete Block Bulkhead Used to Retain Hydraulic Fill at Mount Isa (after Grice, 1989, p. 20)

Table 3-4

Mining Industry Stopping Construction Types and Materials
from Survey of 19 Operating Mines in North America (from Adam et al., 1986, p. 31)

Stopping	Type	Timber	X	X	X			X	X		X	X	X		X	X			X		X	12
		Cement block				X		X	X	X	X	X		X		X	X	X		X		11
		Muckpile			X						X	X		X								4
		Metal	X	X												X	X					4
		Brattice				X		X		X			X	X				X	X	X		8
		Parachute	X																		X	2
		Styrofoam						X														1
		Formed and poured			X																	1
		Fans balancing					X															1
		Balloon							X													1
		Concrete in bags																	X			1
		Brattice and chain link																			X	1
	Sealant	Shotcrete	X		X				X				X								X	5
		Polyethalene		X		X					X	X			X							5
		Mandoseal				X		X		X							X	X	X	X	X	9
		Rags												X		X						2

Table 3-5

Ultimate Compressive, Tensile, and Shear Strengths for 1:2:4 Concrete Mix (from Chekan, 1984)

	Test 1	Test 2	Average
Strength (psi)			
Compressive	2,933	3,074	3,004
Tensile	233	255	244
Shear	NA	NA	766
Slump (in.)	NA	NA	1.5

NA = Not applicable.

for explosion-proof stopping construction (Stephan, 1990a and b). Notably, cementitious foams emplaced between forms have gained acceptance for abandonment bulkheads in underground coal mines.

Auld (1983b) discusses the important attributes of concrete for underground construction. In addition to strength, workability is important in the confined space where plugs are cast underground. Auld considers plasticizing admixtures essential for successful construction underground. Different plasticizing admixtures are available for different purposes. Auld suggests a plasticizer based on a processed calcium ligosulphate for normal plug construction.

A polyhydroxycarboxylic acid derivative is recommended as a retarding plasticizer if long distances need to be travelled underground. A superplasticizer containing synthetic sulphonated naphthalene/formaldehyde condensates is recommended for very tight situations, such as placing the upper sections of a plug near the roof, where ultimate workability is required. Auld presents two mixes that have been used for plug construction in the United Kingdom, shown in Table 3-6. The first was used to construct an emergency plug in a gypsum mine roadway; the second was used to construct a temporary consolidation plug during shaft sinking.

Table 3-6
Cement Replacement Mixes Previously Used by Cementation Mining Ltd. for Underground Plugs (after Auld, 1983b, p. 210)

Emergency plug in roadway: Grade 30 (OPC replacement with PFA); 30 N mm^{-2}	
Site	British Gypsum Ltd., Sherburn Mine
Supplier	Topmix Ltd.
Total cementitious content	400 kg m^{-3} (250 kg m^{-3} OPC, 150 kg m^{-3} PFA)
Sand	770 kg m^{-3} (Elvasion Zone 2)
Sand % of total aggregate	42%
Coarse aggregate	1050 kg m^{-3} (Elvasion Gravel)
Water	180 litres m^{-3}
Water/cement ratio	0.45
Slump without plasticizer	50 mm
Plasticizer	Flocrete N (Cementation Chemicals Ltd.)
Slump with plasticizer	160 mm

Temporary consolidation plug in shaft: Grade 55 (OPC replacement with Cemsave ground granulated blast furnace slag); 5.5 N mm^{-2}	
Site	National Coal Board, North Selby No. 1 shaft
Supplier	Topmix Ltd.
Total cementitious content	500 kg m^{-3} (30% OPC, 70% Cemsave)
Sand	595 kg m^{-3} (Blaxton Zone 3)
Sand % of total aggregate	34%
Coarse aggregate	1150 kg m^{-3} (Blaxton Gravel)
Water	180 litres m^{-3}
Water/cement ratio	0.36
Slump without plasticizer	60 mm
Plasticizer	Flocrete N (Cementation Chemicals Ltd.)
Slump with plasticizer	160 mm

3.5 Construction Method

3.5.1 General/Site Preparation

The bulkhead emplacement method will be affected by the type of bulkhead, the construction environment, and the plug location. The plug location should be selected considering the use of the plug and the local geological conditions, particularly with respect to how they will affect leakage. The rock surface against which the bulkhead will be attached must be carefully prepared. Loose rock should be scaled and additional ground support installed in the plug area, if required. If the plug requires a keyway, this should be carefully excavated by nonblasting methods, if possible. Some pregrouting via radial holes may be performed prior to bulkhead construction.

3.5.2 Block Bulkheads

Block bulkheads are relatively straightforward to install. First a keyway or hitch is constructed in the ribs; if they are coal, they are easily excavated. If it is an important stopping, a concrete floor may be placed. Concrete blocks are hand-carried to the site and stacked in rows. Partial blocks may be used near the roof and ribs. Various materials are inserted into gaps between the wall and the roof. The entire bulkhead may then be sprayed or painted with a sealant to minimize air leakage.

3.5.3 Cast-in-Place Concrete Bulkheads

Bulkheads made of cast-in-place concrete must first be formed. Forms will be built in the usual manner following site preparation; special forms will be required for inundation plugs with watertight doors. For bulkheads less than 3 ft thick, plywood forms are appropriate; otherwise, retaining bulkheads of concrete, block, or brick are recommended by Chekan (1984). Any water that has collected in the keyway should be pumped out prior to placement.

There are two methods of handling concrete delivery: (1) batching the concrete on the surface and delivering it to the bulkhead site and (2) transporting the concrete components underground and mixing them at the site. A surface batch plant location enables better quality control at the plant but may necessitate long-distance transportation of the mix. Concrete specifications should be similar to pumped concrete, and plasticizers may be required (Auld, 1983b). Abandonment bulkheads may be of the double-bulkhead type depicted earlier in Figure 3-4b, with block bulkheads used as forms for the concrete core. If

access is not available underground to construct block bulkheads, gravel bulkheads may be emplaced and grouted from the surface prior to placing the core (Beck, 1981) (Figure 3-14). The installation can be inspected using a borehole television camera. Bulkhead construction in an underground coal mine is often performed using a portable concrete mixer. The concrete is placed behind form work built at the site. Particular care must be taken to place the concrete tight against the back. Auld (1983b, p. 29) discusses a method for casting concrete up to the excavation roof level.

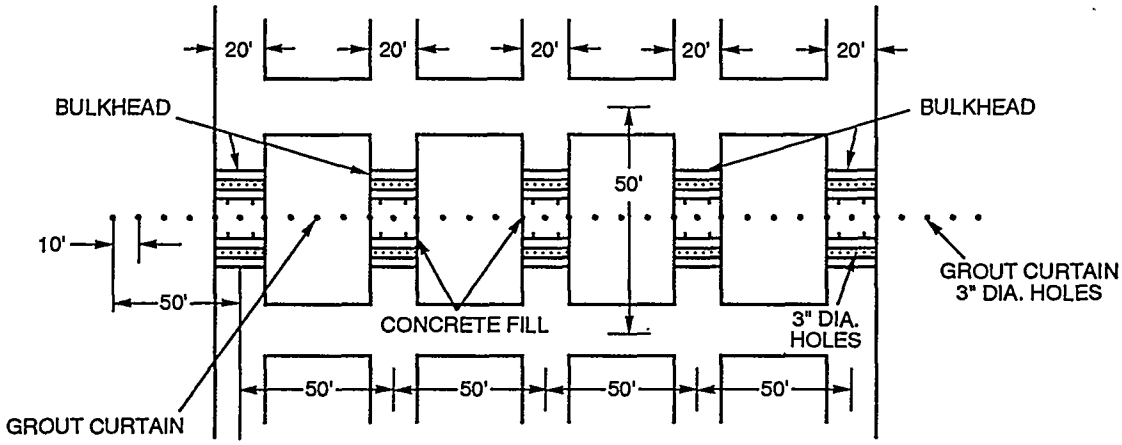
> *"In tunnels with special steel shutters containing porthole access ports, little problem exists as pumping pressures can be employed to fill the shutter to roof level. For an inset roof, access for concreting is normally through the stop end. Concrete pipes are led through the stop end to the farthest extremity and withdrawn gradually as placing proceeds. With this process, the concrete pipe must be situated as high as possible in the crown, to ensure complete filling to that level as the pipe is withdrawn. Concrete containing a superplasticizer is needed for this particular application. Poker vibrators attached to rigid poles are used only to assist the almost self-compacting action."*

Cast-in-place bulkheads will shrink during curing and must generally be grouted if they must contain fluids under pressure. Pipes may be cast in the concrete to grout the boundary fracture and place a radial preload on the plug.

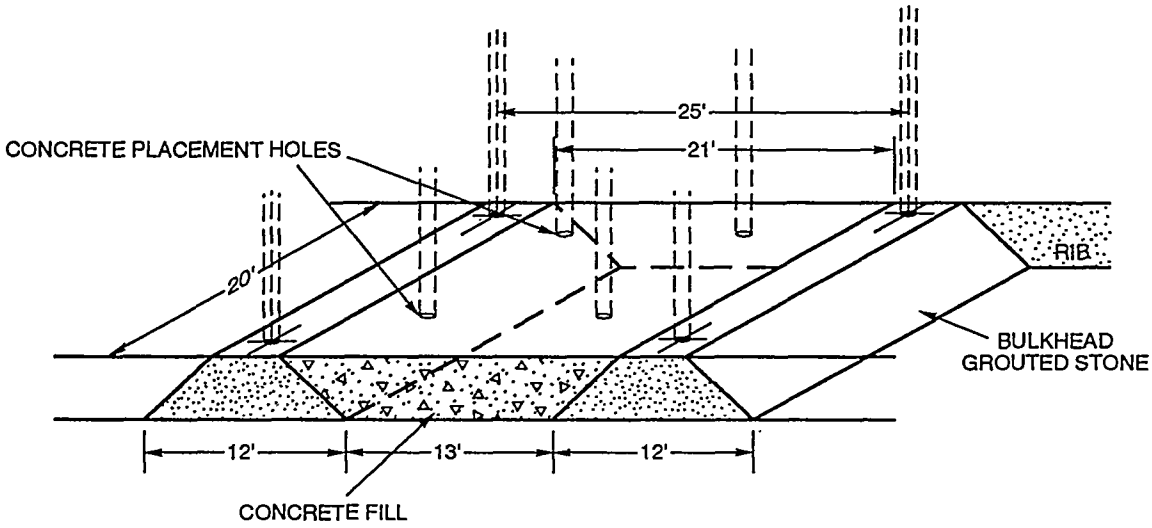
3.5.4 Grouted Concrete Bulkheads

Known also as Colcrete, Prepakt, and Preplaced Aggregate Concrete (Houlsby, 1990, p. 6), grouted concrete is a very effective method of constructing underground bulkheads. The process was originally patented in the United Kingdom in 1930 and has been used worldwide for many years. Examples of Colcrete inundation plugs in French mines are provided in Estivalet (1984). Regarding South African practice, Garrett and Campbell-Pitt (1961, p. 1298) state,

> *"The placing of concrete underground is, in any case, a difficult business where mixing conditions are not at their best and the control of water/cement ratio is precarious in the extreme. Moreover, the physical difficulty of placing concrete in the confined space close to a hanging wall is such that a good well-consolidated job is nearly impossible. This is of major importance when the inevitable roughness of a*



a) Plan View of Sealed Entries



b) Isometric of Sealing Method

Figure 3-14

Method of Sealing Abandoned Coal Mine Entries from Surface When Underground Access Is Not Available (Beck, 1981, p. 38)

hanging is considered against the need to get adequate contact between it and concrete.

The use of grouted concrete with a pre-placed aggregate, however, answers most of these problems. It makes possible the use of broken rock from the workings as the coarse aggregate, and such rock is much more easily handled in confined spaces than is concrete. Since a much smaller volume of sand and cement is needed; it can be mixed in a special machine and pumped long distances, up to 2000 ft. This makes possible the mixing of grout under controlled conditions with an adequate control of water/cement ratio, and the pumped grout has the effect of filling the interstices between the stones fully and making a reasonably good contact with the hanging wall. Moreover, pre-placed stone tends to interlock mechanically and provides a better resistance to shear."

Garrett and Campbell-Pitt recommend +2-in. clean aggregate, a water/cement ratio of 0.6, and a sand/cement ratio of not more than 2.0. They caution against problems with laitance, or low-density grout, that accumulates on the surface of the grout as it rises. If this is not drained off, it causes leakage planes through the plug during grouting stoppages. They recommend dividing a long plug into convenient sections with vertical partitions so that each section can be placed to the roof in one continuous operation.

Grouted concrete is frequently used in construction of emergency plugs to control water inflows. Figure 3-15 shows a method of constructing bulkheads to control water inflows in South African mines. Figure 3-16 shows a similar construction method used in Hungarian mines subject to inrushes. A dam is built first to handle the water temporarily (Figure 3-16a) until the plug can be emplaced. Then, concrete end plugs are placed filling the intervening space with aggregate, which is grouted to make the plug (Figure 3-16b). The water valve can then be closed and further grouting performed to completely seal the inrush (Figure 3-16c).

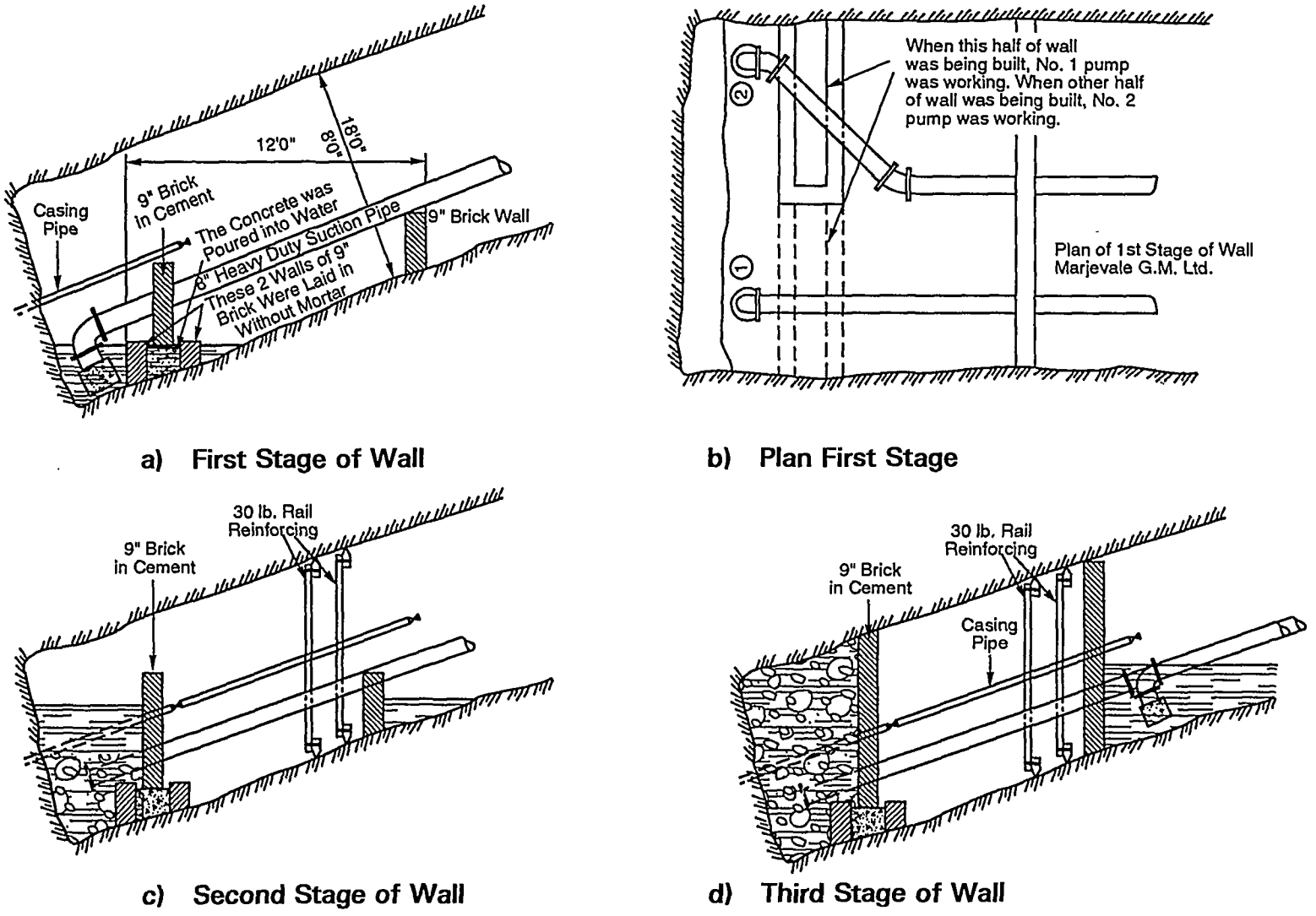


Figure 3-15

Method of Constructing Bulkheads to Control Water Inflows in South African Mines (space between walls is subsequently filled with concrete and grouted) (after Jeppe, 1946, pp. 758-759)

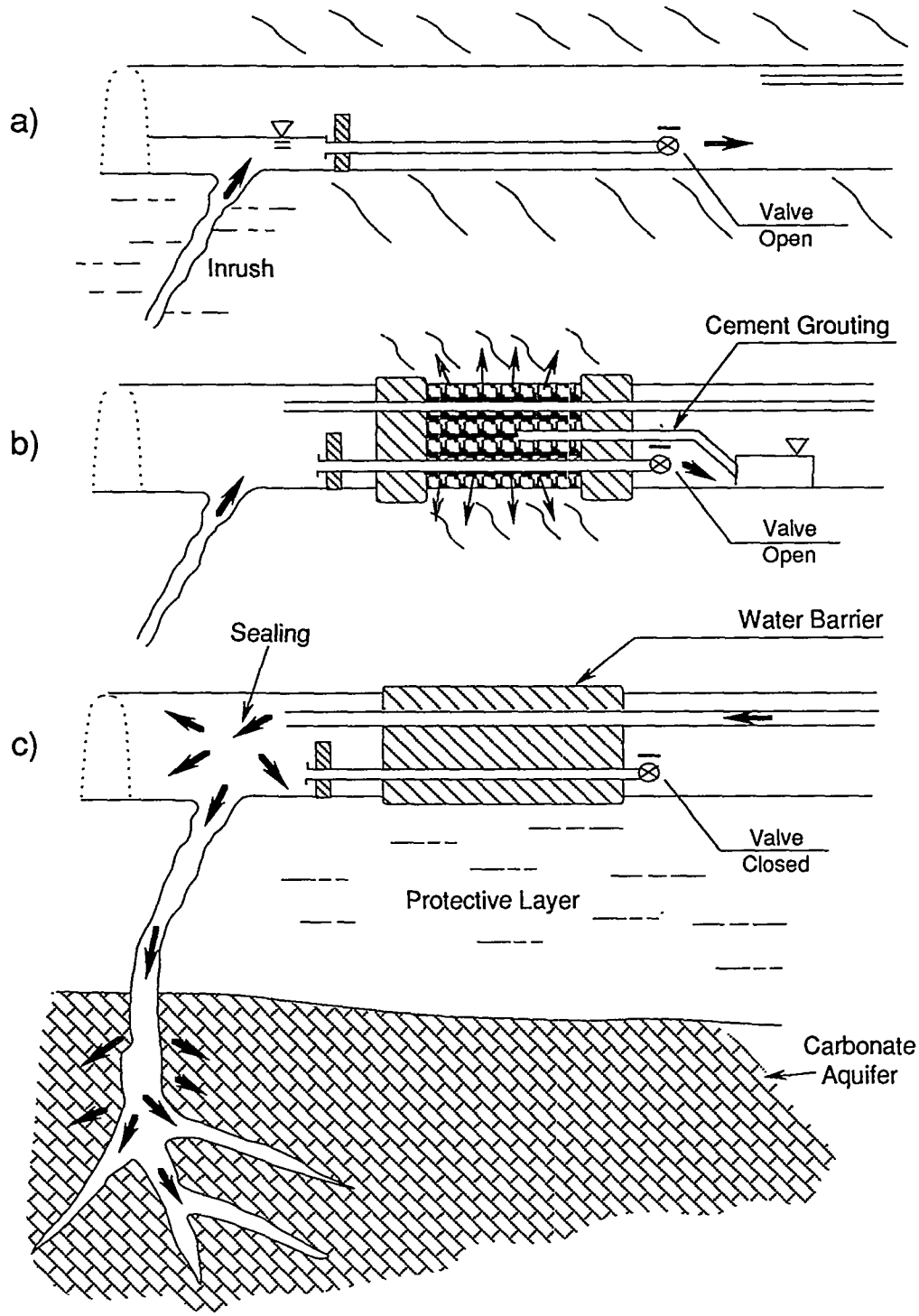


Figure 3-16

Use of Bulkheads to Handle Groundwater Inrushes in Hungarian Mines
(from Kapolyi et al., 1979, p. 135)

4.0 Available Technology for Grouting

4.1 General

For the purposes of this report, grouting may be defined as the process of injecting materials into cracks or voids in geologic materials, generally for the purpose of strengthening them and/or reducing their permeability. The specific uses of grouting in civil and mining applications include:

- general permeability reduction of porous materials (penetration grouting),
- fracture grouting of jointed rock masses,
- strengthening or compaction of soils or weak rock masses (consolidation grouting),
- repair of collapsed zones,
- soil replacement (jet grouting),
- soil jacking,
- anchoring rockbolts and tendons,
- filling contact joints between concrete structures and the ground (contact grouting),
- repair of cracks in concrete structures, and
- in situ concrete batching (grouted concrete).

Materials used for grouting must be fluid enough to be easily injected into the formation, yet must set up with sufficient strength to achieve the desired effect and resist displacement. There are three broad classes of grouts: (1) chemical grouts, (2) cementitious grouts, and (3) clay grouts. Selection of the type of grout is based primarily on penetrability and cost, although nuclear waste applications also involve questions of durability. Figure 4-1 compares the penetrability of the various classes of grouts.

Grouting has a long history of successful application in the civil and mining industries. Used at least as early as ancient Roman times for repair and strengthening of civil structures, the early practice was severely limited by drilling and pumping technology. Houlsby (1990) describes a number of civil grouting jobs from 1802. Perhaps the first systematic application of grouting was for the foundation of the New Croton Dam in New York state, to seal and strengthen a jointed limestone formation. Curtain grouting was used to cut off water flows beneath dam foundations for the first time on the Estacada Dam project near Portland, Oregon, reported in 1915. Most early grouting was done with cementitious or bituminous materials. Chemical grouting became viable in 1911 when sodium silicates were used on a

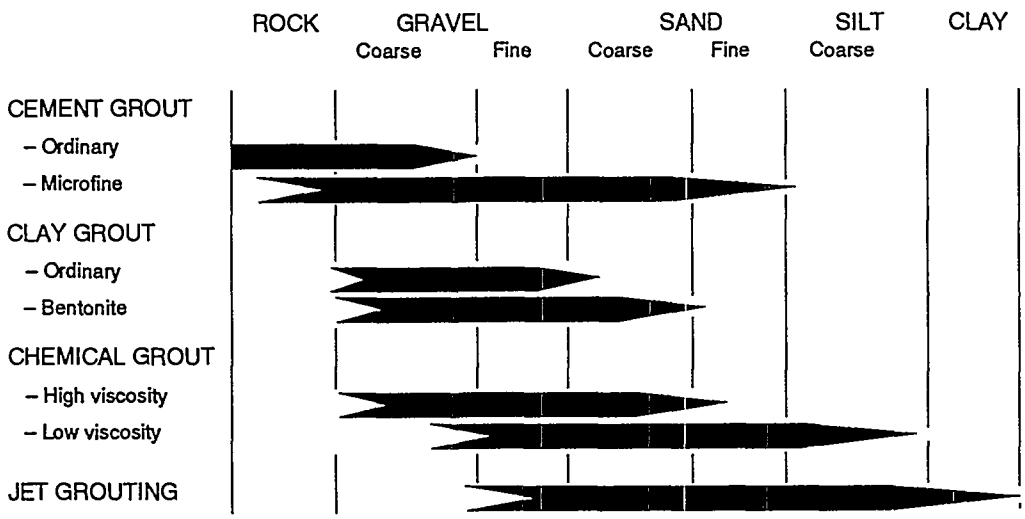


Figure 4-1
Approximate Penetrability of Various Classes of Grouts
(after Houlsby, 1990, p. 8)

coal mine shaft-sinking job in England and when Joosten patented the two-stage sodium silicate process in 1925. A variety of chemical grouts were subsequently developed and used for special applications.

In 1932, the U.S. Bureau of Reclamation (USBR) commenced work on the Hoover Dam project, the first in a long series of dams involving extensive cement grouting. The Tennessee Valley Authority and the U.S. Army Corps of Engineers also embarked on dam-building phases involving grouting at about this time. Considerable testing of grout mixes was performed by the Waterways Experiment Station. Most of these dam projects involved fluid mixtures of cement and water. USBR practice culminated in the New Waddell Dam project (Appendix B.2), where superplasticizers were used to permit injection of stiff mixes into a fractured volcanic sequence; the entire grouting process was monitored in real time using a computerized system.

Technological improvements, including the development of modern drilling equipment and pumps, along with improved understanding of the grouting process, have led to the current state-of-the-art as described in this report.

The current hydrogeologic data indicate emplacement of grout in fractured, welded tuff (Tiva Canyon and Topopah Spring Members) and possibly in nonwelded tuff. Fracture grouting could be associated with the following specific sealing components: shaft and ramp seals, borehole seals, single embankments, and bulkheads at the repository horizon. Grout curtains associated with these sealing components can be vertically oriented, such as below a single embankment, or horizontally oriented, such as around a shaft seal.

4.2 Materials

4.2.1 Chemical Mixtures

A number of different chemical mixtures has been used for grouting. The most common is sodium silicate, a viscous liquid that must be mixed with water to make a colloidal solution suitable for injection. The concentration varies, depending on the strength and penetrability requirements for the grouted soil. Structural grout mixtures are typically 50% sodium silicate, $\pm 10\%$. Dilute mixtures of 25% sodium silicate are necessary to penetrate fine materials. Reactants are required to cause the grout to gel. Accelerators, such as metal salts, are often part of the reactant formulation. Early practice used a two-stage injection process where the sodium silicate solution was injected first, followed by the reactant mixture. However, single-stage mixtures have since been developed that perform well and greatly simplify the grouting process. Modern silicate grouts can be prepared with gel times from one minute to over an hour. Peak strengths are generally achieved in one week or less.

Baker (1982) estimates that approximately 90% of contemporary chemical grouting practice employs sodium silicate grouts. Other grouts used for geotechnical applications include acrylamides, acrylate grouts, and polyurethane grouts. However, there are questions of toxicity with some of these grouts, especially acrylamides, and most are seldom used despite often excellent performance. Acrylamide has a very low viscosity that remains constant until gel time and has superior penetrability, but it is neurotoxic and used little today. However, an acrylate grout (AC-400) has been developed that has many of the desirable properties of acrylamide grout without the toxicity.

Polyurethane grout is composed of prepolymer, plasticizer, diluter, surfactant, and catalyzer. When polyurethane grout comes in contact with groundwater it reacts, in the process expanding and increasing the grouting radius. Polyurethane grouts are relatively expensive, but their use results in a high-strength grouted soil mass.

Bruce (1989) reports on a new type of chemical grout (Silacsol), which is a true solution of activated silica rather than a colloidal suspension such as sodium silicate grouts. When mixed with a calcium-based inorganic reagent, crystalline calcium hydrosilicates are formed similar to portland cement. Silacsol has many of the advantages of cement grouts, including lack of environmentally objectional organic reagents, but has superior penetrability and may be more widely used in the future.

Treatment of chemical grouting in this report is limited, because questions concerning toxicity and durability of this class of materials make them currently less interesting as candidates for repository use than cementitious- or clay-based grouts. Additionally, in contemporary practice their use is concentrated in soil-grouting applications rather than in rock formations where seals will be located. More complete discussions of chemical grouts are contained in numerous publications, including a comprehensive textbook by Karol (1983).

4.2.2 Cementitious Materials

Cementitious grouts are used in most geotechnical grouting jobs in the United States today for both rock and soil grouting. Types of cementitious grouts include portland cement grouts, blast furnace cement, and microfine cements. These grouts have the advantages of relatively low cost, versatility, low toxicity, and known properties. Disadvantages are limited penetrability in very fine formations, although microfine cements can have nearly as good penetrability as most chemical grouts. Complete information on cement grouting is available in numerous technical references, including Houlsby (1990); only an overview is presented here.

Portland cements are composed of four basic components: (1) tricalcium silicate (C3S), (2) dicalcium silicate (C2S), (3) tricalcium aluminate (C3A), and (4) tetracalcium aluminoferrite (C4AF). ASTM classifies portland cements into various types that have different characteristics depending on the relative proportion of the four basic components and the fineness of the grinding. Most grout is done with Type I portland cement, which is the cheapest. Type II cement is moderately resistant against sulfate attack because of reduced C3A content. Type III cement develops high early strength because of its finer average particle size. High early strength is not generally a characteristic useful for fracture grouting, nor is the particle size sufficiently finer to significantly increase penetrability. Type IV cement is formulated to give lower heat of hydration, again not typically a concern in grouting. Type V is formulated to be high sulfate-resisting. The sulfate-resisting types are useful for grouting in chemically aggressive environments.

Blast furnace cement is made from blast furnace slag and has a high resistance to sulfate attack.

Because cement particle size partially controls the penetrability of cementitious grouts, microfine cements have been developed. These cements are mixtures of portland and blast furnace cement that have been ground very fine (Figure 4-2). Their penetrability is very high and for some purposes approaches that of chemical grouts. Microfine cements have been used successfully in the Helms Pumped Storage Project (see Appendix B.1), as well as at the NTS (see Appendix C.1).

A number of additives are often added to cementitious grouts to improve their properties and/or as a cost-saving measure. When large voids must be filled, bulk fillers, such as sand and coarser aggregates, may be added to the grout to reduce the amount of cement required. Reactive additives include pozzolans, such as fly ash and silica fume, which are sometimes added to grout as cement-saving additives and to improve resistance to chemical attack.

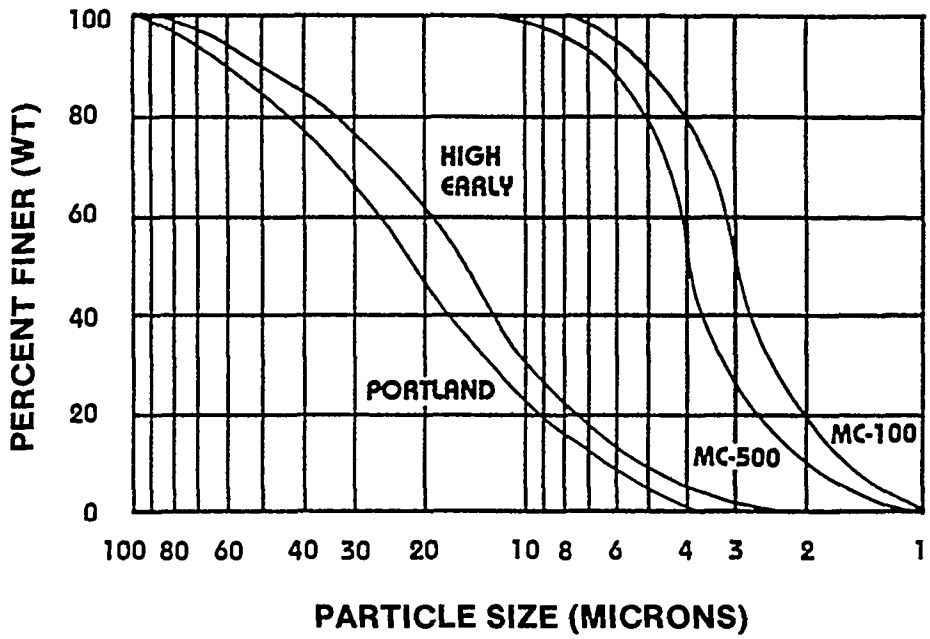


Figure 4-2
Particle Size of Type I and Type III Portland Cement Compared to Two Microfine Cements (after Clarke, 1987)

Additives to improve properties include sodium silicate and bentonite, which are useful as accelerators to prevent cement grout curtains from being washed away prior to setting. Bentonite is also useful for reducing bleed. Superplasticizers are useful for reducing the water-to-cement ratio while improving penetration. A case history of the successful use of superplasticizers to grout the New Waddell Dam foundation is described in Appendix B.2.

Cementitious grouts are particle suspensions that are subject to two constraints on their penetrability. First, the maximum diameter of the particles must be smaller than the fissures or pores to be grouted. The conventional rule of thumb is that the width of the void to be grouted must be 3 times the particle diameter. This constraint can be addressed by the use of microfine grouts, as discussed earlier. The second constraint is pressure filtration, which causes the water to drain off and may include rapid viscosity increases, resulting in clogging of fine fissures and voids. Bruce (1989) reports on a new class of cement-bentonite grouts (MISTRA) that have the following properties:

- very low filtration rates,
- no bleeding,
- low values of yield point and plastic cohesion over an adjustable time period, and
- high strength and permeability relative to conventional grouts with the same cement content.

Finally, considerable research specifically directed at immobilizing or containing radioactive and chemical waste is under way to improve performance of waste repositories. In a recent issue of Waste Management, the article "Cementitious Materials in Radioactive Waste Management" focused on numerous issues such as:

- immobilization of radioactive waste using portland cement-based materials,
- transport of gases in concrete barriers,
- cement-based barriers for C-14 isolation, and
- predictive capability of grout performance over long periods.

Research at the STRIPA Project also focused on establishing the properties of cement-based grout (Onofrei et al., 1992). Research concluded that it is possible to manufacture

high-performance cement grouts that can be injected into very fine fractures and divert water in granitic rock. The grouts investigated had negligible hydraulic conductivity, with a very low porosity and were found to be highly leach-resistant under simulated repository conditions. It was also shown that grout containing silica fume as a pozzolanic material and superplasticizer as a water reducer, if mechanically disrupted, has the ability to self-heal. However, further investigation was proposed because of the uncertainty surrounding the performance of grouts older than two years.

Extensive research is also being funded by the DOE Office of Technology Development specifically aimed at developing and testing grout materials. The purpose of this research is to contain and stabilize waste at DOE sites as part of the current environmental restoration project. An example of such research is that currently being performed by Brookhaven National Laboratory Associated Universities, Inc. (Allan and Kukacka, 1993). It was concluded in this research that cement-based grouts can be used to produce low-permeability, high-strength, and high-durability subsurface barriers suitable for in situ containment of buried chemical waste. For grout barriers, the compressive strength under in situ curing condition was 29.4 MPa, and the permeability was 6.6×10^{-11} cm/sec for in situ curing. Soil cements showed compressive strength values ranging from 27.0 to 9.0 MPa for in situ curing and permeability ranging from 1.2×10^{-10} to 9.5×10^{-9} cm/sec.

4.2.3 Clay Materials

Clay-based grouts have been used extensively in Russia for grouting saturated fractured rock (Kipko et al., 1993). Such grouts have been emplaced in vertical and inclined shafts and inclined and horizontal drifts and tunnels. Recently, they have been used to stabilize foundations and isolate sources of groundwater contamination. This technology is in fact the mainstay of the Russian approach to controlling groundwater inflow into underground openings in fractured rock. In comparison, cement grouting has been regarded as the main method of combating the inflow of groundwater in fractured rock throughout the world.

This technology, which has been developed by the SPETSTAMPONAZHGEOLOGIA (STG) Enterprise over the last three decades, represents an integrated method of grouting saturated ground. The method includes a quantitative evaluation of the grouting process, hydrodynamic investigation of the hydrologic environment prior to grouting, development of clay-based grouts, placement of grout, and the quality controls needed to achieve an effective operation.

The clay-based grouts are visco-plastic systems. They are comprised of structure-forming reagents (cement and various chemical additives) and a clay mineral mortar. The most effective grouts for preventing the inflow of groundwater into underground workings are grouts that have a cement content of 8 to 10% of the clay grout by mass. The principal constituent of the grout is clay. Cement is important because it controls the final strength of the grout. There are three groups of clay minerals: kaolinitic, montmorillonitic, and hydromica clays, which have different capabilities to be hydrated or dispersed in a water medium. STG recommends the following percentages in terms of principal oxides: 63 to 69% SiO_2, 19 to 29% [R_2O_2 (Al_2O_3 + F_2O_3)], 1.6 to 2.6% MgO, and 0.28 to 3.29% CaO. Recommended limits are dynamic shear stress, τ_0 = 50 to 200 Pa; viscosity, n = 0.02 to 0.07 Pa sec; static shear stress, θ = 150 to 600 Pa; and plasticity strength, P_m, of the structure one minute after preparation equals 150 to 500 Pa. A considerable amount of effort has been placed on developing this integrated grouting approach. In saturated fractured environments, it has been shown to be very effective in reducing groundwater flow.

4.3 Grouting Methods

4.3.1 General

Selection of grouting methods and procedures requires an understanding of the following geotechnical considerations defined by Houlsby (1990):

- spacing of the joints,
- size of the joints,
- direction of the open joints,
- rock strength,
- rock soundness,
- rock stresses,
- uniformity, and
- proneness to piping.

The most important geotechnical considerations are the spacing (and orientation) of the joints and the aperture of the joints. While there is a broad variation in the strike and dip of the joints at Yucca Mountain, there are two predominant joint sets that strike N12°W and N34°E. Their fractures dip predominantly vertically or near vertically. Apertures are assumed to range from tens of microns to >1 cm. Because welded tuff is highly fractured,

the grout is expected to be fairly uniformly emplaced around the injection hole, with some preferential emplacement along the direction of the strike if the fracture sets.

Because of the potential for encountering fractures of several different apertures, four grouts are proposed for usage for the YMP Repository Sealing Program. For fine fissures with apertures of ~10 to ~100 micrometers (μm), a microfine cement (MC-500) having an initial water/cement ratio of 2:1 by weight is recommended. MC-500 is an inorganic grouting mixture composed of microfine particles, with 50% of the particles <4 μm and the largest ~13 μm (Geochemical Corporation, 1990). For apertures 100 μm to 1 mm, the water/cement ratio of the microfine cement grout should be halved to achieve a higher strength. For fissures with apertures between 1 mm and 1 cm, the use of a neat Class A portland cement grout is proposed. For fractures with apertures greater than 1 cm, the addition of silica sand is proposed together with a Class H cement. A small amount of aqua gel (a fine bentonite clay) is recommended to reduce settling of the cement grains and sand particles.

4.3.2 Equipment

Grouting equipment includes drilling equipment, mixing equipment, circulation equipment, and hole fixtures. This discussion emphasizes equipment for cement grouting of rock formations; specialized equipment for chemical grouting in soils is less applicable to repository grouting and will not be treated here.

Quality drilling equipment is extremely important to the success of a grouting program. In rock, drilling may be performed by rotary, rotary percussive, or downhole hammers. Early specifications generally called for rotary drilling methods or diamond drilling to ensure that fractures did not become clogged with fine drill cuttings that may be caused by percussive drilling. However, Deere (1982) reports that this criterion was beginning to be relaxed, although the general philosophy of avoiding fracture clogging by water flushing is still important. Houlsby (1990) suggests inspecting drill cuttings to see if they appear water-flushable prior to selecting the drilling method. If the chips are hard and angular, Houlsby feels that rotary percussive drilling could be specified to optimize costs. Bruce (1989) reports on a recent dam project where the use of air-powered downhole hammers, which minimize hole deviation, is being permitted.

Mixing equipment includes mixers and agitators, while circulating equipment includes pumps, circulation lines, and hole fixtures. Figure 4-3 shows a typical grouting system.

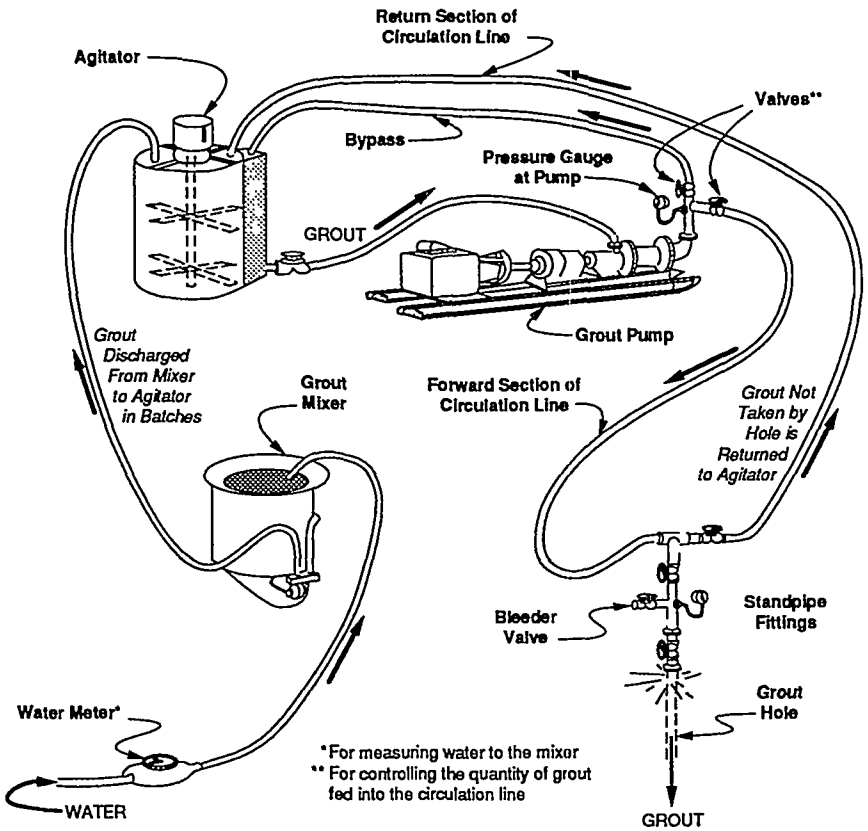


Figure 4-3
Typical Grouting System (after Houlsby, 1990)

High-speed colloidal mixers are recommended for cement grouting, because they improve the penetrating characteristics of the grout by mechanically separating clumps of particles and completely wetting individual particles. Agitators are necessary to prevent settling of the grout after mixing during temporary storage prior to injection.

Pumps have improved dramatically since the early days of grouting. Current practice involves the use of helical rotor (mono) pumps for most grouting duty (Figure 4-4). These pumps are available in pressures up to 15 psi. Advantages of this type of pump include low maintenance and a steady pressure output. Positive displacement pumps may be required for extremely high-pressure grouting (>150 psi), but these are high-maintenance items that deliver a pulsating pressure. Circulation lines must be of limited diameter to maintain adequate velocities and to minimize line clogging. Fixtures at the grout hole include the header assembly (standpipe fittings on Figure 4-3), standpipe (if used), and any downhole equipment.

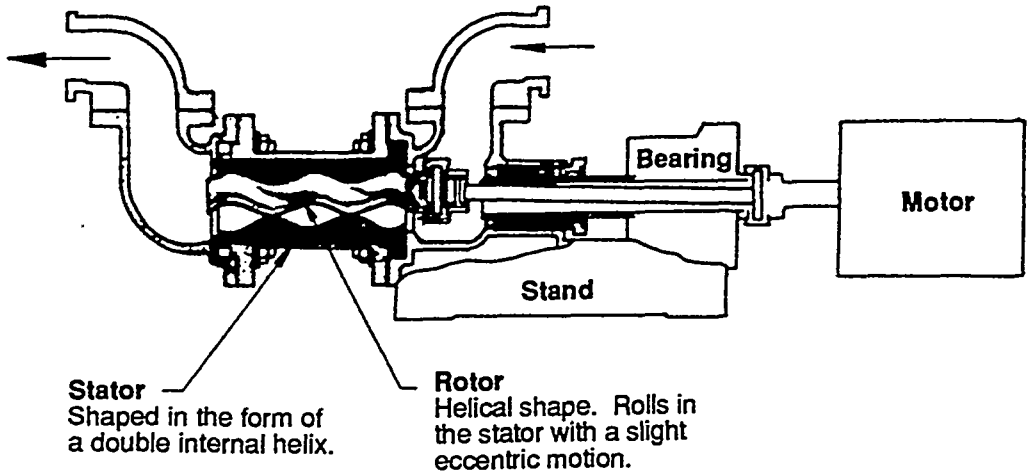


Figure 4-4
Helical Rotor Pump (after Houlsby, 1990)

4.3.3 Procedures

4.3.3.1 Fractured Rock Masses

Procedures for grouting fractured rock masses are relatively mature and are shown in Figure 4-5. The selection of a grouting method is dependent on the nature of the formation to be grouted and especially on the stability of the grout hole itself. Basically, grout holes may be drilled and grouted in stages from the collar down, with or without packers to isolate previously grouted zones; or they may be completely drilled and grouted in stages from the bottom up using packers. In some cases, circuit grouting may be required, where a pipe extending to the bottom of the hole allows grout to be recirculated. Circuit grouting was formerly used extensively on TVA projects, but was never entirely satisfactory. Bruce (1989) reports on a recent technological innovation for grouting very difficult rock conditions (collapsing holes, large cavities), the multiple packer sleeved pipe (MPSP) method (Figure 4-6). This method involves first drilling and casing the hole. A tube with multiple permanent packers is inserted into the hole, and the casing is withdrawn. The packers are

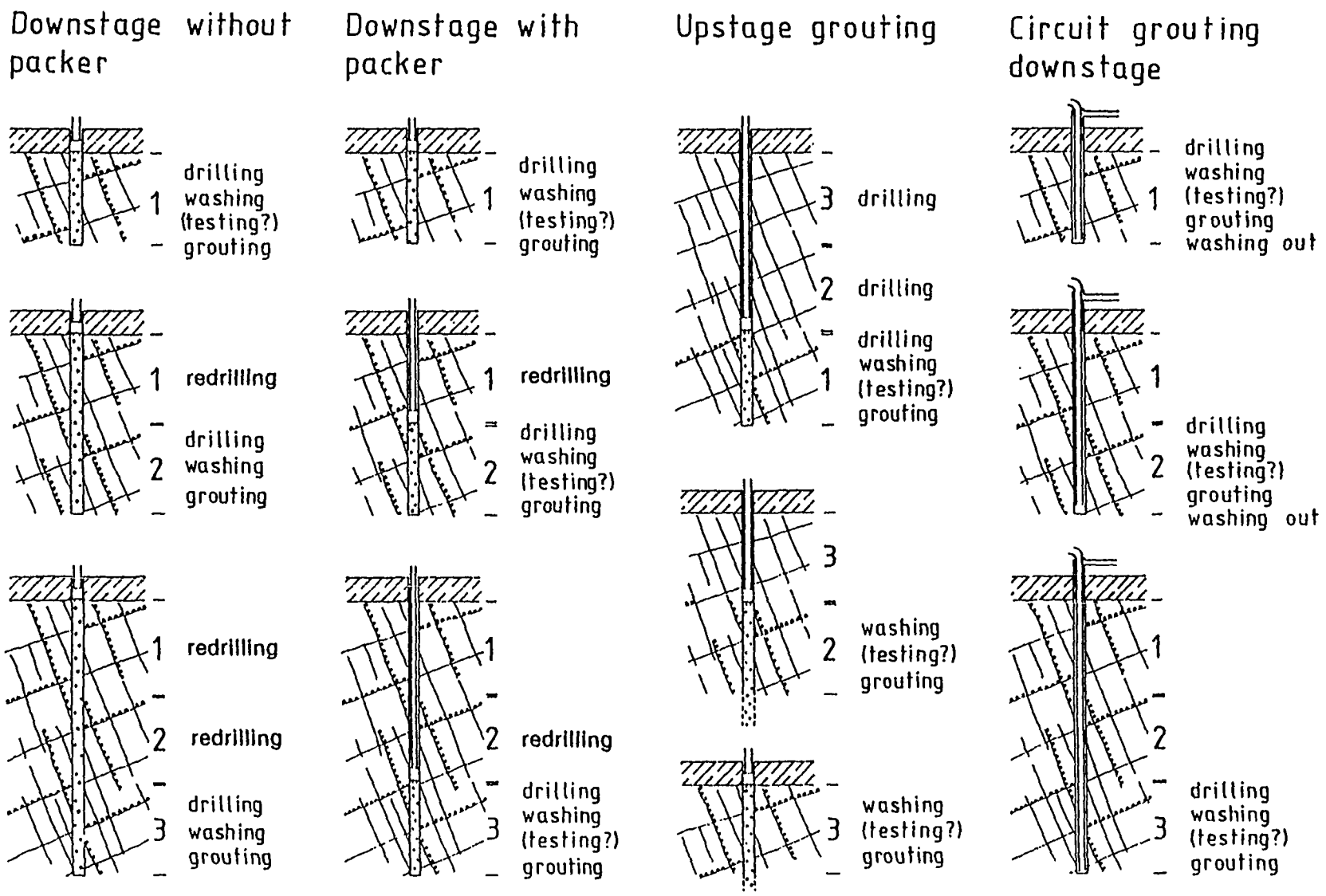


Figure 4-5

Procedures for Grouting Fractured Rock Masses (after Ewart, 1985, p. 333)

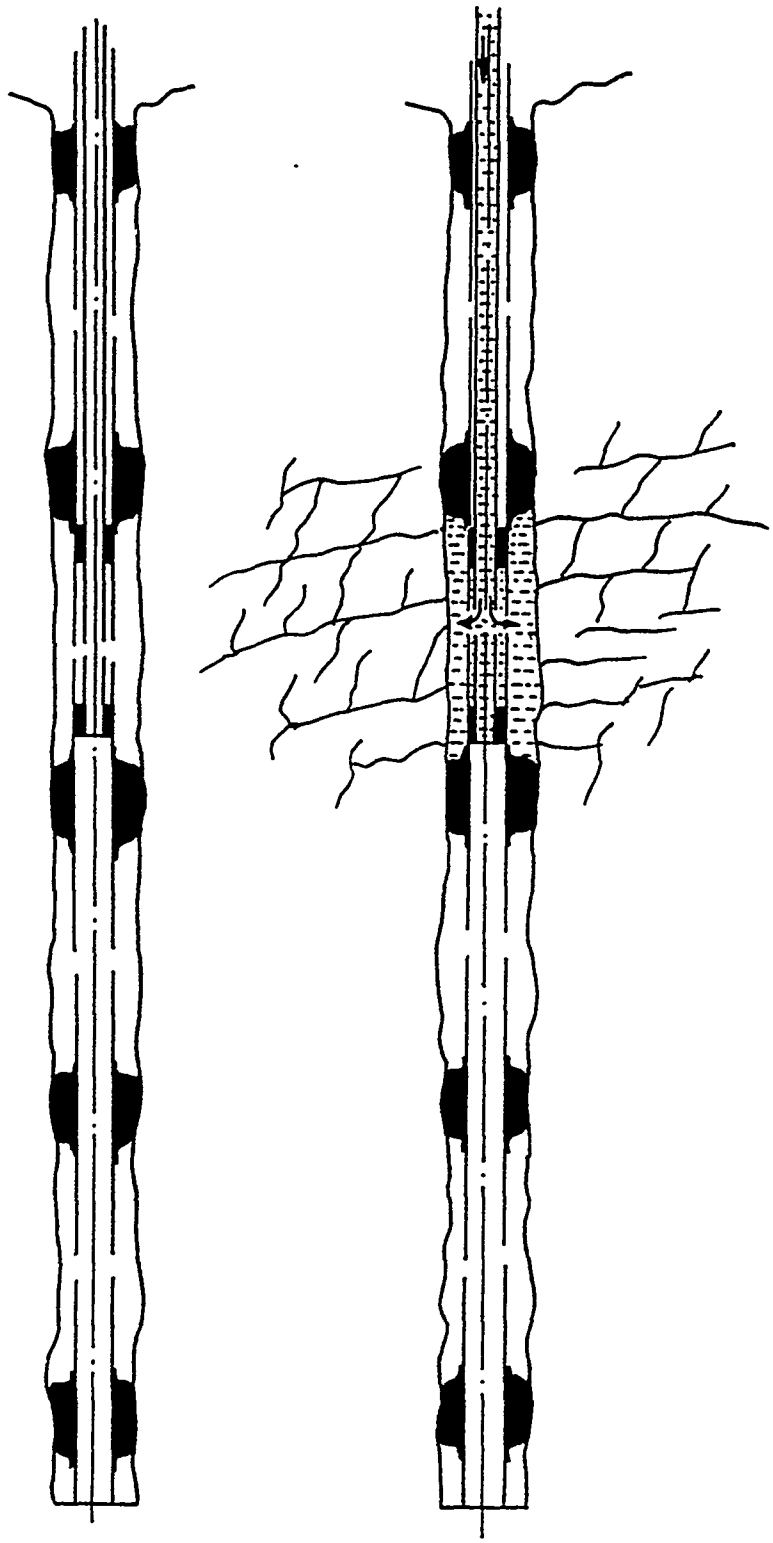


Figure 4-6
Multiple Packer Sleeved Pipe (MPSP) in Place (left); Single Zone Being Grouted (right)

permanently grouted in the hole using cement. The zones between packers then can be treated individually with little concern about hole stability.

Drilling patterns vary depending on the specific application. Closure grouting is the normal practice where quality is important. Closure grouting practice involves drilling widely spaced primary holes, grouting to refusal, then drilling intervening secondary holes, pressure testing them and grouting them to refusal. Tertiary, quaternary, and split-spacing holes may be required until the ground is tight. A case history of closure grouting for the New Waddell Dam is described in Appendix B.2. In this type of program, the primary holes form an important part of the site investigation program for grouting purposes. Nonclosure grouting may also be used, where holes are drilled on a predetermined pattern, in cases where quality control is less stringent and speed is essential. Special grouting patterns are often employed for grouting around shafts, where sealing against water inflows is particularly important if sinking operations are to be successful (Figure 4-7).

An important aspect of cement grouting of fractured rock is determining the proper water/cement ratio. Typically, a relatively thin mixture is used initially to grout the finer fissures, followed by a progressively thicker mixture. Initial mixes as thin as 12:1 were formerly used, but because of questions of durability with such thin mixes, typical current practice calls for a 3:1 initial water/cement ratio, finishing with a 0.8:1 mix.

Pressures for grouting vary. An important consideration in standard U.S. grouting practice is to provide adequate pressure to cause good penetration without excessively hydrofracturing the rock (Figure 4-8). European practice may involve hydrofracturing the rock and grouting the fractures for some applications.

4.3.3.2 Contact Grouting

For sealing cement bulkheads used to control water in underground openings, contact grouting is nearly always required. Contact grouting involves drilling into the interface between the cement plug and the rock and pressure grouting. Contact grouting is discussed in the chapter on bulkhead technology. A case history involving contact grouting is presented in Appendix C.3.

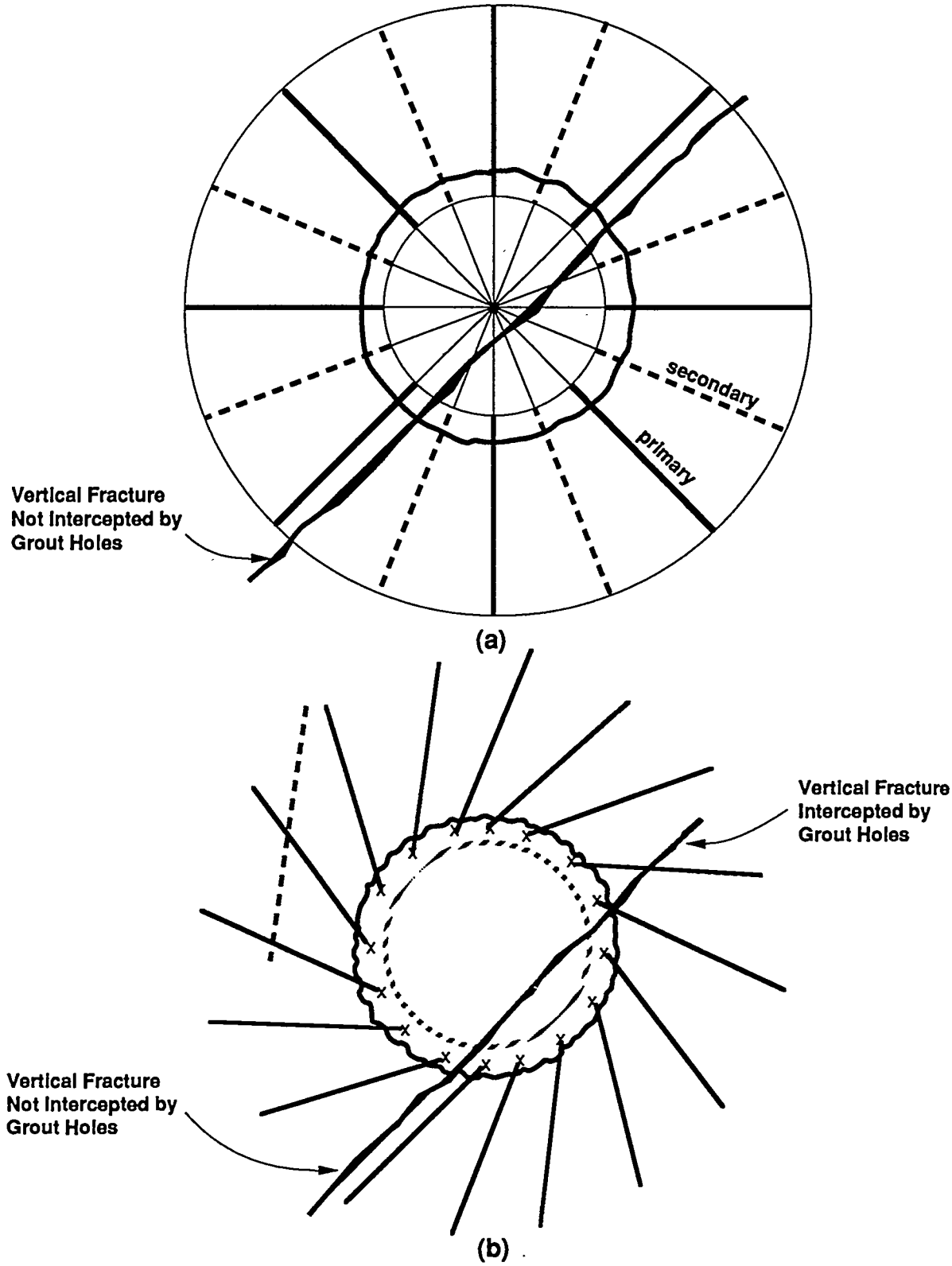


Figure 4-7
Grouting Pattern for Shaft Sinking

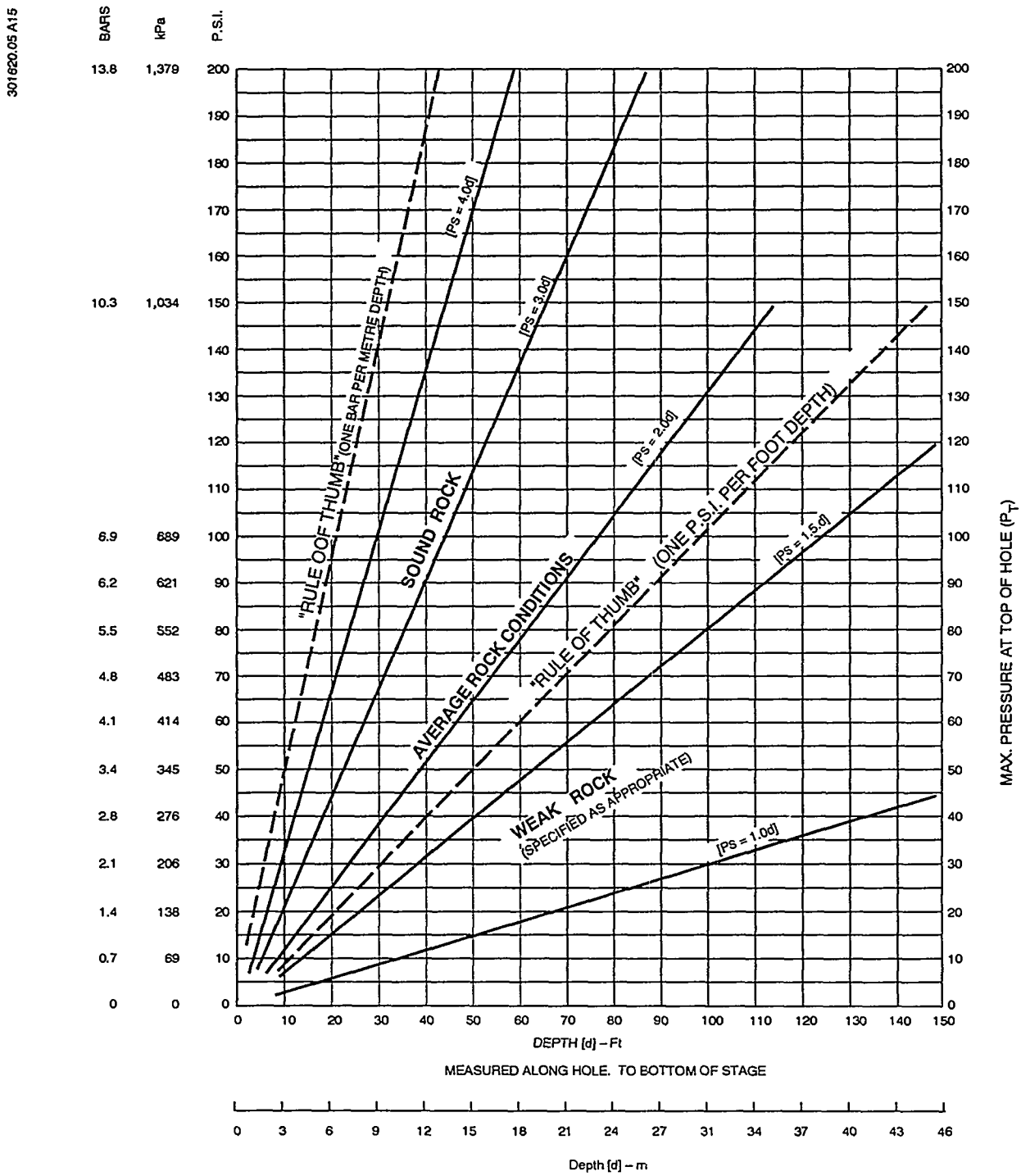


Figure 4-8

Allowable Pressures for Normal Grouting Conditions (Houlsby, 1990)

5.0 Application of Available Technologies to Repository Sealing

In the preceding sections, repository sealing components have been introduced, and available technologies for emplacing similar components from the mining and civil fields have been described. In this section, the available technologies will be applied to emplacement of seals in a hypothetical repository based on the design of record, discussed in Section 1.2.4.

5.1 Work Environment

The work environment during repository decommissioning will influence the methods used for seal component placement; temperature and radiation are the principal factors affecting sealing.

Decommissioning activities will begin following the caretaker period, approximately 25 years after waste emplacement operations are completed, and as many as 50 years after the first waste emplacement. By this time, the waste emplacement drifts and some other repository areas will be experiencing elevated rock temperatures as high as 120°C. Many areas, including waste emplacement areas, will have been isolated from the repository ventilation circuit using temporary bulkheads to reduce the ventilation overhead during the caretaker period. Ambient air temperatures will be close to the rock temperature in these areas.

Prior to reentering abandoned emplacement areas for backfilling, they will first have to be cooled to acceptable working temperatures. Environmental criteria proposed for the SCP-CDR (SNL, 1987, pp. 3-130) include an air-cooling power greater than 500 watts per square meter and a dry-bulb temperature less than 40°C "in areas where mining, drilling, maintenance, or retrieval of waste are occurring." According to analyses conducted for the SCP-CDR, achieving these conditions 50 years after waste emplacement would take 37 days of cooling each drift at a refrigeration load of 201 tons of refrigeration, using air cooled to 10°C (SNL, 1987, pp. 3-119). To date, cooling analyses have not been published that explicitly consider backfilling scenarios. However, with existing repository layouts, the presence of fill in the drifts will complicate ventilation considerably, requiring long runs of auxiliary ventilation tubing.

Placement of seals in a hot environment may require special procedures to ensure quality control. Desiccation and cracking can occur in clay components; cementitious materials may require special procedures for placement. Adverse working conditions will also have an effect on quality control due to worker discomfort. In-drift waste emplacement schemes are likely to produce much higher ambient temperatures for backfilling than borehole emplacement.

Some of the newer repository design options, such as in-drift emplacement, may also involve a significant radiation risk. If acceptable environmental conditions are to be maintained during closure operations, considerable planning will be required.

5.2 Preconditioning

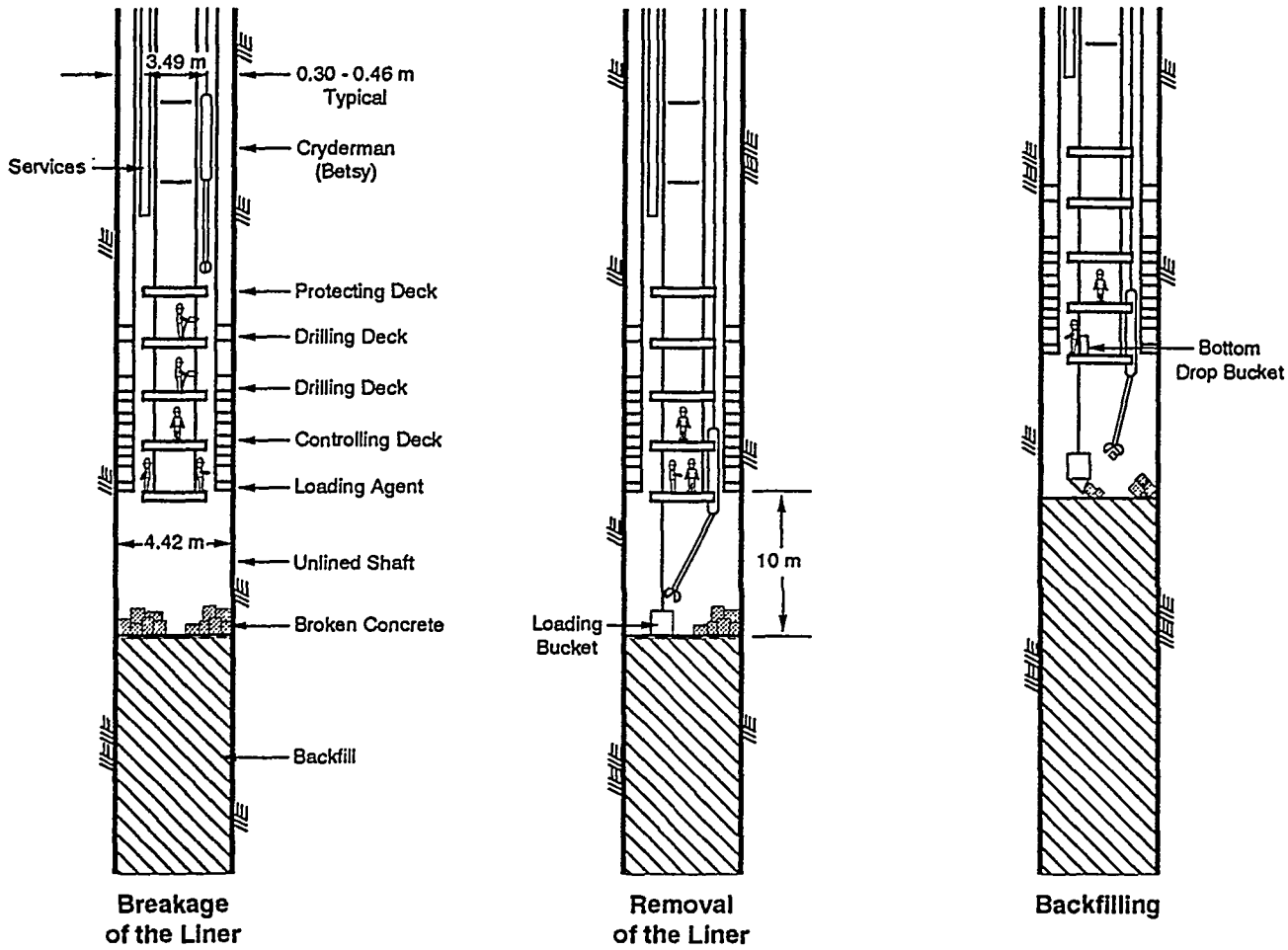
Some of the materials discussed in Section 1.2.4.1 and listed in Appendix D may require removal during decommissioning. In some instances, global removal may not be required, but local removal at specific seal locations may be performed. Deciding which materials will require removal depends on geochemical considerations that are not final at this time and on simple interference with sealing components.

Case histories of specific technologies for preconditioning are not available unlike the case for placement of seal components such as backfill, bulkheads, and grout. Therefore, the preconditioning concepts presented here are not taken directly from case histories, but are based on existing shaft-sinking and mining technology. In particular, complete removal of ground support (concrete liners, rockbolts, etc.) is not performed in the civil or mining industries.

Attention to ground control is a very important consideration when removing ground support. Alternate support, such as backfill and/or hydraulic jacks, must be provided near the removal operation. The operator must be protected from rockfall by means of a special canopy or shield and must retreat toward the mine portal, always remaining under supported roof. In poor ground conditions, the operation may be very difficult, and the opening itself could collapse if substantial temporary support cannot be provided.

5.2.1 Concrete Liner Removal

Figure 5-1 shows a possible sequence for concrete liner removal in a shaft. A multideck shaft-sinking stage (Galloway) will be used as a work platform. Broken concrete falls to the backfill surface, where it can be handled using a Cryderman mucker or, alternatively, a



Notes

1. Breakage Shown is by the Nonexplosive Demolition Agent
2. The Production Cycle is Applicable to the Handheld Pneumatic Breaker, Drill and Blast, and Hydraulic Splitter Methods
3. By Using Handheld Pneumatic Splitters, the Liner is Broken from the Top Down over 10 m.

Figure 5-1

Production Cycle for Breaking and Removing the Liner and Placing the Backfill

Cactus Grab (Figure 5-2). Additional grout-impregnated rock behind the liner may also be removed as required.

Figure 5-3 shows a typical sequence for concrete liner and floor removal in a drift. In this sequence, a roadheader is used to remove the liner, and the floor is removed manually using a pavement breaker. In Figures 5-4a and b, alternate methods for liner removal are shown, and in Figures 5-4c and d, alternate methods for floor removal are shown.

Although liner removal is a nonroutine operation, the machines and equipment shown in Figures 5-1 through 5-4 are currently available technology. At this time, it has not been determined that shaft or drift liners will require removal.

5.2.2 Rockbolt Removal

Figures 5-5 through 5-10 show methods for rockbolt removal. Pull-testing individual rockbolts is a routine operation commonly performed in underground mining operations to test anchorage strength, and special equipment is currently commercially available for this work. Rockbolts to be tested may be fitted with a special pull collar prior to pull testing, as shown in Figure 5-5, and may not be completely removed during testing. Resin-grouted bolts are rarely completely removed during testing.

Large-scale removal of rockbolts is a nonroutine operation, especially after a lengthy service period in a hot environment. If these bolts are corroded or have not been fitted for removal, special equipment may be required. The equipment for removing bolts shown in Figures 5-6 through 5-10 is not available technology, although it would not require much development from existing equipment.

Figure 5-6 shows a grapple puller. This exerts force behind the plate to remove the bolt. However, complete removal may not be achieved in all cases using this method. Jacks would be required adjacent to the pulling operation.

Figure 5-7 shows a suggested procedure for removing rockbolts. A machine-mounted impact wedge is used to break off the bolt head and plate; it can also remove mats, wire mesh, and other external ground reinforcement components. Alternately, a grapple puller could be used. After chipping out the collar of the hole, a special tool consisting of cam locks, a hydraulic cylinder, and a reaction collar is used to grasp the broken bolt and pull it out of the

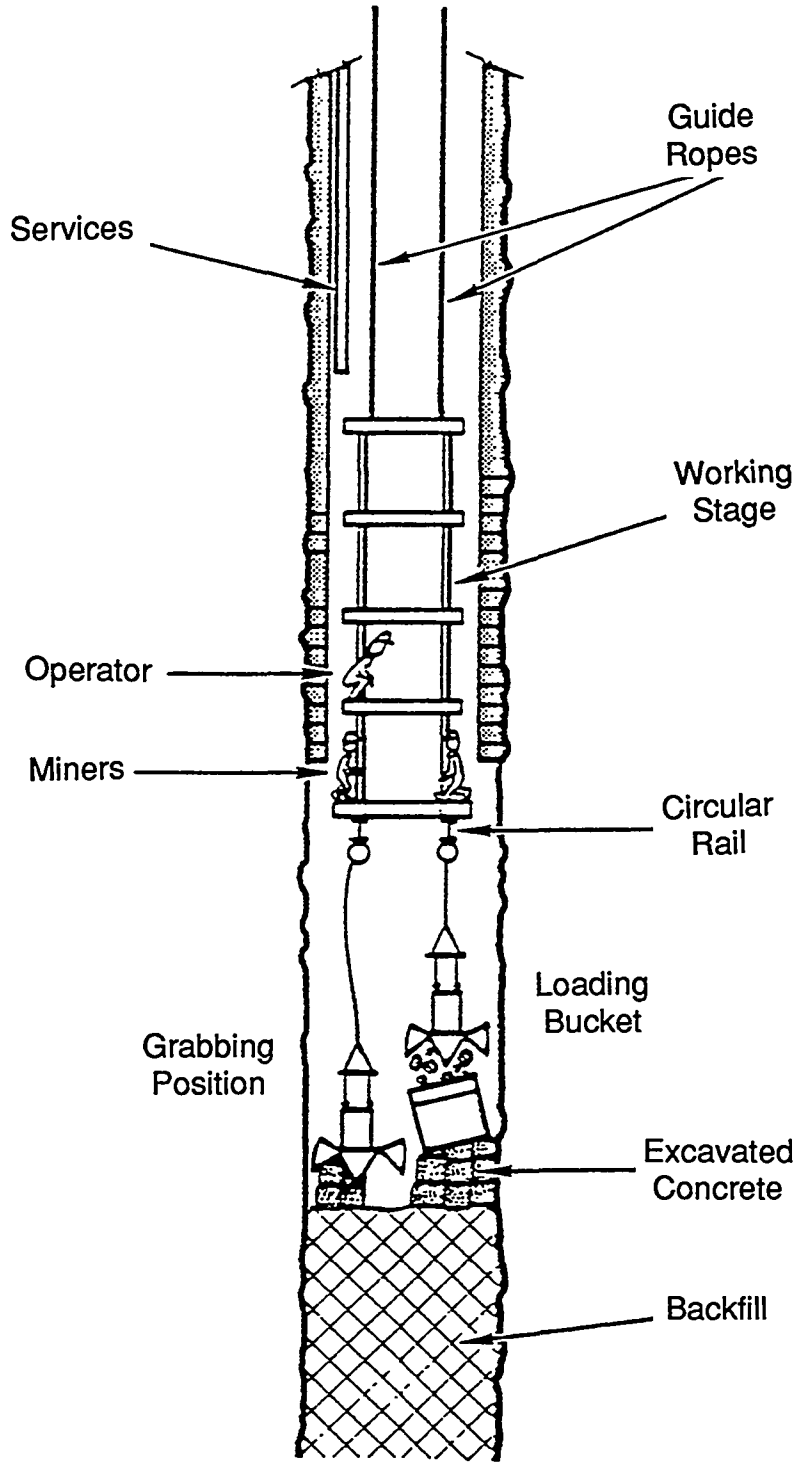


Figure 5-2
Removing Pieces of Concrete Using the Orange-Peel-Grab Unit

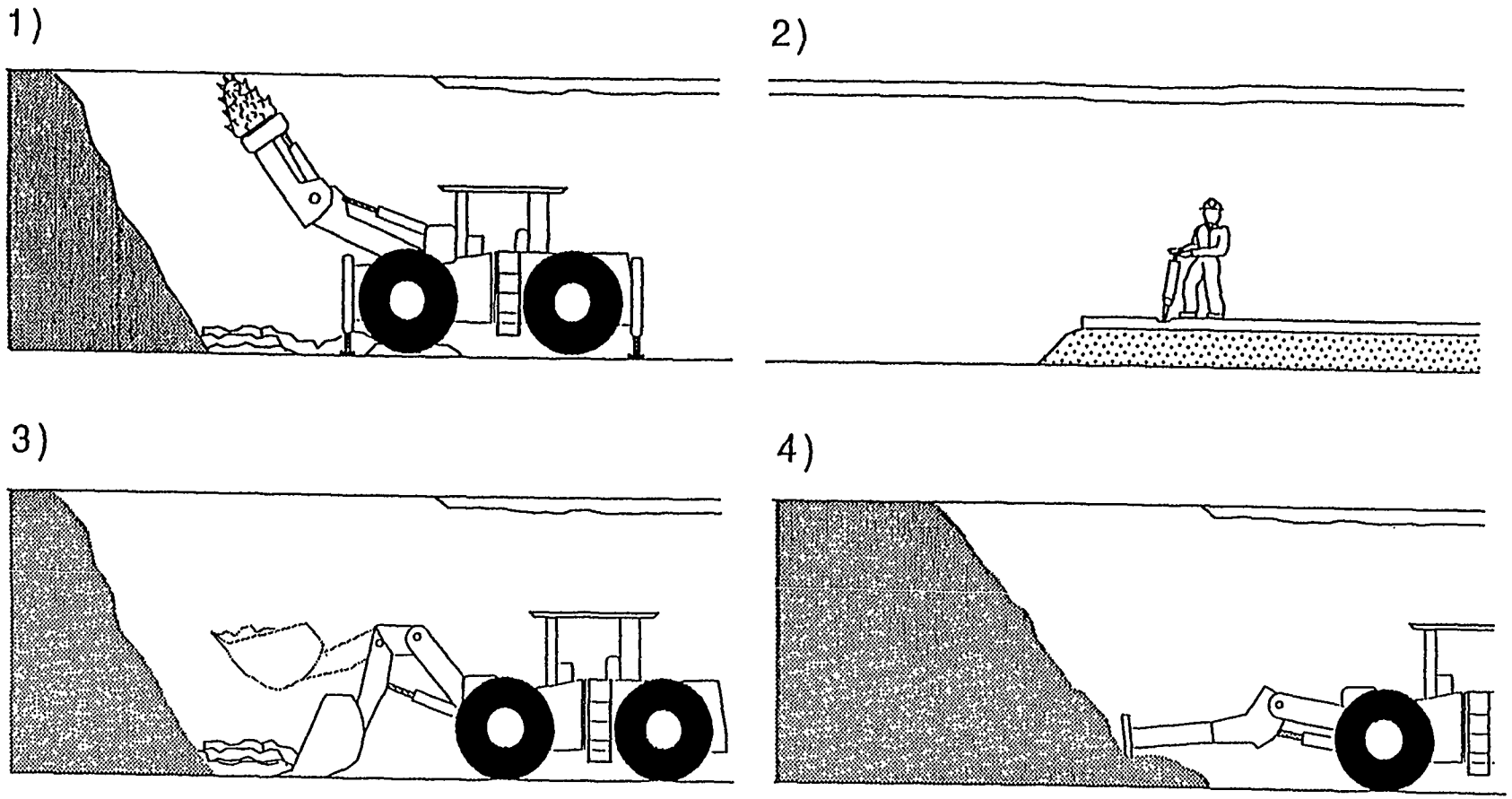
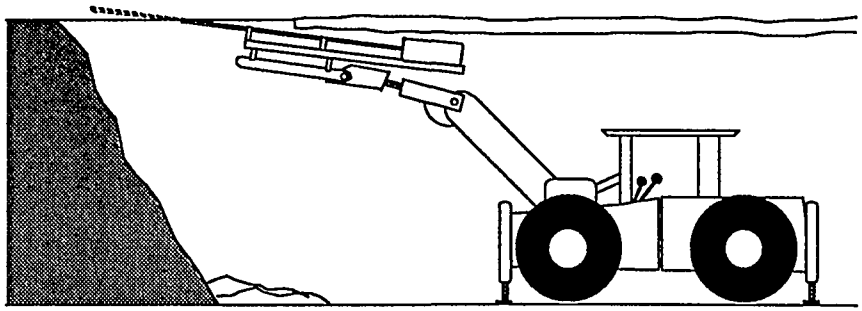
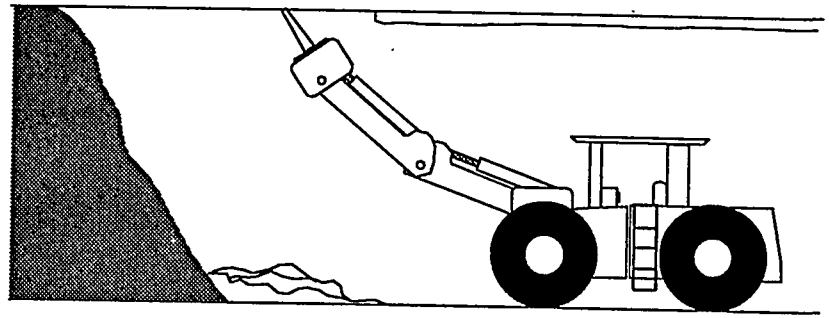


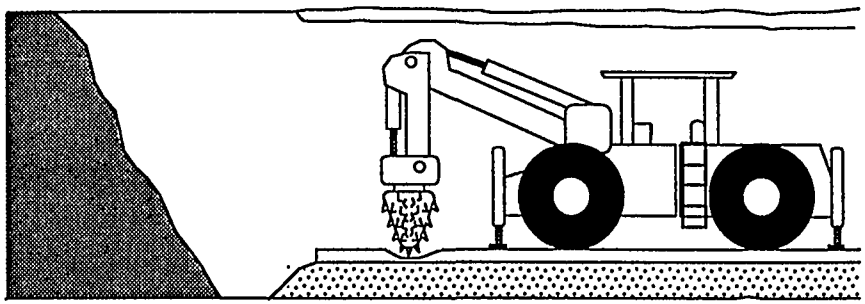
Figure 5-3
Typical Sequence for Concrete Liner and Roadway Removal in a Drift



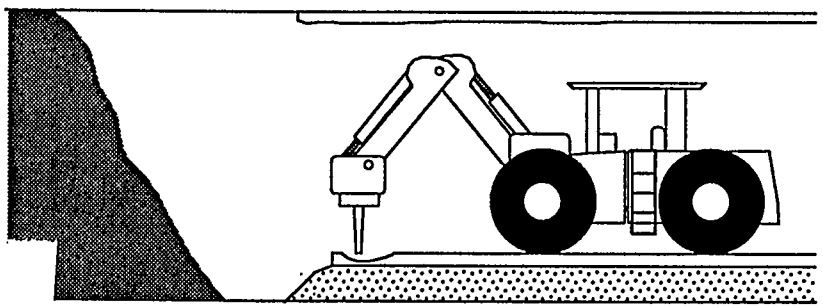
a) Drilling and Splitting



b) Impact Breaker



c) Roadheader



d) Impact Breaker

Figure 5-4
Alternate Methods for Liner Removal (a, b) and Roadway Removal (c, d)

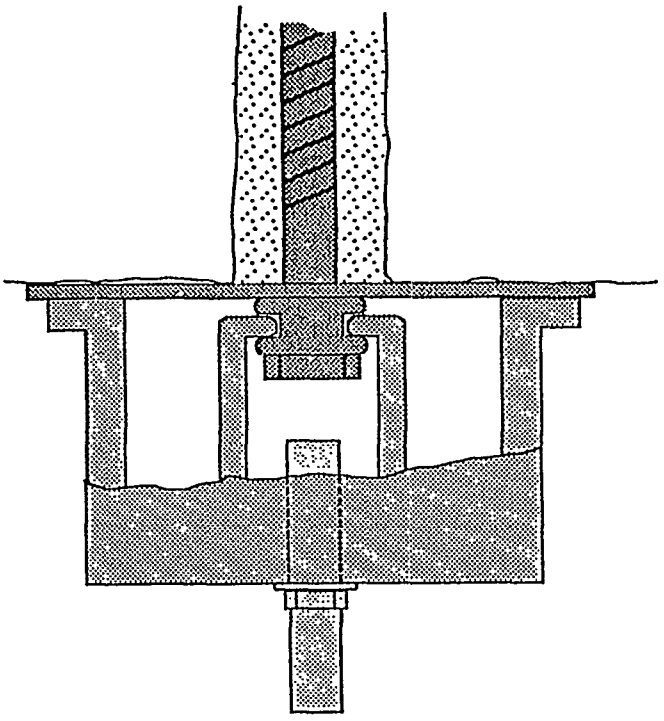


Figure 5-5
Existing Method of Removing Bolts with Preinstalled Pull Collar and a Crow's Foot

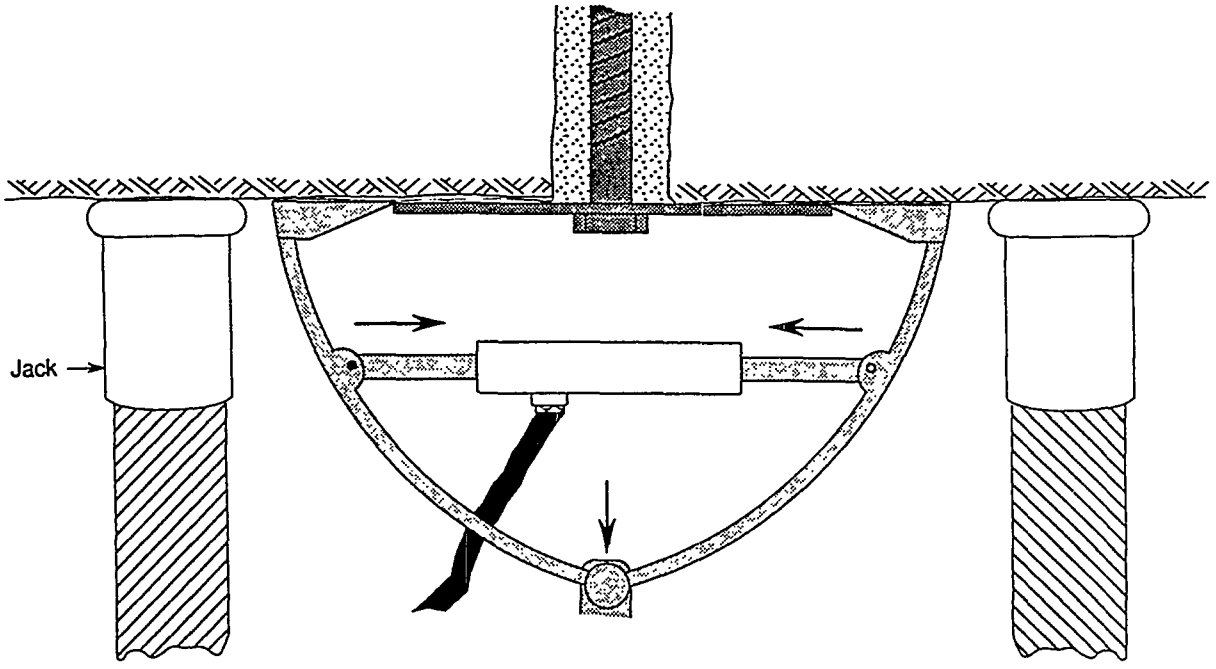


Figure 5-6
Concept for Removing Grouted Bolts Using Grapple Puller

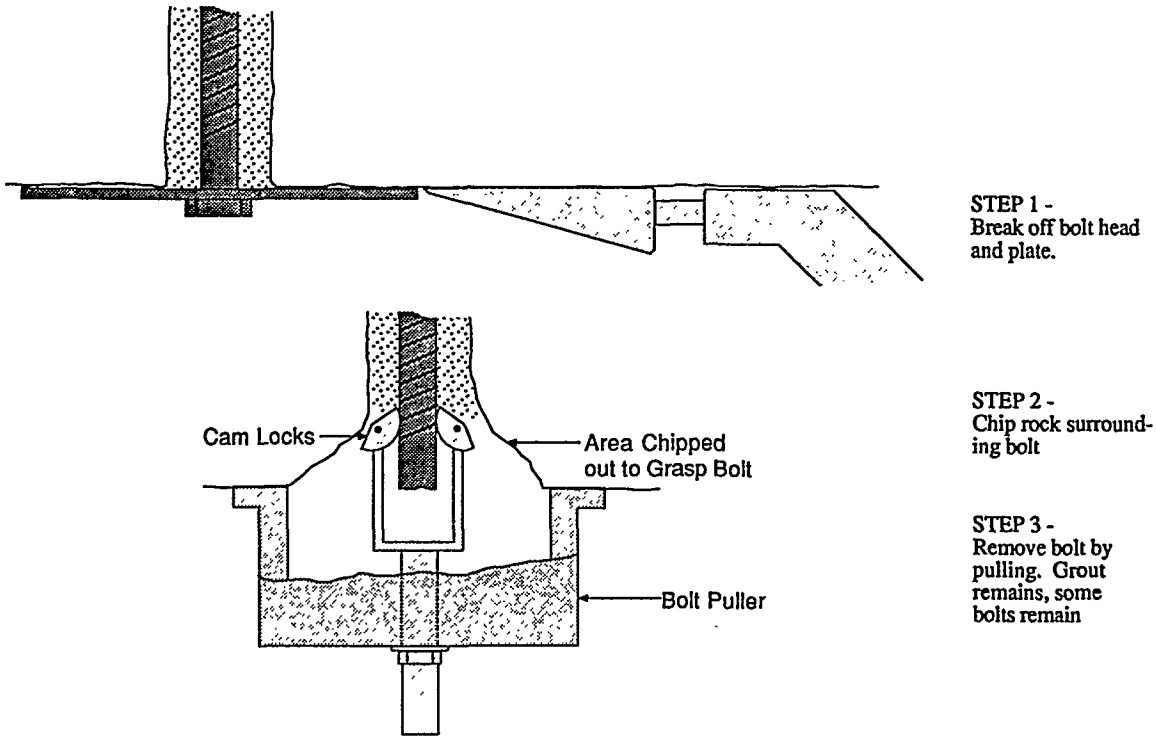


Figure 5-7

Concept for Removing Grouted Rockbolts Using Cam Locks

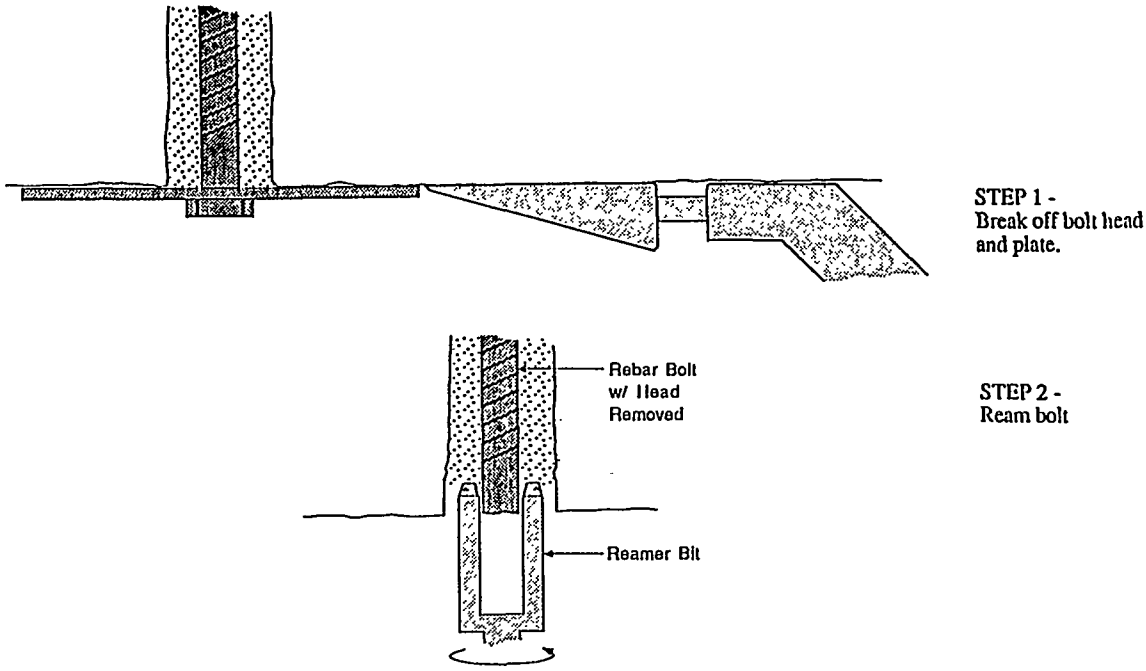
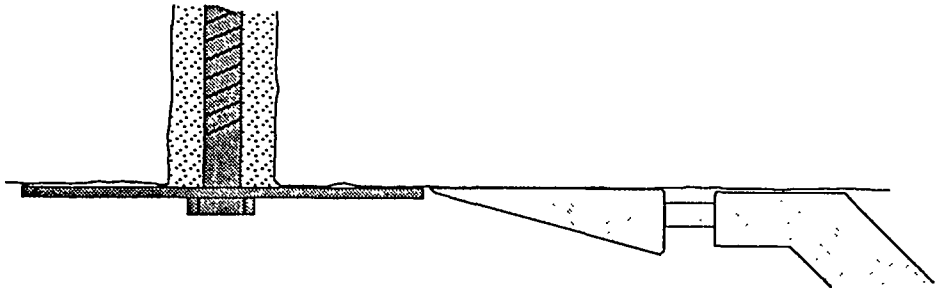
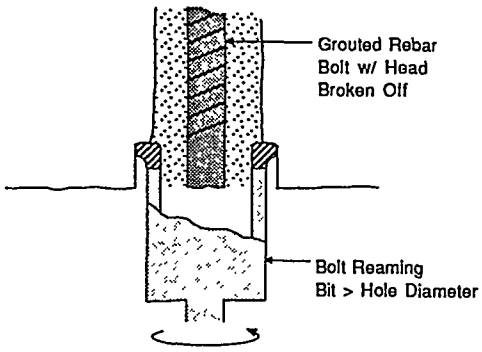


Figure 5-8

Concept for Removing Grouted Bolts Using Reaming Bit

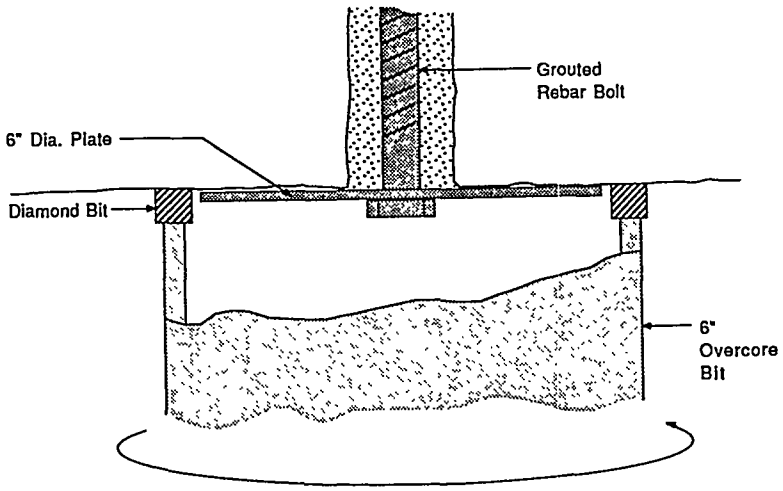


STEP 1 -
Break off bolt head and plate.



STEP 2 -
Bit is used to cut grout and ream hole to remove grout completely

Figure 5-9
Concept for Removing Grouted Rockbolts Using Large-Diameter Reaming Bit



OVERCORING -
Bolt Plate and Grout

Figure 5-10
Concept for Removing Grouted Rockbolts by Overcoring

hole. It is possible that the entire bolt may not be removed by this procedure, especially if it is a fully grouted bolt. A more complete method of bolt removal using overcoring is shown in Figure 5-10. In this case, a special reamer bit is used to chase the bolt once the head is removed. The system becomes more robust as larger-diameter bits are used, at the expense of leaving a larger hole (Figures 5-9 and 5-10).

The overall operation of rockbolt removal must be considered a special operation that may require special techniques and possibly the development of remote control equipment for safe and efficient completion. At this time the need for global removal of rockbolts has not been established.

5.2.3 Postclosure Drainage Area

Figure 5-11 shows the decommissioning plans for the postclosure drainage area (see Figure 1-2). The pump station is decommissioned, and the old sumps are used as a first-stage sedimentation pond. The concrete block floor and underlying gravel are removed. If required, the floor is cleaned and treated to enhance its permeability. This could be done using compressed air and mechanical brushes, perhaps after a pass with a roadheader or mobile miner to remove surface rock. The floor in the drainage area may also be perforated by drilling to enhance its vertical permeability, or a slot could be cut with a diamond saw. The technologies used here are considered available technologies, although their effectiveness requires field demonstration.

5.3 Seal Component Placement

5.3.1 General Fill, Drifts and Ramps

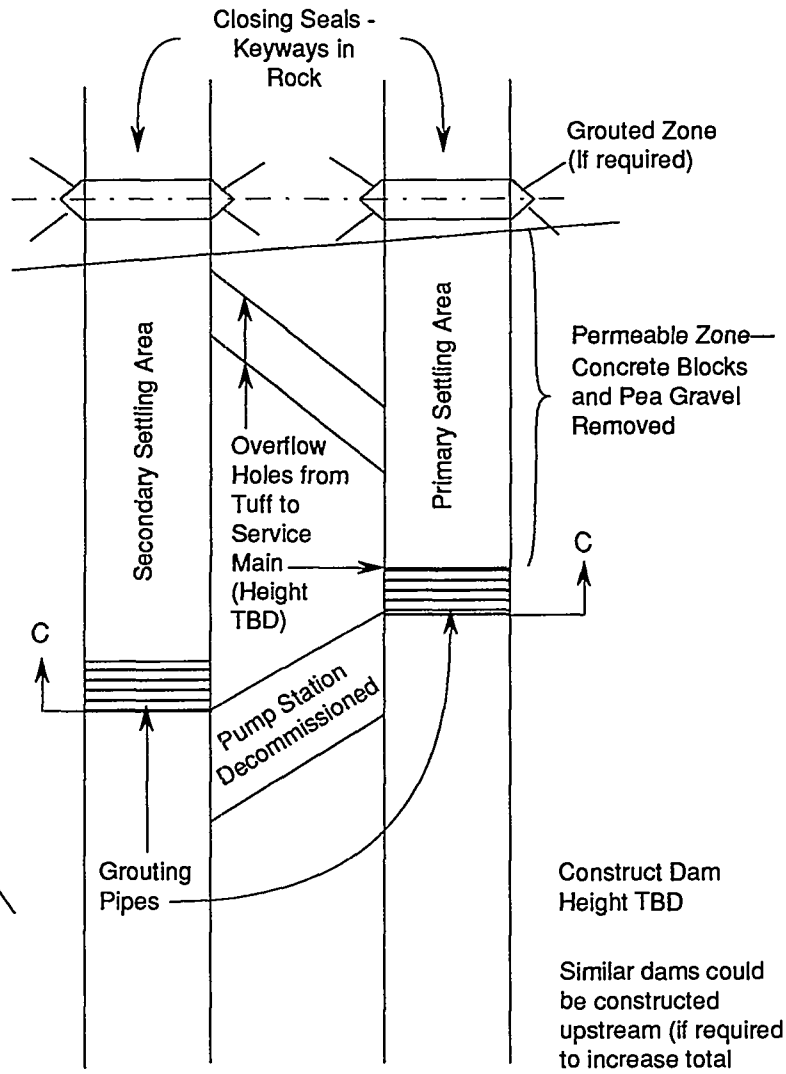
5.3.1.1 Criteria

In Chapter 2, five main methods of placing fill in underground mines are presented: hydraulic, mechanical, throwing (pneumatic and slinging), gravity, and manual. Combinations of these methods are also commonly used.

The criteria for selecting a method for emplacing general fill for the repository are that it

(1) minimizes water use for fill placement (must),
(2) uses existing openings for fill transport (must),

1. Pump stations are decommissioned.
2. Sumps could be used for sedimentation purpose.
3. Decommissioned pump stations equate water level in parallel mains.
4. Concrete blocks and pea gravel above permeable zone are removed. Ground is cleaned to restore original permeability.
5. Dams are built as required to increase settling area.
6. Closing seals are built.



Similar dams could be constructed upstream (if required to increase total settling capacity).

Section C-C

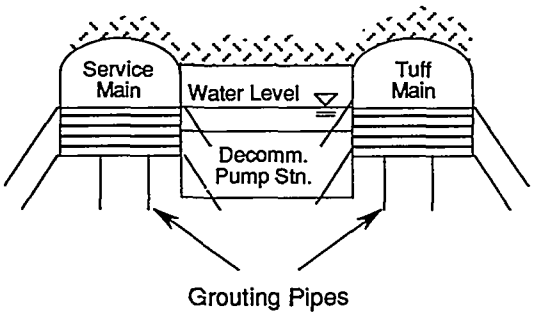


Figure 5-11

Decommissioning of Postclosure Drainage Area (see Figure 1-2)

(3) has the capacity to place large volumes of fill (must),
(4) facilitates tightfilling,
(5) minimizes degradation/fines production,
(6) enables compaction and quality control,
(7) minimizes cost and schedule, and
(8) provides maximum worker safety.

Of these criteria, only the first three are sufficiently established in the program to qualify as "musts" and can be used in an initial screening of methods. The other criteria are all important, but cannot be used as discriminators until fill specifications are developed.

5.3.1.2 Screening

Figure 5-12 shows the results of the initial screening based on the three "must" criteria, according to the following logic:

Criterion #1—Water Use. The unique advantages of an unsaturated-zone repository may be lost with the introduction of large quantities of water. Some amount of water may be required for fill placement and compaction, although the precise amount permitted has not yet been determined. It is unlikely that large amounts of water will be acceptable, such as the quantities required for hydraulic transport and placement of fill.

Criterion #2—Use of Existing Penetrations. Present plans call for a number of penetrations of various sizes into the repository for exploration and underground access, and a number of penetrations already exist. These penetrations will all require sealing as part of repository closure. It makes little sense to create additional vertical penetrations from the surface solely for backfilling underground drifts at closure. Studies (Fernandez et al., 1987) show that vertical penetrations have far more significance to the total system performance than horizontal openings. This criterion would eliminate gravity filling, as practiced in the mining industry, as a primary means of backfilling, because the considerable horizontal extent of the repository would require vertical or steeply inclined holes at numerous locations to deliver fill.

Criterion #3—Placement Capacity. The requirement for a large placement capacity stems from the large excavated volume of the repository. It is estimated that

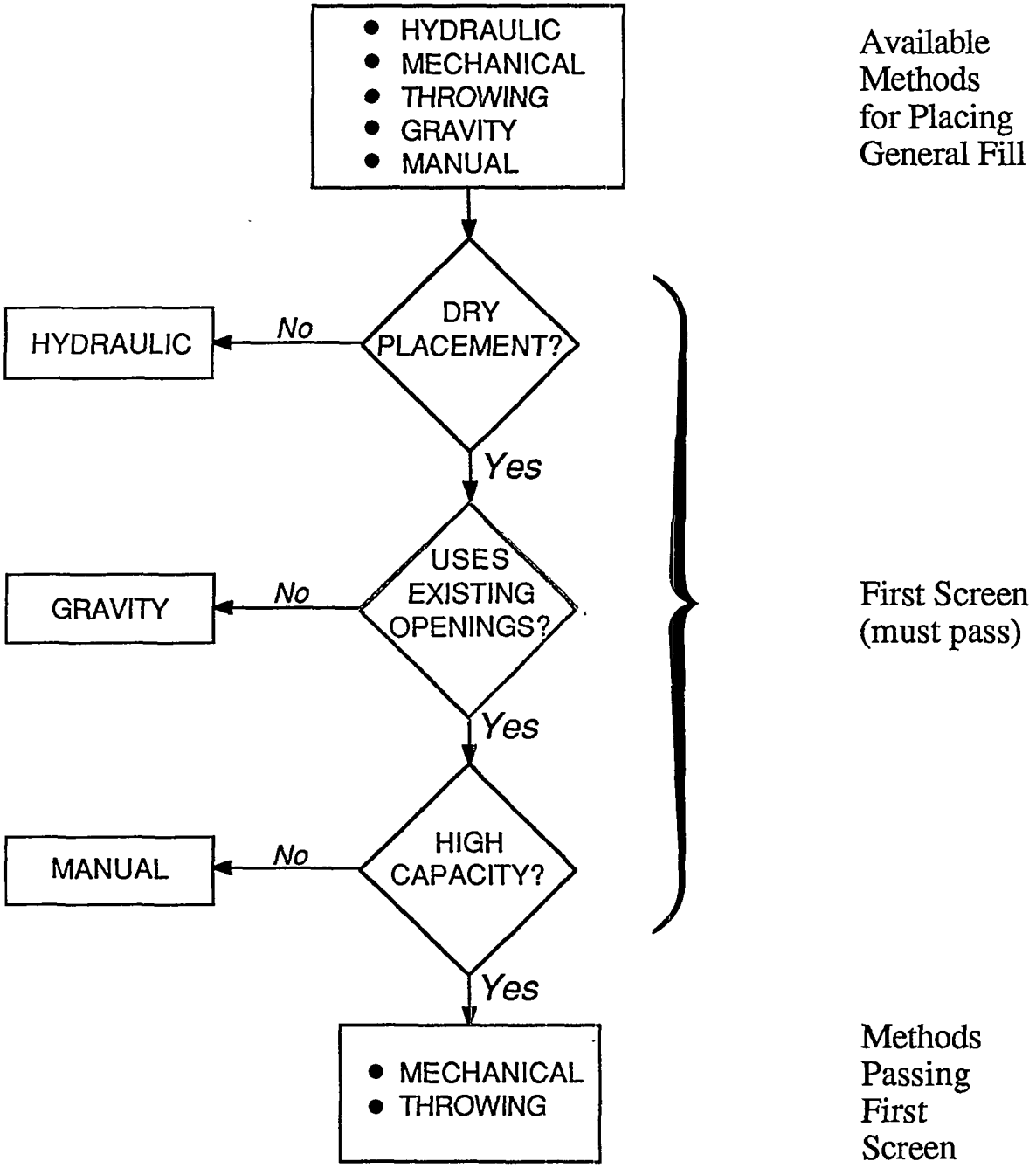


Figure 5-12
First Screen for General Fill Structures

approximately 340 million ft^3 of material will be placed as general fill. Manual methods are eliminated by this requirement. Mining industry experience suggests that manual methods have a higher placement capacity than pneumatic methods, and may be favored for most general fill.

The only methods remaining after the initial screening are mechanical and throwing methods. Discrimination between these methods will depend on the respective factors established for the other five selection criteria. Assigning relative weighing factors will not be possible until important backfill strategy decisions are made and integrated with total system performance assessment, and the methods are tested with selected materials under field conditions. The following questions must be answered as part of the overall backfill strategy:

- Is tightfilling required?
- Are there obstacles in the drift?
- What in-place fill size gradation is required?
- What specific materials must be placed? Is cemented fill required? How much compaction is required?
- What are requirements for thermal conductivity?

Some general observations can be made at this time.

Tightfilling, or placing fill in close contact with the back, is considered available technology. Both mechanical and throwing methods can be used to tightfill. At the Cannon Mine (Appendix A.1), cemented fill was mechanically placed with a nominal 2-in. gap between the fill and the back; contact was achieved in many locations. Tightfilling becomes much more difficult if the back is irregular and much easier if the drift is inclined. At the American Girl Mine (Appendix A.7), excellent contact was obtained by placing cemented fill in steeply inclined drifts (22.5% grade) using rubber-tired equipment. Earlier attempts at tightfilling using pneumatic methods were less successful due to reported "turbulence" in the final gap. However, tightfilling was achieved at the Sullivan (Appendix A.4) and Apex (Appendix A.3) mines using pneumatic methods in locations where it was required. Excellent contact was achieved during pneumatic placement of uncemented fill for abandonment seals at Castlegate (Figure A.5-4). However, it should be noted that tightfilling with uncemented aggregate will be temporary because of settlement. Settlement can be greatly reduced by placing well-graded compacted fills, such as that commonly placed in civil construction; however, case histories were not available for placing such fills in underground mines.

All backfilling methods will cause some fill size degradation. Careful mechanical placement probably results in less degradation and size separation than pneumatic placement. Degradation and separation during pneumatic placement of crushed limestone have been studied by the USBM at the Lake Lynn Laboratory (Appendix A.9, see Section 5.4.1); similar studies are recommended using welded tuff.

Mechanical methods have a significant maintenance advantage over pneumatic methods when placing abrasive fills, especially materials with a high silica content. This is one reason why several mines have switched to mechanical methods after experimenting with pneumatic. Rotary air-lock feeder maintenance, pipeline wear, and wear at elbows were experienced at the Billie Mine and American Girl Mine (Appendix A.7). At the Lake Lynn Laboratory, promising prototype feeders less prone to wear have been developed, but have not been used by the mining industry. It is important that fill pipelines are very straight.

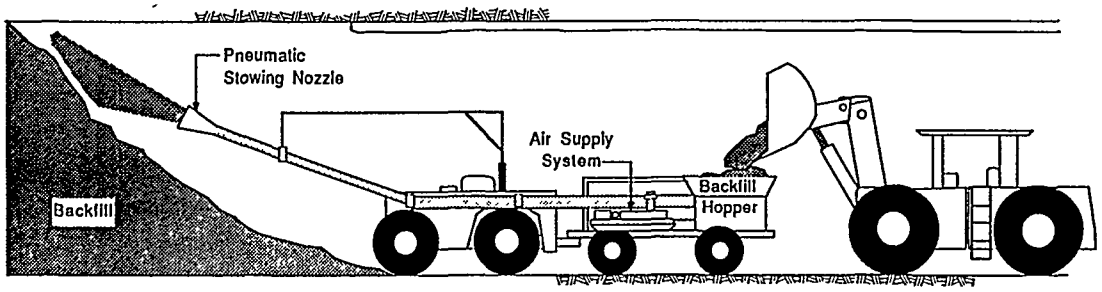
Mechanical methods are simpler and considerably more flexible than pneumatic methods. This is probably the main reason why rubber-tired methods are currently preferred by the mining industry. With pneumatic methods, it is difficult to transport fill over long horizontal distances, and difficult to change filling locations. These problems could be alleviated by feeding a mobile stowing unit at the face (Figure 5-13a).

If obstacles, such as waste containers, must be worked around during backfilling, then throwing methods may be required. This is especially likely if environmental conditions (temperature, radiation) are adverse at the filling location.

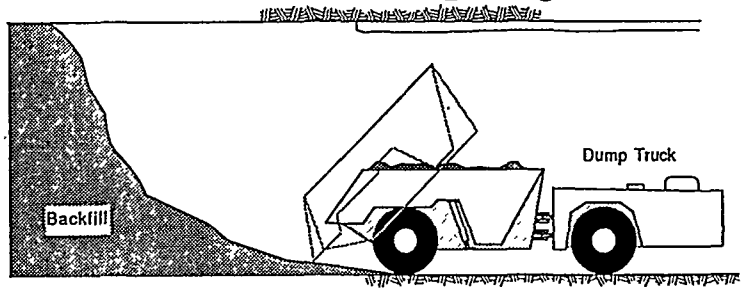
Based on the initial screening and use of qualitative criteria, it appears likely that a mechanical or combination mechanical-pneumatic method will be selected. For the general fill, the final screening will likely determine the relative proportion of mechanical and throwing methods, rather than eliminate one or the other method.

5.3.1.3 Suggested Sequence

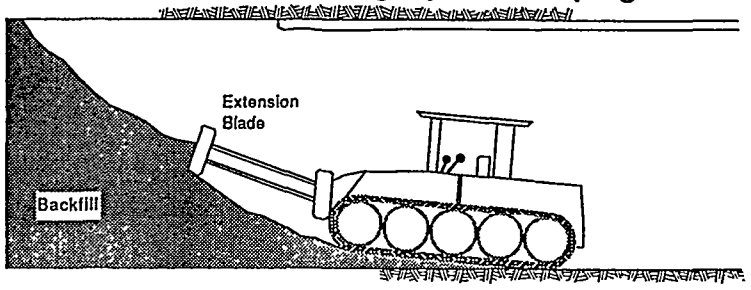
Figure 5-13 shows acceptable methods for mechanical and hybrid placement of general fill.



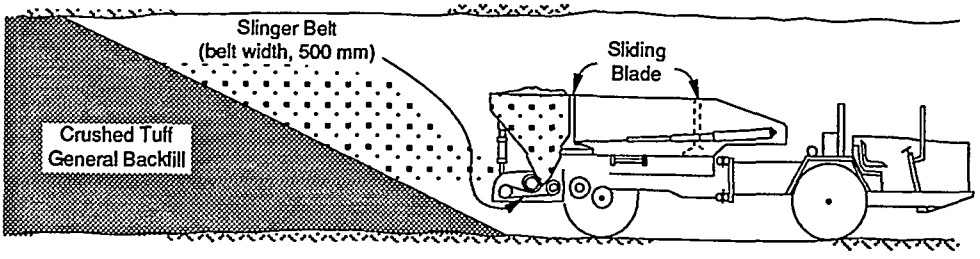
a) Method for Pneumatic Stowing Using Mobile Stowing Unit



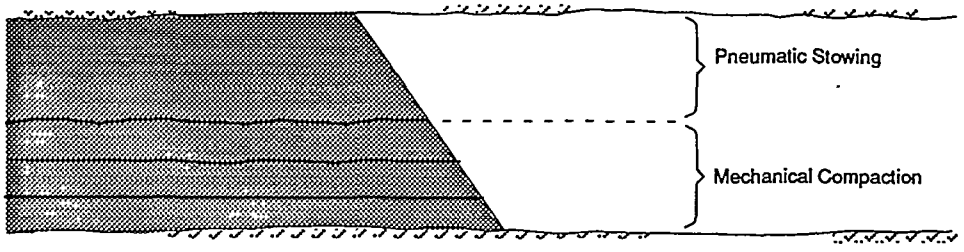
b) Mechanical Backfilling Cycle, Dumping Element



c) Mechanical Backfilling Cycle, Tamping Element



d) Truck-Mounted Belt Slinger



e) Combination of Mechanical Backfilling and Pneumatic Methods

Figure 5-13
Methods for Backfilling

5.3.1.4 Availability of Technology

Technology for placement of general backfill in underground openings under normal environmental conditions is generally available and is being routinely used in dozens of underground operations today. Some of these are described in Appendix A. However, many of these operations have specific criteria for backfill that may differ from that required for a nuclear waste repository in the unsaturated zone. Little information on the properties and quality of the in-place fill is typically recorded or even measured; whether the fill enables economic construction or ore extraction is generally considered sufficient evidence of quality. Adjustments to fill quality generally can be made if required until the optimum tradeoff is achieved.

The question of whether or not the technology for placing fill in a repository is "available" depends partially on the specifications for fill placement. For instance, it can be said with a high degree of confidence that crushed tuff of any desired gradation can be prepared in a surface backfill plant. Case histories are presented in the appendices of this report to show that fill can be placed underground and compacted to within 6 in. or less from the back using available technology. As part of the emplacement cycle, ground support can be removed, if required, using available technology with minor modifications, but which has not been demonstrated in a full-scale operation of this kind. Some efficiencies may be achieved by developing special equipment using generally available components that have been modified and packaged for specific tasks.

However, if very stringent specifications are established for the backfill that exceed present industrial practice, new technology may have to be developed. For example, if specifications call for fill that must be very tightly compacted against the roof and contain less than 0.01% particles finer than 1 mm in-place, a backfill contractor using available technology could not ensure performance without a field-testing program to demonstrate that available technology exists. Such a field-testing program has been proposed (Fernandez et al., 1993).

Mining operations do not commonly provide any compaction to their fill other than what is provided by the normal emplacement method. This is because standard compaction methods commonly used in civil earthwork (Figure 5-14) are difficult to apply underground due to limited clearance. However, equipment capable of compaction in very tight areas could be designed based on established methods and components (Figure 5-15).

GENERALIZED RANGE OF APPLICATIONS FOR COMPACTORS ACCORDING TO MATERIALS

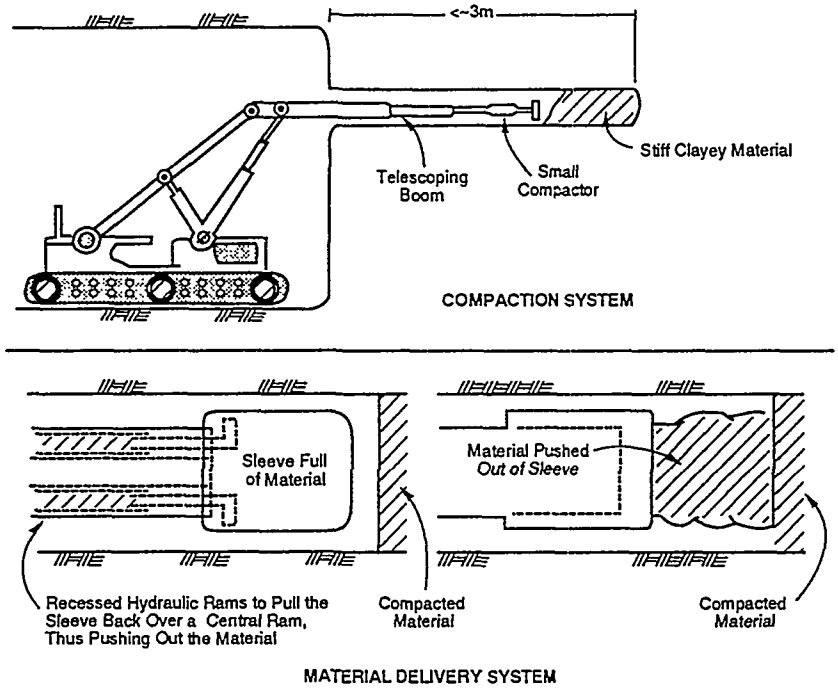
KIND OF COMPACTOR	RESIDUALS			WEATHERED ROCK-EARTH RIPPED			SEMISOLID AND SOLID ROCK BLASTED	
	CLAYS	SILTS	SANDS	MAXIMUM WEATHERED	AVERAGE WEATHERED	MINIMUM WEATHERED	WELL BLASTED	POORLY BLASTED
SHEEPSFOOT	X	PRESSURE; KNEADING			X			
TAMPING FOOT	X	PRESSURE; KNEADING; IMPACT; VIBRATION					X	
VIBRATORY FOOTED DRUM		X	PRESSURE; KNEADING; VIBRATION		X			
SMOOTH DRUM					X	VIBRATION		X
PNEUMATIC TIRES				X	PRESSURE; KNEADING		X	

KIND OF COMPACTOR	ALLUVIA		
	SANDS	GRAVELS	COBBLES
VIBRATORY, SMOOTH DRUM	X	VIBRATION	X
PNEUMATIC TIRES	X	PRESSURE; KNEADING	X

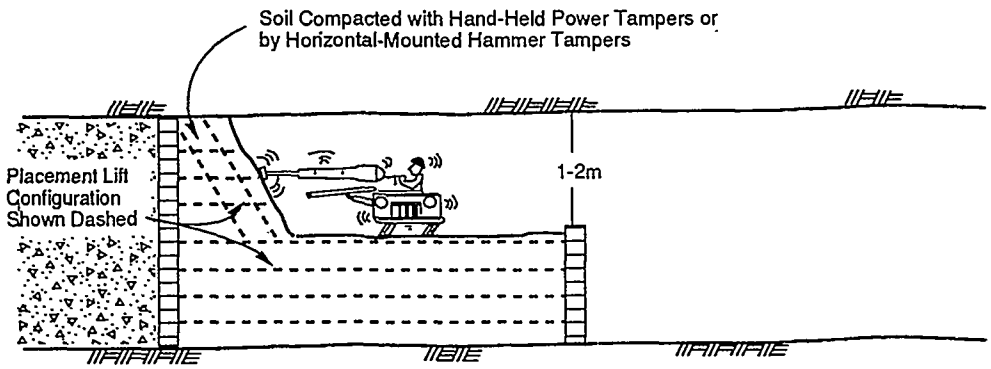
SEALING COMPONENTS ↑ IMPERMEABLE COMPONENTS ↑ FILTER ↑ GENERAL FILL

Figure 5-14

Compaction Methods and Their Application (after Church, 1981, pp. 14-27)



a) Research and Development Concept for Placing Clay Fill Horizontally



b) Research and Development Concept Compaction of Upper Portion of Tunnel

Figure 5-15

Methods for Compacting Fill in Tight Areas Using Nonavailable Equipment (after Smith, 1980)

If adverse environmental conditions require remote placement of fill at a high density, new technology may be required. This might include remote control equipment or robotics.

5.3.2 Graded-Fill Structures, Drifts and Ramps

5.3.2.1 Criteria

Graded-fill structures involve placement of vertical or inclined layers of earthen materials with varying gradations within the general fill. The methods used must be compatible with those used to emplace the general fill. All of the criteria that apply to the general fill also apply to graded-fill structures. Additionally, the following criteria apply:

- provide the ability to cut a keyway,
- maintain flexibility to emplace different materials,
- enable compaction of different materials, and
- provide flexibility to emplace materials of different sizes.

5.3.2.2 Screening

The first screen for the general fill also generally applies to graded-fill structures. Hydraulic and gravity methods will be eliminated for the same reasons. However, since the volume requirements are much less, manual methods will remain after the first screening. Figure 5-16 shows the process for the first screen.

Additionally, graded-fill structures may require cutting of a keyway or cut-off trench as part of site preparation. The drill-and-broach method has been used for cutting slots in rock perpendicular to the free-working surface. A row of holes is first drilled, then the intervening webs are removed by broaching. The slot can be widened by drilling a second adjacent row of holes and breaking the rock out using a hydraulic splitter or expansive agent. Sandia National Laboratories has developed a rock-cutting diamond saw, which has been demonstrated in Yucca Mountain welded tuff as part of the high-level waste program. Such a saw could be truck-mounted and used as a very flexible method of rapidly cutting high-quality slots in rock. If narrow slots are not required, a roadheader or mobile miner can be employed to create a keyway.

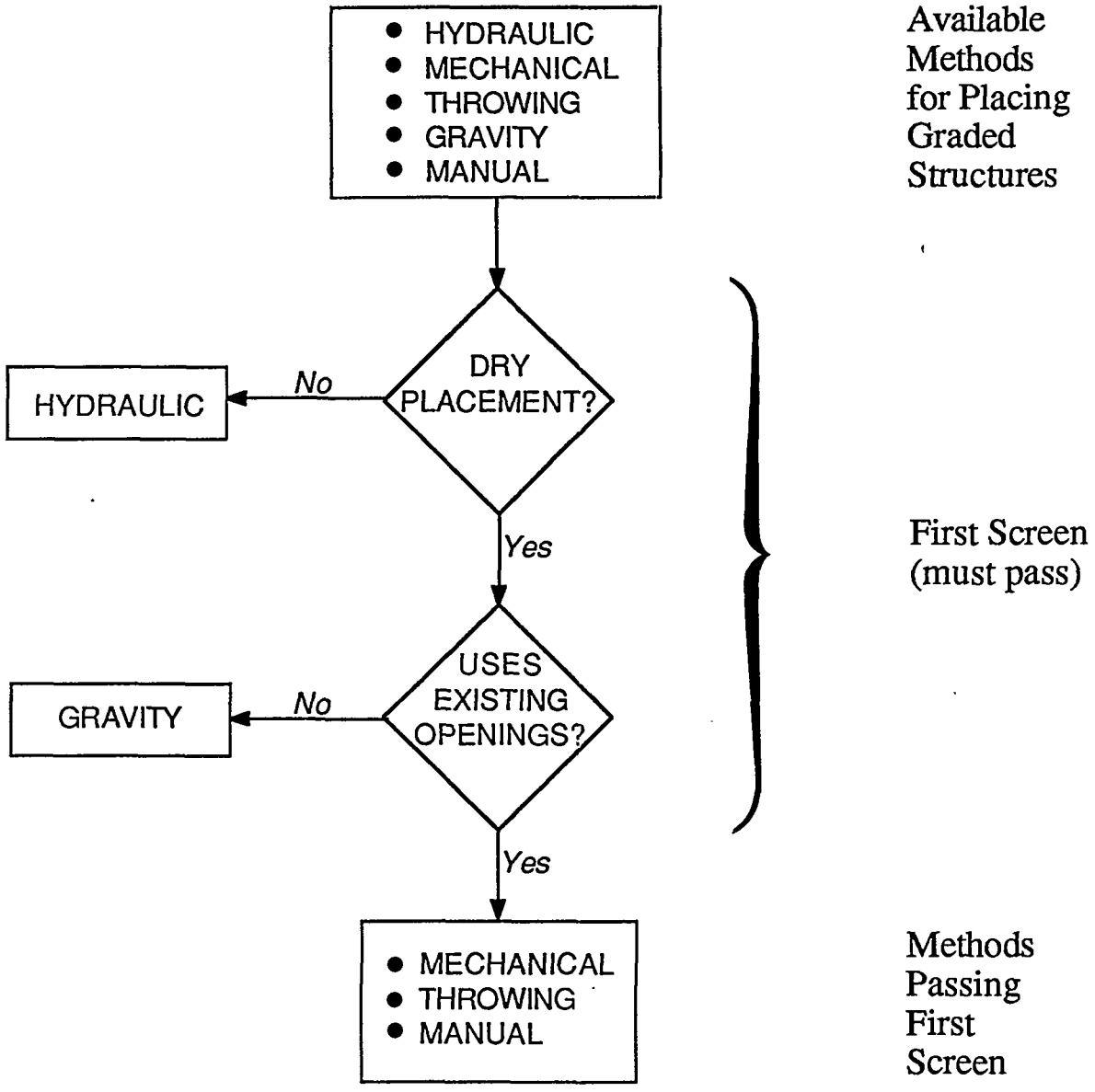


Figure 5-16
First Screen for Graded-Fill Structures

5.3.2.3 Suggested Sequence

Figure 5-17 shows a suggested sequence for placing graded-fill structures. First, general fill is placed at the angle of repose up to the location of the structure. Although not shown in this sequence, any preconditioning of the area would be performed in conjunction with placing the general fill, possibly including ground support removal, temporary floor removal, floor perforation by drilling, and cleaning activities. Next, a keyway may be excavated using a truck-mounted rock saw. Drilling and grouting may be used to supplement the cut-off radius of the keyway. After keyway preparation, a filter material is placed using pneumatic methods and compacted manually or with a special machine. The impervious core is manually placed using precompacted bentonite blocks and sandwiched using a front filter. The general fill cycle is then continued to the next graded-fill structure site.

5.3.2.4 Availability of Technology

The same comments relative to the general fill apply here. Additionally, keyway cutting and grouting technology must be addressed.

The drill-and-broach and diamond saw can be considered available technology. Some research and development would be required to convert the saw from a research tool to a truck-mounted production unit. As discussed in Chapter 4, a wide range of technology is available for special grouting applications.

5.3.3 Bulkheads and Plugs, Drifts and Ramps

5.3.3.1 Criteria

There are many methods for constructing bulkheads of cementitious materials. However, they are based on one of the following general approaches: (1) manual construction using precast blocks, (2) cast-in-place construction from the surface, (3) cast-in-place construction from underground, (4) grouted concrete from the surface, and (5) grouted concrete from underground. Variations on the different methods involve whether or not a keyway is used, details of the grouting process, and multilift construction sequences for very long plugs. Bulkheads and plugs are in many ways similar to graded-fill structures and are intended to perform the same function. However, because they may experience considerably higher hydraulic pressures under extreme off-normal (but possible) scenarios, different criteria will be used to evaluate construction methods. The criteria for constructing a bulkhead or plug are listed below:

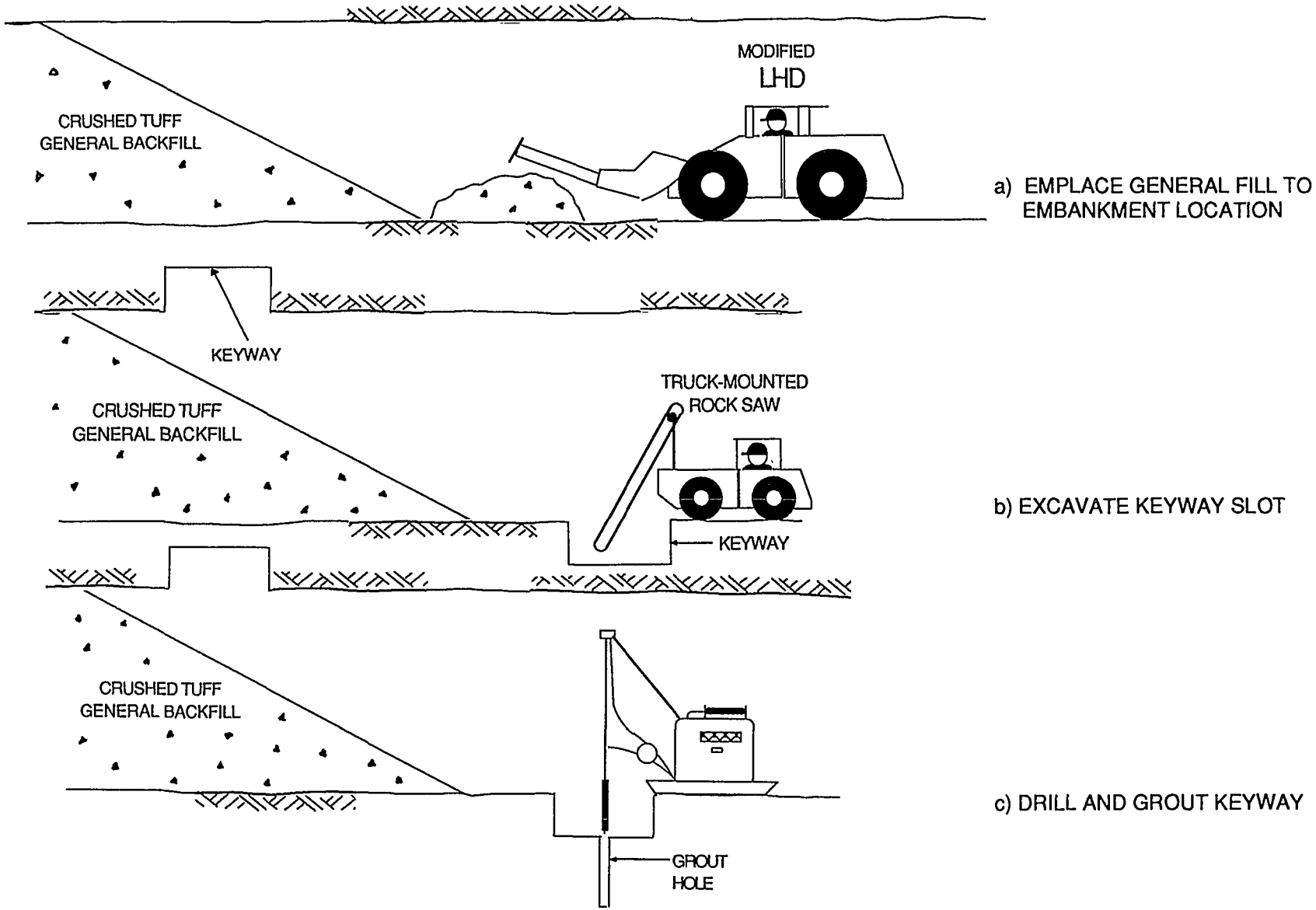
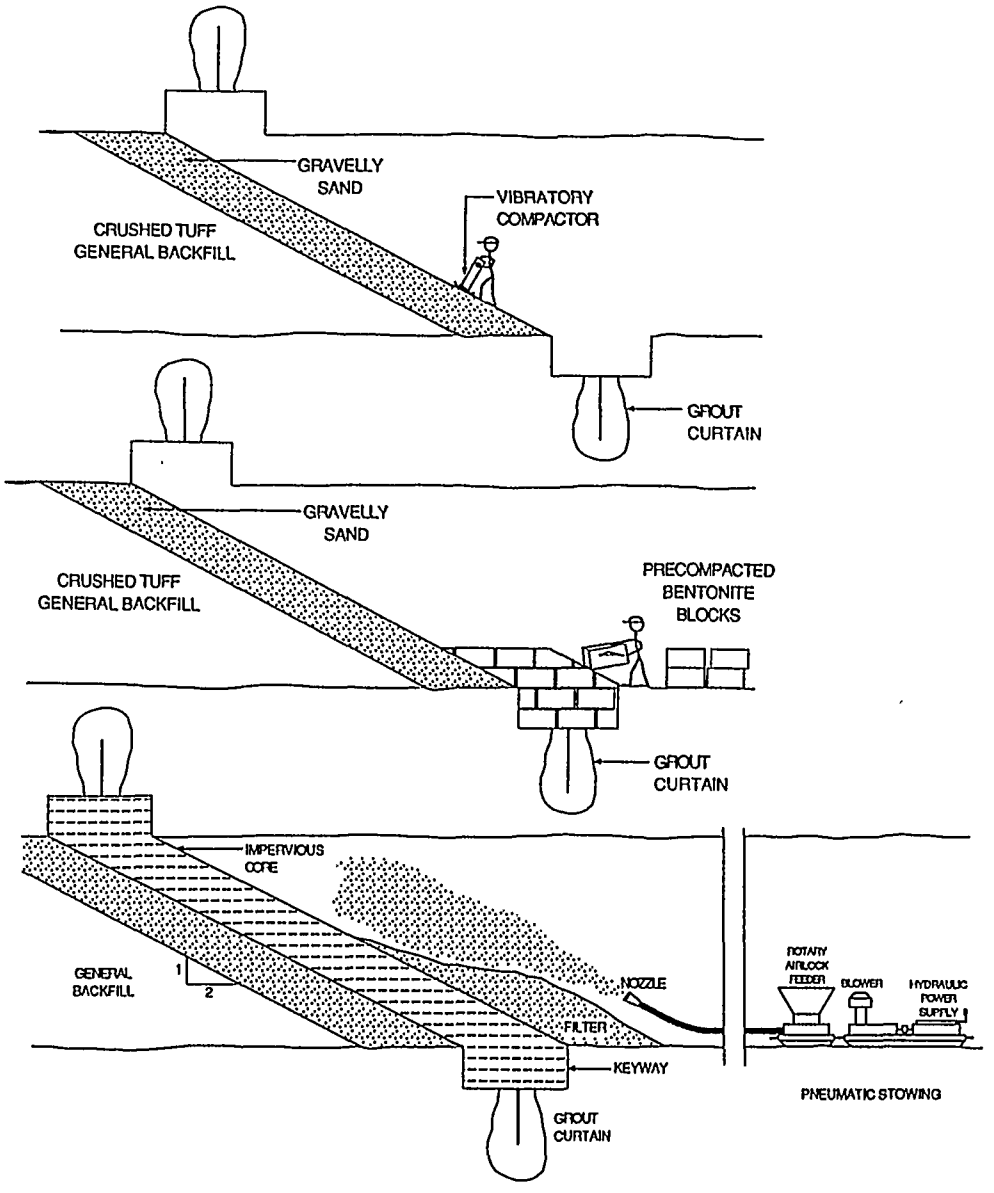


Figure 5-17
Suggested Sequence for Placing Graded-Fill Structures



d) EMPLACE AND COMPACT FILTER MATERIAL

e) EMPLACE PRECOMPACTED BENTONITE BLOCKS

f) EMPLACE FRONT FILTER

Figure 5-17

Suggested Sequence for Placing Graded-Fill Structures (concluded)

- minimize water use for concrete/grout placement,
- be suitable for placing special durable concretes,
- use existing openings for concrete/grout transport,
- provide the ability to cut a keyway,
- minimize cost and schedule, and
- provide maximum worker safety.

5.3.3.2 Screening

The first three criteria can be considered "musts." The surface-based construction methods, useful for sealing abandoned mines where access is not available underground, can be eliminated by the requirement to use existing openings for transport. Figure 5-18 shows the application of the first screen to plug construction methods.

Further screening will be based on such considerations as cost and schedule. Performance of all remaining plug construction should be adequate if used in conjunction with special grouting techniques.

5.3.3.3 Suggested Sequence

Figure 5-19 shows a suggested sequence for constructing a tapered cementitious plug using premixed concrete pumped from underground. General fill is emplaced up to the plug location. If desired, shotcrete may be used to coat the fill surface to make an anterior form for the plug. The keyway is excavated using a roadheader. Grout holes are drilled radially from the keyway; any geologic features are given special treatment. A front form is constructed, and concrete is mixed and placed beyond the form in one continuous pour to eliminate cold joints. A superplasticizer, if geochemically compatible with the waste package, may be required near the top to assure a good contact. Postgrouting is then used to seal off remaining voids and to preconsolidate the plug. Additional stages of grouting with higher pressure and with finer grouts may be employed as required by the specifications.

5.3.3.4 Availability of Technology

As discussed in Chapter 3, the technology for emplacing cementitious seals is well-established. The issue remaining is whether the seal and its grout curtain will survive for the required time period. This requires special quality control during placement in addition to scientific selection of seal materials.

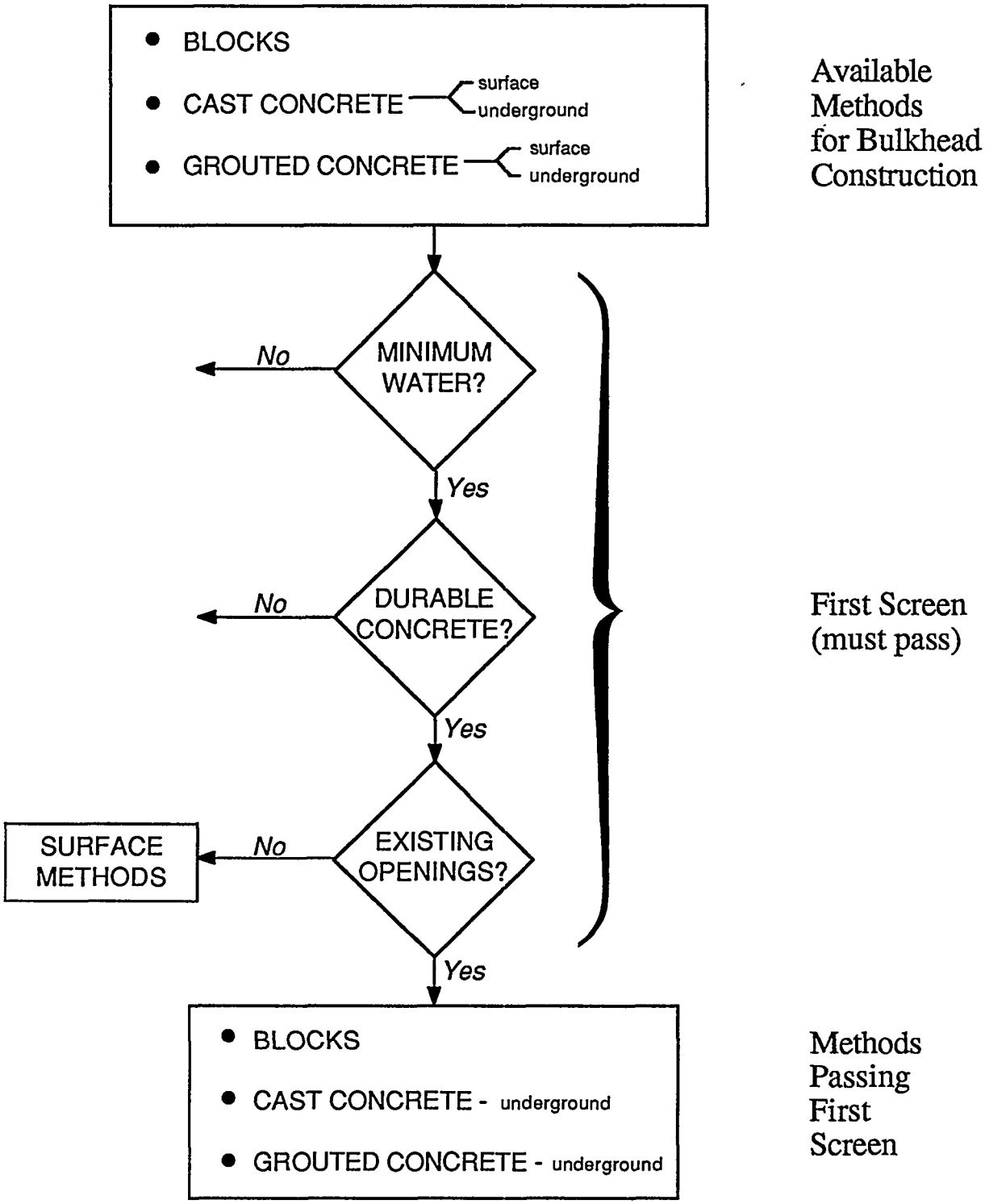


Figure 5-18

First Screen for Selection of Bulkhead Construction Methods

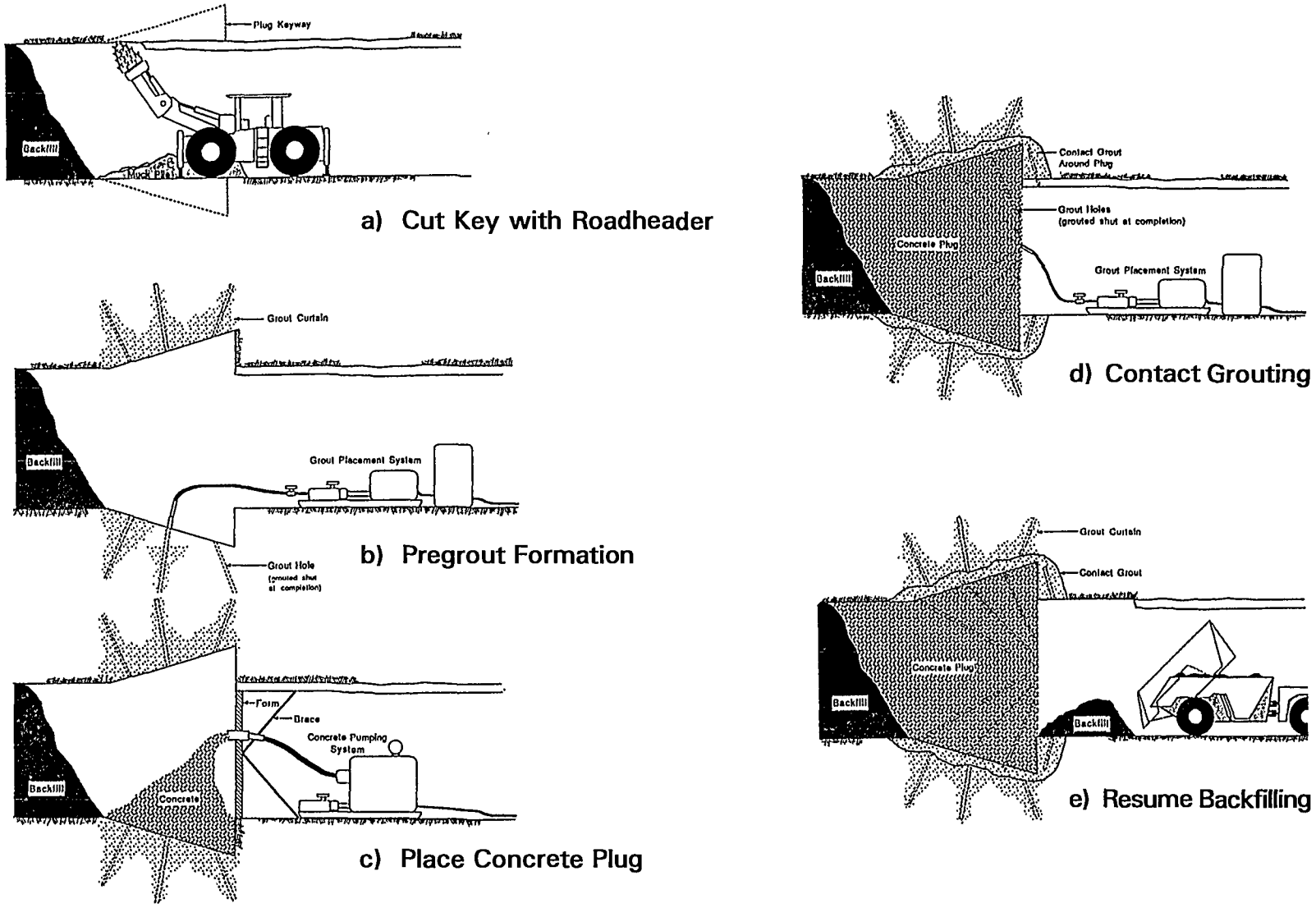


Figure 5-19

Suggested Sequence for Constructing a Tapered Cementitious Plug
(Note: Temporary ground support required although not shown)

5.3.4 General Fill, Shafts

5.3.4.1 Criteria

Shaft backfilling and sealing present different problems than drifts and ramps. Shafts are more difficult work environments than drifts, requiring special staging for access to the shaft walls. Men, equipment, and supplies must be hoisted, unlike drifts where rubber-tired equipment can provide convenient access. Men are continually exposed to heights and falling objects, making safety a special priority. However, for backfilling and sealing operations, shafts offer certain advantages. Unlike drifts, gravity assists shaft sealing by helping with material compaction and filling close to the walls. There is considerable clearance overhead, enabling compaction equipment and concrete vibrators to work above the fill or plug being constructed.

The criteria for selecting a method for emplacing general fill in repository shafts are to

- minimize water used for fill placement,
- use the existing opening for fill transport,
- have the capacity to place a moderately large volume of fill,
- facilitate compaction and quality control,
- minimize degradation/fines production,
- enable concurrent lining removal if required,
- minimize cost and schedule, and
- provide maximum worker safety.

Placement of fill in abandoned mine shafts is traditionally done by dumping waste rock and other convenient materials down the shaft until it is full, after which a simple cap may be placed over the opening. It is unlikely that such procedures will be followed in a repository, where a higher degree of quality control will be required to ensure seal performance. Additionally, shaft filling operations may be conducted concurrent with liner removal operations, requiring careful placement of fill underneath the working platform to protect equipment. Also, unrestrained transfer of fill to the shaft bottom by gravity is certain to cause undesirable degradation of the fill.

Due to the vertical shaft configuration, all options for placing fill involve gravity, to a large extent. Mechanical equipment will be used to transfer equipment underground, in some cases, and will be used primarily for spreading and compaction once the fill arrives at the

shaft bottom. Compaction is probably more important in shafts than in drifts because the large vertical extent makes relatively large absolute settlement amounts possible over long periods of time and with repeated earthquake episodes. Because of the effects of gravity, it is unnecessary to use pneumatic or slinger methods to transfer or place fill.

Based on the qualitative criteria, it appears likely that a gravity/mechanical combination will be selected for shaft filling.

5.3.4.1 Suggested Sequence

Figure 5-20 (a and b) shows a typical method for mechanical/gravity placement of general fill. After liner removal and surface preparation operations, a measured quantity of backfill would be carefully transferred to the shaft bottom, either in skips, as shown, or using a fill pipe. Mechanical equipment would then be used to spread the fill. Depending on the fill characteristics, rubber-tired tamping or vibrating compaction equipment would be operated on the surface of the fill until the specified density was achieved. The cycle would be repeated in lifts, the thickness of which would be selected to optimize compaction efficiency.

5.3.4.3 Availability of Technology

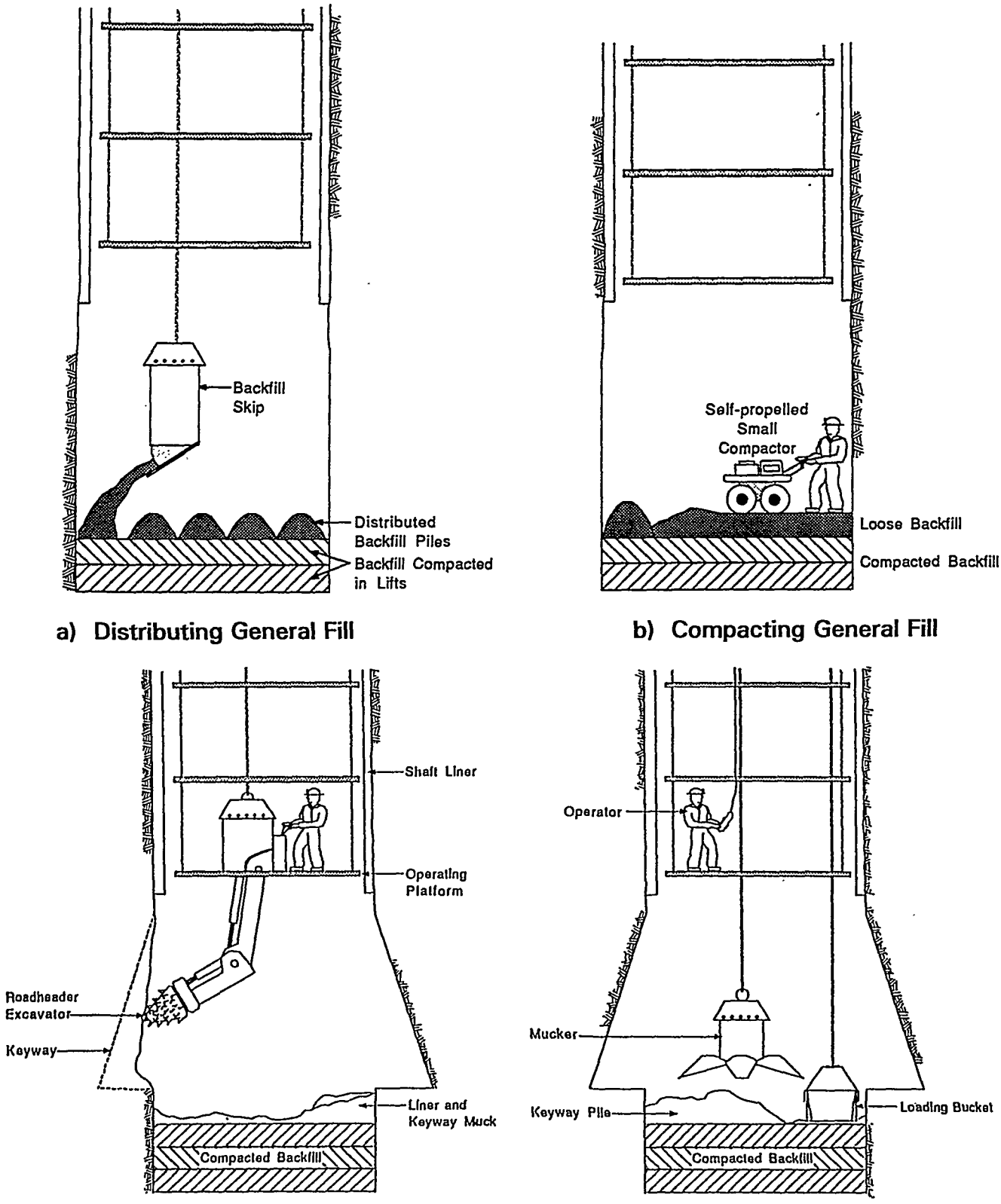
The discussion in the previous section pertaining to drift and ramp backfill is also generally appropriate to shafts. Technology for placement of general backfill in vertical shafts, such as that shown in Figure 5-20, is currently available or could be readily developed. Testing with the selected backfill materials and methods is required to ensure that specified placement and compaction are possible without excessive degradation.

5.3.5 Bulkheads and Plugs, Shafts

5.3.5.1 Criteria

Criteria for selecting methods of emplacing shaft bulkheads and plugs are similar to those for drifts, although the methods themselves are somewhat different because of the different configuration. The following criteria apply:

- minimize water use for concrete placement,
- be suitable for placing special durable concrete,
- use the existing opening for concrete placement,
- provide the ability to cut a keyway,



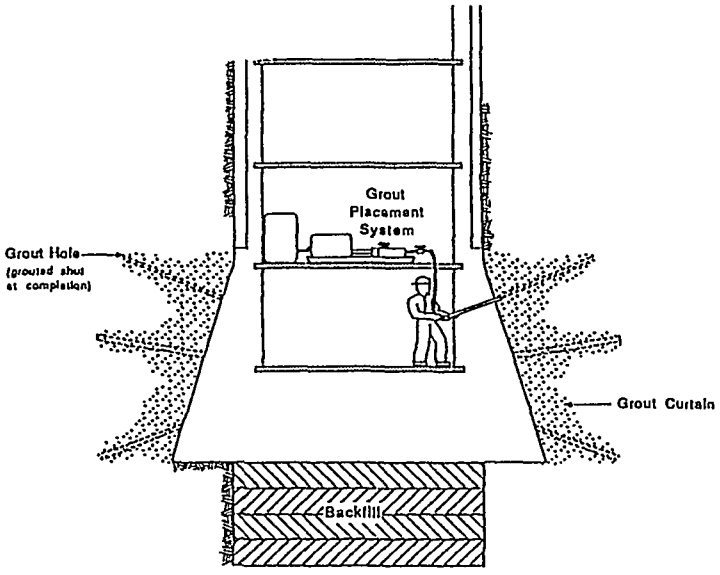
a) Distributing General Fill

b) Compacting General Fill

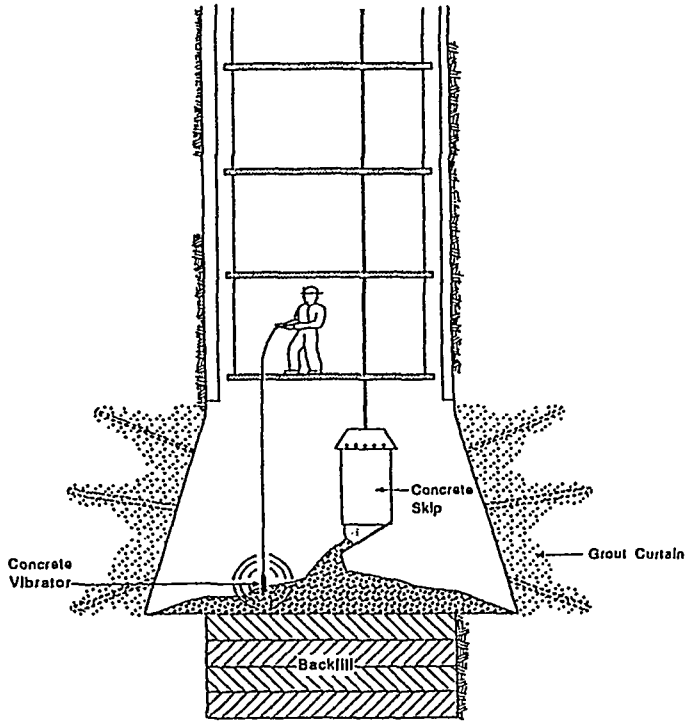
c) Excavating Keyway Using Mechanical Excavator

d) Removing Keyway Debris

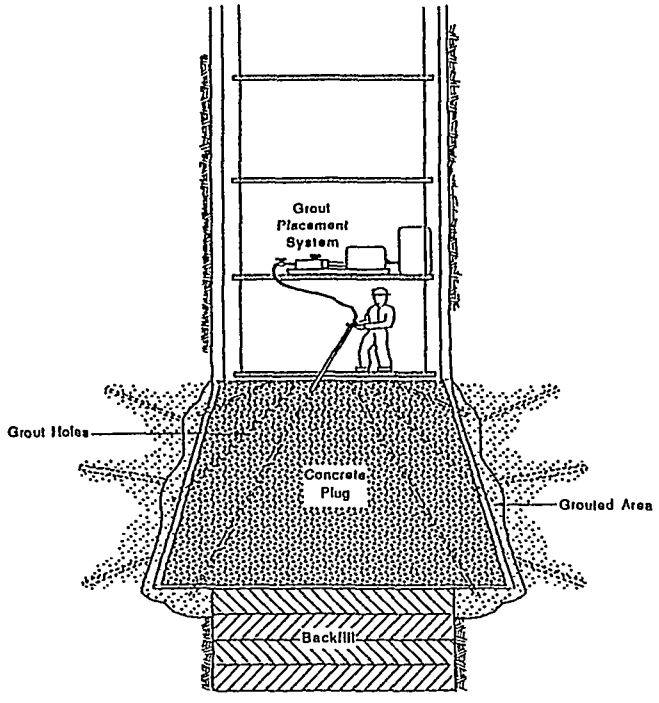
Figure 5-20
Typical Sequence for Sealing Repository Shaft



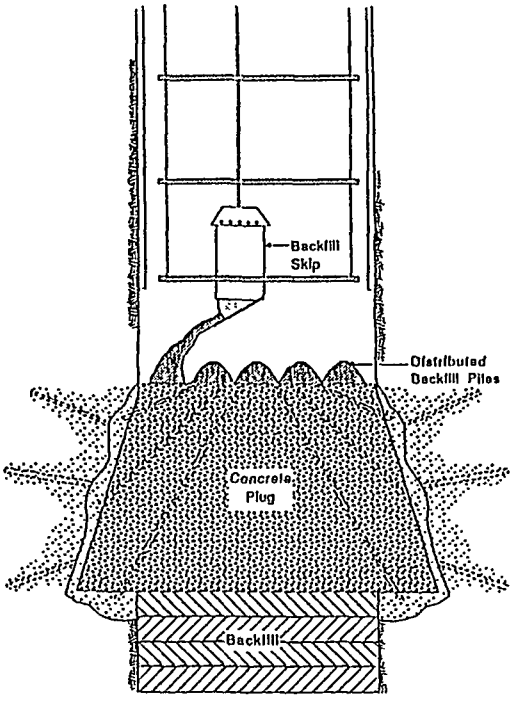
e) Drilling Holes for Pregrouting



f) Placing Concrete



g) Contact Grouting



h) Resumption of General Backfilling

Figure 5-20
Typical Sequence for Sealing Repository Shaft (concluded)

- minimize cost and schedule, and
- provide maximum worker safety.

5.3.5.2 Suggested Sequence

Figure 5-20 (c through h) shows a suggested sequence for constructing a tapered cementitious plug in a shaft. The liner is removed and a keyway is excavated using a roadheader. If the roadheader cannot excavate the material, drilling and splitting will be performed. Excavated muck is pushed into the shaft and mucked out with an orange peel grab, or similar mucking device, and hauled to the surface. Grout holes are drilled radially from the keyway and grouted in stages using successively finer materials and higher pressures, if required; any geologic features are given special treatment. The shaft fill is then smoothed and compacted. Grout pipes are positioned, and concrete is placed in one continuous pour from the surface. Remixing may be required on the work deck. After curing, holes will be drilled through the plug, and contact grouting will be performed to seal off remaining voids and preconsolidate the plug. Additional stages of grouting with higher pressure and with finer grouts may be employed as required by the specifications.

5.3.5.3 Availability of Technology

Methods for emplacing cementitious seals in the shaft are generally available technology. As discussed in Chapter 4, a wide range of technology is available for special grouting applications.

5.4 Some Long-Term Performance Concerns

5.4.1 Backfill Size Control (Fines Creation)

One important concept in sealing a nuclear waste repository in the unsaturated zone is to allow any water that has entered the repository to drain vertically downward without contacting the waste packages. Sumps and other areas will be developed where water can collect and drain into the host formation. The natural permeability of the rock mass in these areas therefore becomes an important sealing component, and care must be taken to preserve and enhance it. The most likely manner in which this permeability could be reduced dramatically is by migration of fine materials through the fill and their ultimate deposition in the fractures that give the rock mass its free-draining characteristics. Accordingly, careful

control of the backfill material size characteristics may be required to achieve the necessary specification.

The life cycle of the crushed tuff expected to comprise the bulk of the general backfill for the potential Yucca Mountain repository will have a major influence on the size characteristics of the fill as ultimately placed. Although the design of the complete mining and material handling system is conceptual at present and will not be finalized until a later date, the technology for sizing and handling mined material intended for backfill is well-established and can be discussed in general terms with reasonable confidence. Figure 5-21 is a block flow diagram showing the major features of the conceptual system. Although the complete life cycle of the mined material will affect its ultimate character and will be addressed here, the portion following treatment in the backfill plant is of utmost importance to the sealing program, because it will be this portion that determines the ultimate size of the fill. Because fine materials can be removed prior to backfilling, the only constraint that repository sealing will place on the size distribution of the mined material prior to sizing operations is that a sufficient volume of appropriately sized material be produced to fill the total excavated volume. This constraint can be relaxed if plans are made to introduce makeup material from suitable nearby tuff quarries.

Excavation. The first step in Figure 5-21 is rock excavation. As discussed in Chapter 2, the SCP-CDR design assumes conventional drill-and-blast methods will be used for excavating much of the repository, with TBMs used for certain long drifts. More recent design concepts employ machine mining almost exclusively, although conventional mining may be required for rooms with geometries not amenable to mechanical excavation. It is likely that TBMs and other mechanical excavators, such as the Robbins Mobile Miner, will excavate the welded tuff at the repository horizon; roadheaders will be used for alcoves and irregular rooms in the nonwelded tuff of the Calico Hills Formation. It is possible that shafts will also be mechanically excavated through welded and nonwelded units using blind shaft borers, shaft boring machines, raise borers, or the V-mole.

The excavation method selected will influence the run-of-mine rock size and gradation. Conventional drill-and-blast mining will produce a relatively coarse product, with considerable variability depending on rock mass characteristics, blasthole pattern, and powder factor used for primary breakage; widely spaced blastholes and low powder factors tend to provide coarser fragmentation. A typical blasting round can produce boulders up to 3 ft in diameter and considerable rock over 8 in., which is too large to be effectively handled using

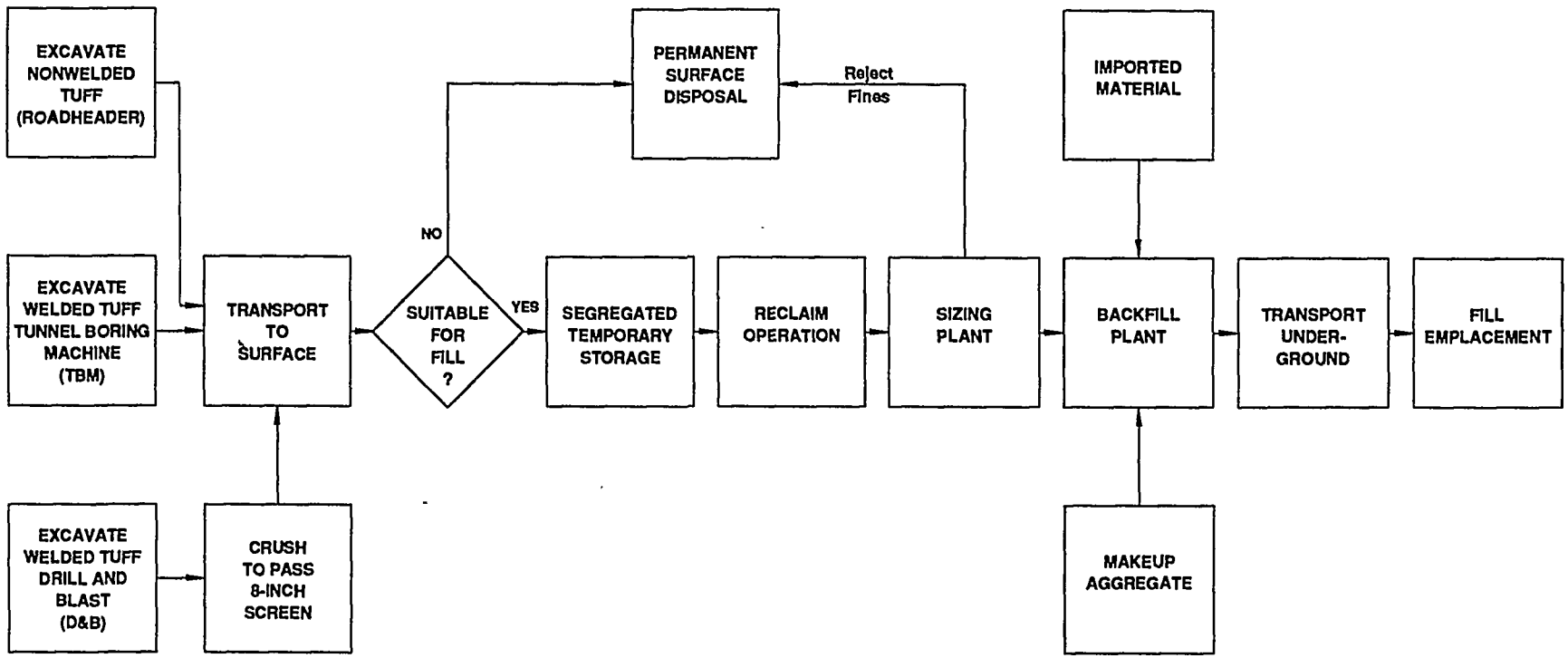


Figure 5-21

Material Flow Diagram for Material Preparation and Handling of Backfill

a conveyor system. A primary crusher or feeder breaker will be required at the face to reduce the top size to 8 in. prior to conveyor haulage. In addition to oversized material, fines will be generated by blasting; the proportion of fines will depend on the rock material characteristics and powder factor. A feeder breaker to size shot rock at the face will produce additional fines.

Mechanical excavators will generally produce a finer product than drill-and-blast methods. The material size distribution produced by roadheaders depends on the properties of the rock mass and the design of the cutter head. Bit type and angle, cutter head lacing pattern, and drum rotational speed can affect the product size. Although crushing will not be required, the gathering-arm loading system and conveyer will cause some additional size degradation. TBMs will also produce a material size dependent on cutter head variables, including bit type and spacing, and on the mechanical action of the loading system.

Some testing on Yucca Mountain welded tuff has already been performed at the Earth Mechanics Institute of the Colorado School of Mines using a linear cutting machine (LCM) (Gertsch and Ozdemir, 1991). The LCM provides a full-scale method of testing cutter performance in the laboratory. In this test, a single cutter is mounted on the machine and repeatedly dragged across a sample of the site rock (Figure 5-22). Cutter penetration and spacing of the cuts can be varied for each pass of the test. The specific energy required to break the rock is measured and used to optimize the cutter type and configuration. There are some differences between laboratory tests using the LCM and the performance of a machine in the field that may affect the size distribution of cuttings produced. The actual path of the bits in the field is a circular arc rather than a straight line, and multiple bits rather than a single bit will be cutting the rock at any given time. Another bias common to laboratory testing is that the sample tested may contain fewer fractures than average, affecting breakage characteristics. Also, the face muck-handling system required on practical mining machines will produce some secondary degradation not accounted for in the laboratory tests. The linear cutter results in welded tuff, which is hard and resists secondary degradation, are considered fairly representative of actual results for machine mining at the face prior to loading.

The LCM was used to test samples of welded tuff taken from Fran Ridge at the NTS, representing the Tsw2 unit that is the potential repository horizon. Two different types of cutters were tested: disc cutters, such as those used on TBMs, and point-attack cutters used by roadheaders. These tests were used to determine optimum cutter types and cutting

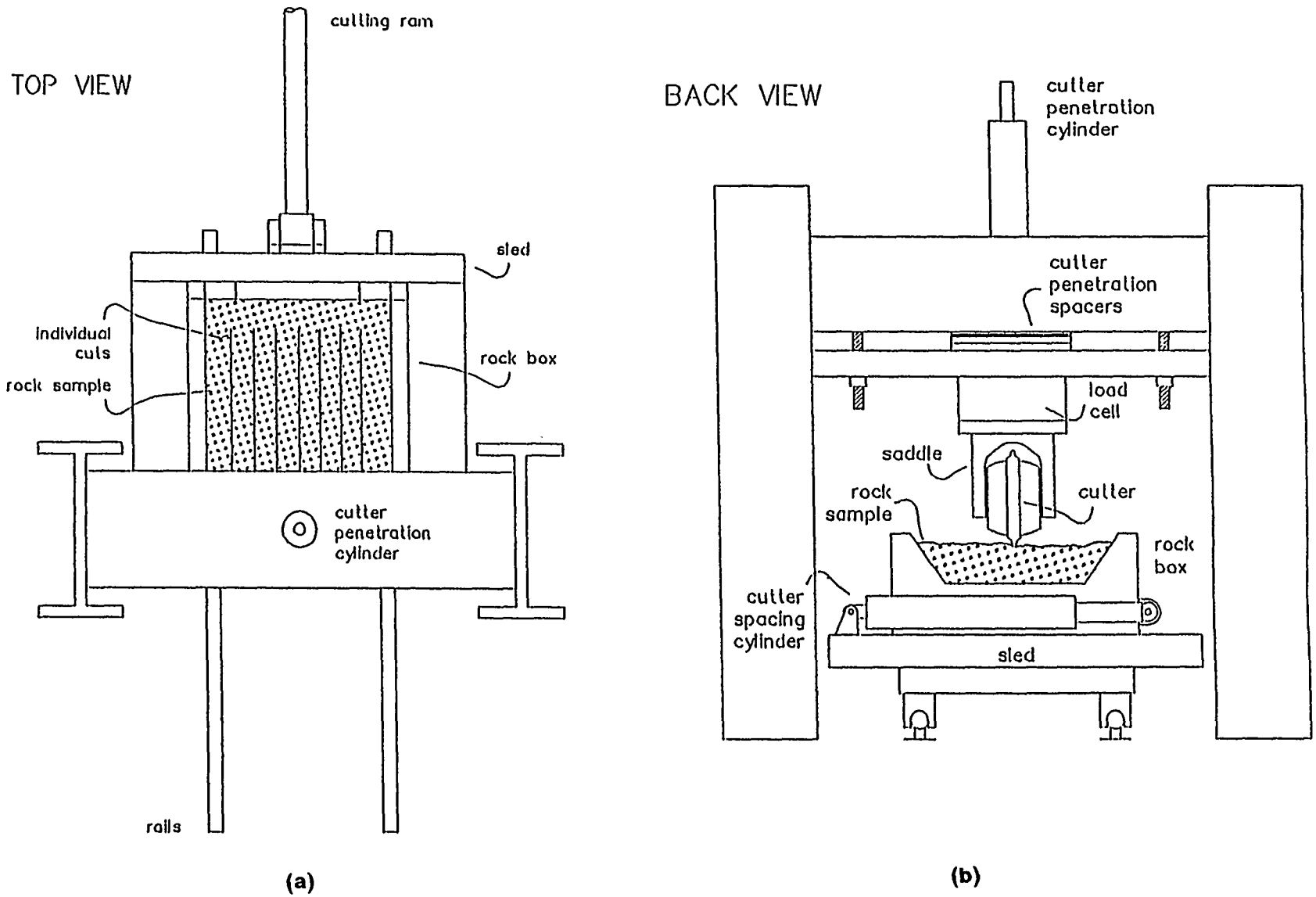


Figure 5-22

Schematics of Colorado School of Mines Linear Cutting Machine (from Gertsch and Ozdemir, 1991)

parameters. Nine separate tests involving three penetrations (0.2 to 0.4 in.) at three spacings (3 to 5 in.) were performed for each of the four cutters, for a total of 36 tests. Cuttings from the tests were analyzed for particle-size distribution. Because the primary interest was the fine fraction that might become airborne, the largest opening size was 25 mm; particle-size distribution above this size was not obtained. The overall average of the 18 size distributions for each of the two basic types of cutters are shown in Figure 5-23. Curve 3 was generated using a disc cutter and is representative of TBM cuttings, while curve 4 shows results from a pick cutter, as might be used in a roadheader. For comparison, curves 1 and 2 show size gradations for tuff crushed in the laboratory with jaw openings of 3 and 1.5 in., respectively. Curves 5 and 6 are field results obtained from mining operations at Little Skull Mountain at the NTS. Both of the NTS curves represent nonwelded tuff from the Wahmonie Formation and were obtained from screening cuttings from a TBM with disc cutters at two locations in the test tunnel. Curve 7 is an estimated size distribution from a typical conventional heading round. The following observations from Figure 5-23 can be noted:

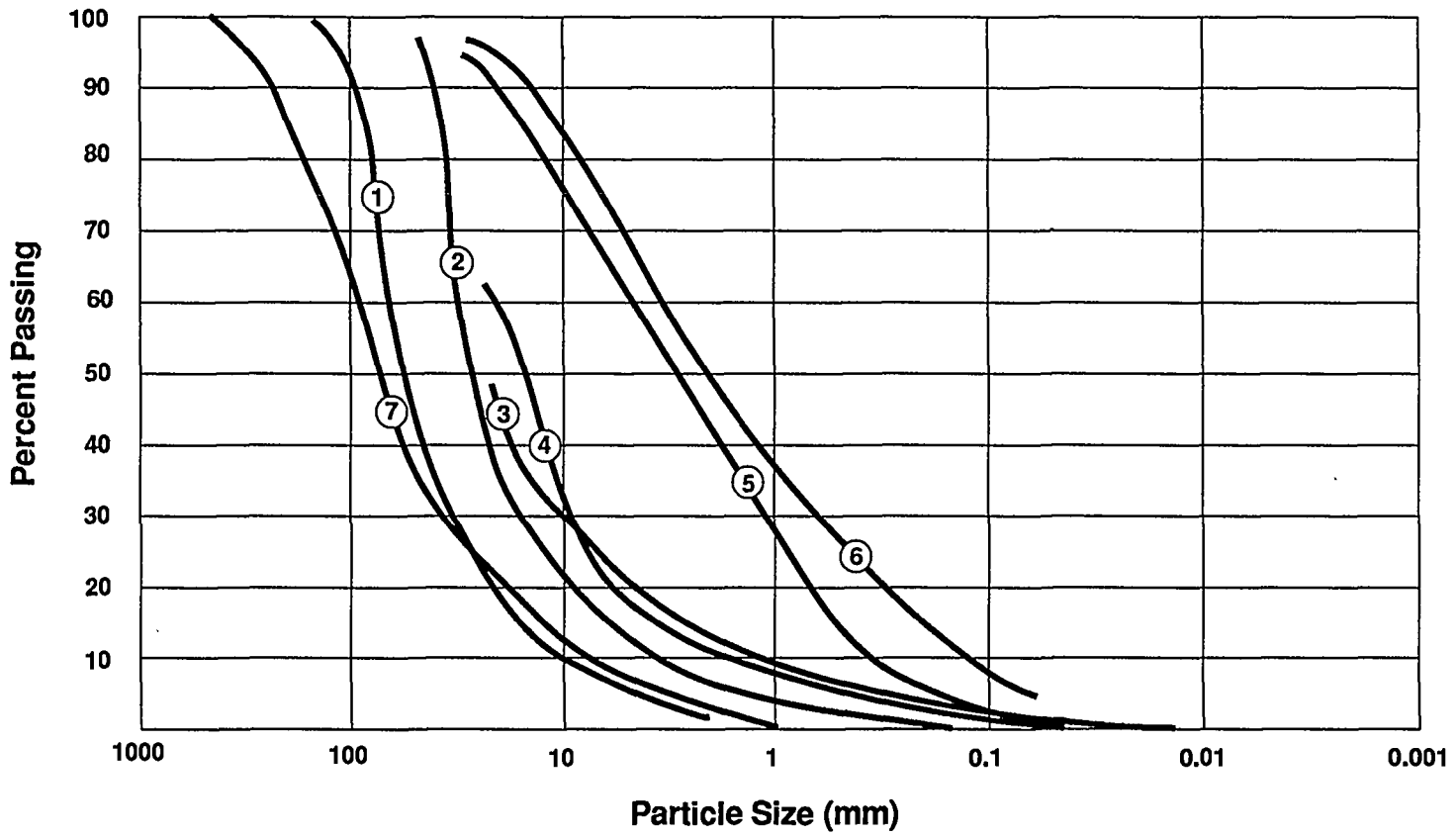
- Conventional mining will typically produce more oversized material and a coarser average particle size than mechanical mining.

- Size distributions from pick and disc cutters will not be too dissimilar in the same rock. Both cutter types produce a large fraction of plus 25-mm material, with the disc cutter producing slightly more coarse material (54% versus 38% for the pick) in the LCM tests.

- The TBM in nonwelded tuff produces more fine rock than any of the breaking operations in welded tuff.

- A mechanical crusher will produce some material coarser than its opening size and some fines.

Transport to Surface. Size characteristics of the mined material may be altered somewhat by size degradation during subsequent transportation. Belt conveyors and/or rubber-tired vehicles are likely options for transporting excavated material from the face areas to the surface; belt conveyors were chosen for the SCP-CDR design (Figure 5-24). Conveyors may also be used to transport material back underground. Devices used for feeding belts can degrade sensitive materials, and additional minor degradation will also occur as a result of interparticle movement on the belt itself. This can be minimized by limiting the conveying



Crushed Tuff Laboratory Jaw Crusher		Colorado School of Mines (CSM) Linear Cutter		Mined Tuff Little Skull Mountain		Drill & Blast
3" Opening	1.5" Opening	Disc	Pick	Bedded Gray Tuff Wahmonie Formation	Lower Red Tuff Wahmonie Formation	(Estimated)
1	2	3	4	5	6	7

Figure 5-23

Size Distributions for Rock Broken by Various Methods

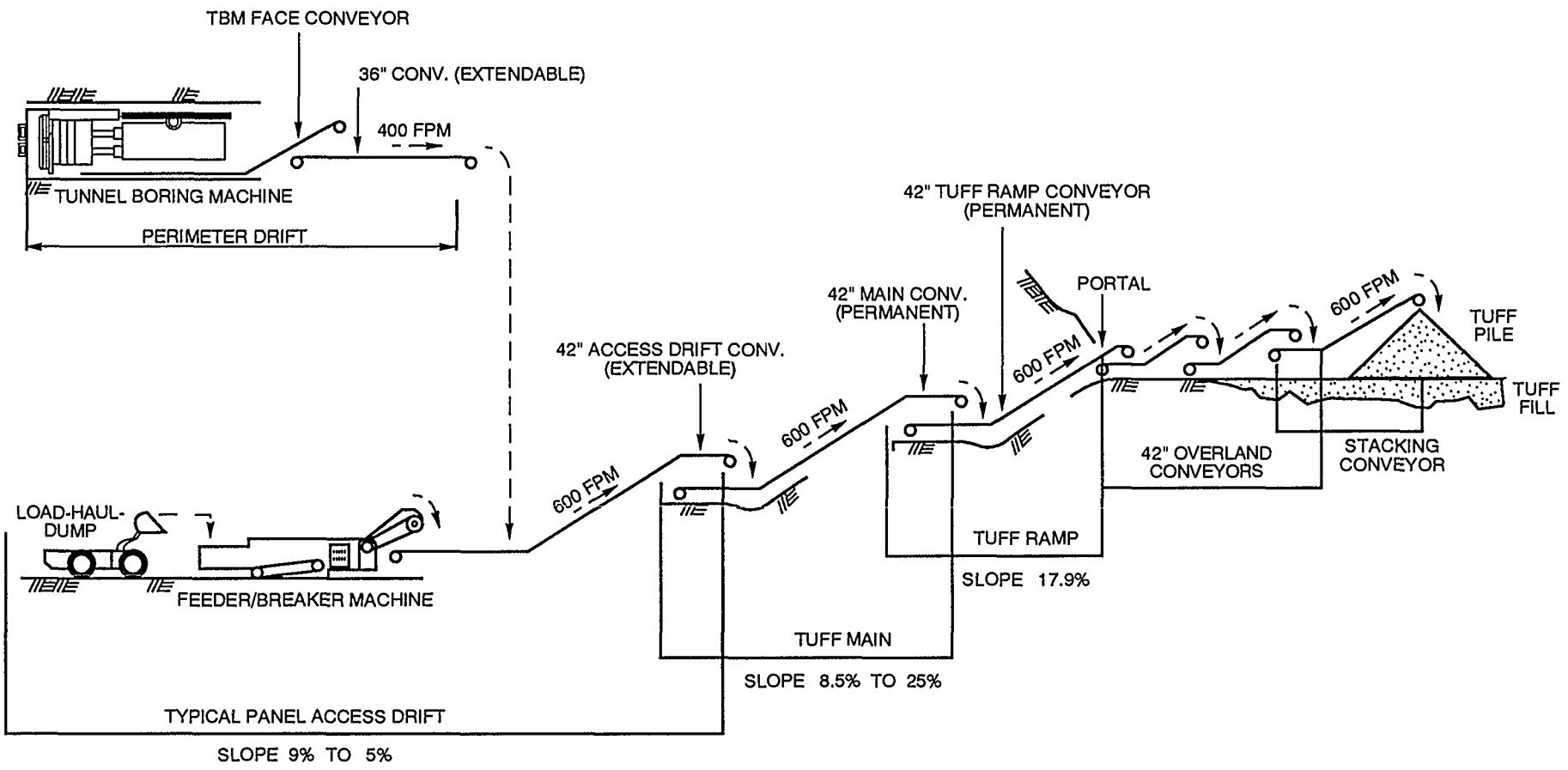


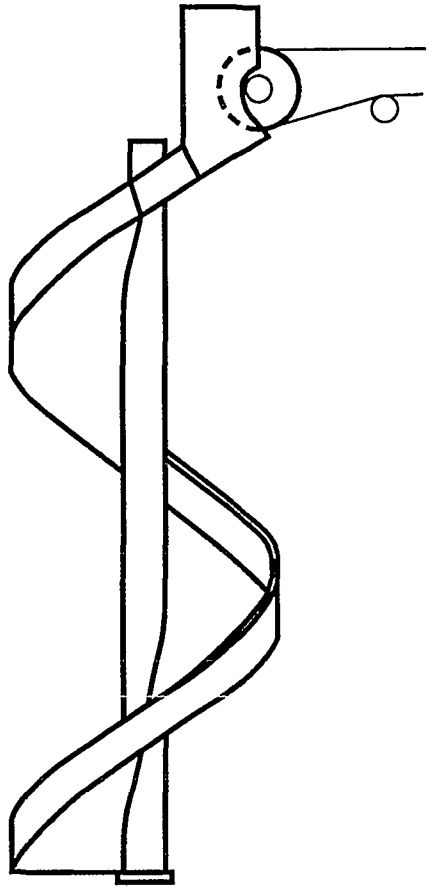
Figure 5-24
Material Flow Diagram—Tuff Handling (from SNL, 1987)

speed for degradable materials (CEMA, 1979). With slower speeds, wider belts will be required to maintain the required capacity, although this involves cost tradeoffs. Further degradation will occur at transfer points between different conveyor sections. Little quantitative work is available in the published literature on degradation of various materials by conveying systems. Published qualitative recommendations suggest degradation will not be a problem in crushed limestone, although materials, such as shales, require careful handling (CEMA, 1979). The transport characteristics of welded tuff are expected to be similar to crushed limestone, suggesting limited degradation is likely. Nonwelded tuff will exhibit behavior more like soft sandstone during transport. Testing of the actual materials will be required for final design of the materials handling system.

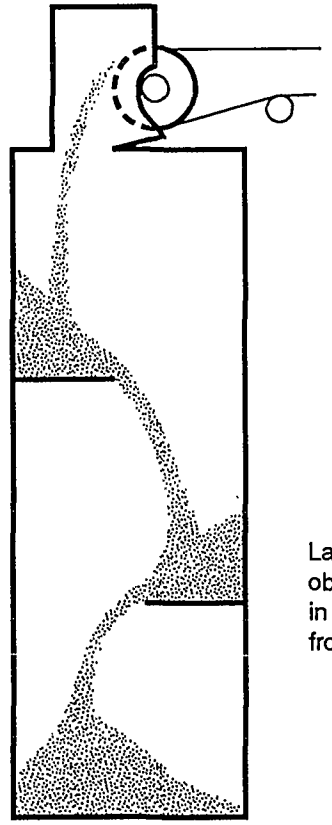
It is possible that the method used to excavate the backfill may influence its ability to resist degradation during handling. This should also be evaluated.

Material Segregation and Storage. After transport out of the mine, the material may be directed to one of several storage or disposal areas. Material suitable for fill will be directed to a temporary storage pile, the "tuff pile," for 50+ years until repository decommissioning. Some materials may be encountered during repository development that do not meet backfill specifications; for instance, fault gouge may be considered too fine or friable for use as backfill. Other materials may be deliberately segregated into separate piles for backfilling different underground locations (such as welded and nonwelded tuffs). Alternative handling systems will be required to allow segregation and transport of the different material grades without interrupting face operations. One method used by industry involves surge bins to which mined ore or waste rock can be alternately directed and stored, permitting continuous operation of the main conveyor.

In the conceptual design, conveyors, a radial stacker, and mobile equipment are used to transport excavated tuff from the tuff ramp portal to the "tuff pile(s)" for temporary disposal. Placement of the material on the temporary storage pile can affect the subsequent reclaim and processing operations. Stacking or dumping the excavated material from high (greater than 20 ft) stacker positions or benches may produce accidental size segregation of the material as it cascades down the pile or bench face. This can be carried through the reclaim process, creating surge loads in crushing and screening operations. Size degradation can occur during placement of the material in the surface storage pile. Special discharge structures can be used to reduce degradation, such as those shown in Figure 5-25. Also, a stacker boom that can be maintained close to the discharge surface can accomplish the desired results. The



Spiral lowering chute



"Rock ladder" lowering chute

Ladders can be obtained in 10-ft increments from 30 to 60 ft

Figure 5-25

Special Concepts for Discharge Systems to Minimize Degradation (CEMA, 1979, p. 271)

action of vehicle traffic on the storage stockpile during placement and reclaim operations may also cause degradation.

Care must be taken to locate and design the surface storage piles to minimize erosion by surface water. Also, because the backfill materials will be subject to weathering for 50+ years before reclaim and backfilling, consideration of weathering characteristics will be necessary.

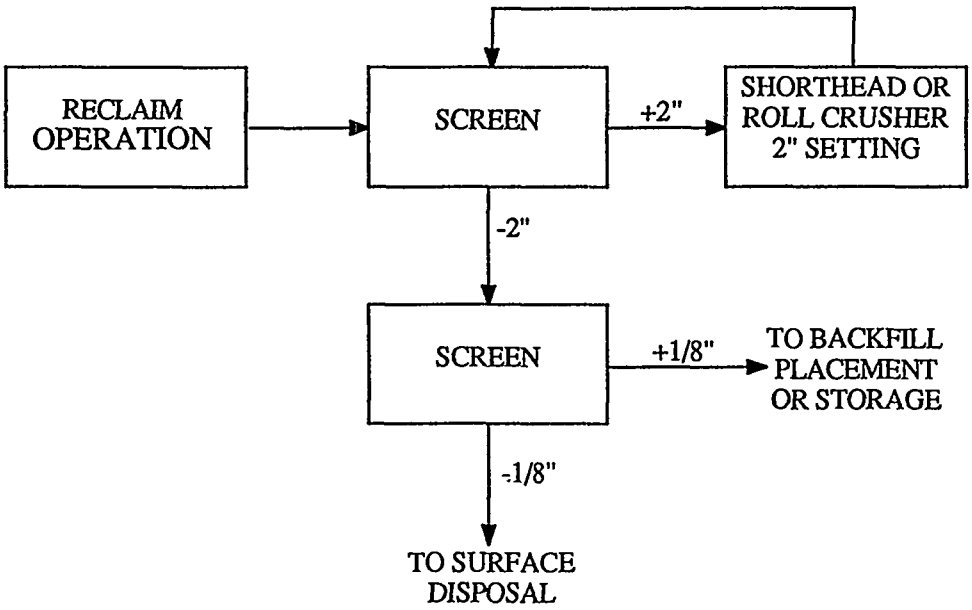
Sampling and study of outcrops of similar materials near the site will provide insight into the weatherability of the backfill and the need to provide additional protection from the elements during the caretaker period.

Prior to the start of backfilling operations, a system for reclaiming material from the pile and transporting it to the backfill plant will be designed and installed. Bucket wheel reclaimers, shovel/truck combinations, or rubber-tired loaders in combination with trucks or conveyor belts can be used.

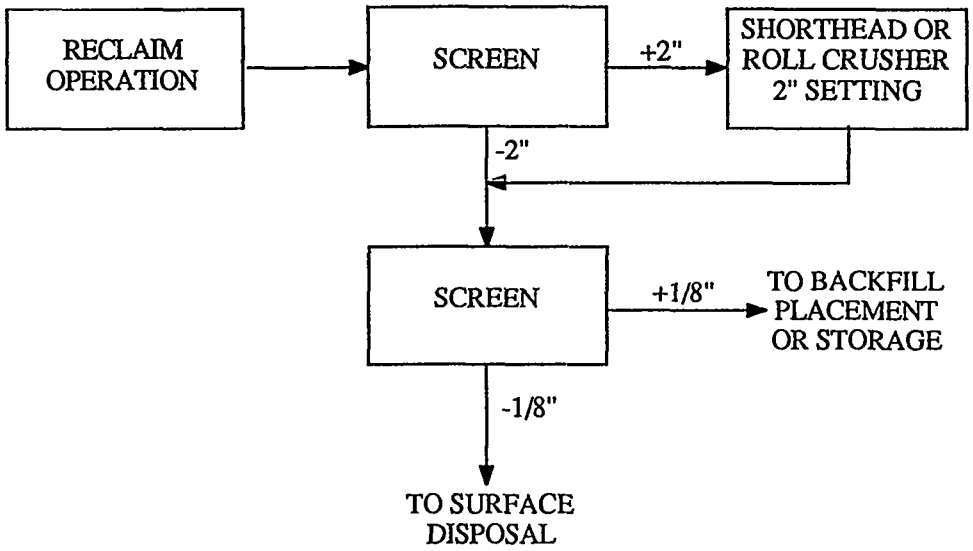
Backfill Plant. Because of the variation in mined material size and the various sources of size degradation discussed above, the rock reclaimed from the storage area will probably require processing prior to use as a backfill material. A sizing plant employing crushing and screening operations will produce a clean, sized backfill with the desired permeability characteristics.

A typical plant would include a screening step to separate oversized material from the stream of reclaimed material. The oversized material would be fed into a crushing system that would employ one or more stages of crushing to achieve the desired size reduction(s). Crushers for secondary size reduction applications can be categorized as gyratory, cone, roll, or impact-type machines. The selection of the specific type will depend on the capacity, reduction ratios, and product characteristics desired, as well as cost considerations. In general, gyratory- and cone-type crushers offer higher capacity advantages compared to roll or impact crushers with a similar capital cost. Roll and impact crushers, on the other hand, provide a more cubic product with lower levels of fines generation.

Figures 5-26a and b are simplified block diagrams for crushing systems operating in closed and open circuits, respectively. Closed-circuit operations (a) are employed where a maximum top size of the product is critical. The closed system requires additional



a) Closed Circuit System



b) Open Circuit System

Figure 5-26

Schematic Diagram of Crushing and Screening of Reclaimed Material for Backfilling

conveyor(s) to return the crushed product to the initial screening step and will also require the use of a higher capacity crusher and screen ahead of the crusher to handle the additional load of recirculated material. Open-circuit operations (b) are applied in situations where some flexibility in top size can be allowed. They require fewer pieces of equipment, and somewhat lower capacity screen and crushing units for similar system throughput capacity. Further screening operations will follow the crushing step(s) to separate unwanted fines and to segregate size fractions, if necessary. The plant will also include transport systems to handle and stockpile the product and reject streams. Stockpiling of the product stream will provide a surge supply that can be used to support downstream operations during periods of crusher maintenance, or to allow the continued operation of upstream systems while downstream facilities are out of service. Additional storage areas and reclaim operations may be necessary if surge capacity is required for product fill.

In addition to producing clean, sized material for general backfill, specialty materials may be handled at the backfill plant. Some finer earthen materials may be desired for use in graded filters, and some ultrafine materials may be required in areas with lower permeability requirements than the graded fill. Further size reduction of mined materials through the application of fine crushing or grinding will produce material with finer particle-size gradations that may be useful for filtering or sealing applications. Additional crushing stages or grinding systems and associated conveyors and equipment could be added to the backfill plant. Some imported materials may be required, including make-up aggregate, clay materials, and cements. The transportation system used for crushed tuff backfill could be designed with sufficient flexibility to handle other bulk materials. Rubber-tired equipment could supplement conveyor systems if additional flexibility is desired.

Transport and Emplacement of Sized Fill. Methods used to transport sized material underground will be similar to those for transport to the surface and will cause some additional degradation, as described earlier. While degradation occurring during transport to the surface is not a major concern if the material will be sized prior to backfilling, degradation of the sized material during decommissioning is a potential concern and requires careful consideration. Care must be taken to design conveyor speeds, transfer points, and discharge points to minimize degradation during transport.

The methods selected to emplace and compact fill underground can also influence the size distribution of the emplaced fill. Pneumatic conveying and stowing will cause considerable

degradation and fines production. In fact, dust may be a problem, requiring control by water sprays and ventilation during pneumatic stowing.

In the course of research performed by the USBM to assess pneumatic stowing as a method of backfilling abandoned mines (Masullo, 1990), the size degradation of backfill material was studied. Number 67 coarse aggregate was fed into the RALF and emplaced in the test chamber to simulate the mine backfilling process. The size distribution of the feed and the fill at the nozzle are compared in Figure 5-27. It can be seen that a significant fines fraction is produced by the pneumatic stowing process.

Some fines will also be produced during mechanical backfilling and compaction. Figure 5-28 shows test results performed on laterite gravels used for road aggregate (Gidagasu, 1991). In Figure 5-28a, the gravel degrades from the original gradation during laboratory compaction by two different methods. Figure 5-28b shows that some degradation also occurs during field compaction with a 10- to 12-ton vibratory roller, and that fines increase with the number of passes of the roller.

Conclusions. It is evident that all aspects of the material handling system affect backfill size control. Technology is available and routinely used by several industries to control the size characteristics of a wide variety of materials during bulk handling. The details of a specific system will be influenced by the ultimate specifications on the emplaced fill and on the handling characteristics of the material being handled. There are currently no indications that welded tuff will degrade or weather significantly during bulk transport. However, if future specifications require a very uniform fill size to maintain the natural permeability of the rock mass, careful design and testing will be required to develop a system capable of mining, transporting, storing, and emplacing to specifications.

5.4.2 Seal Durability

General. Civil engineering structures are typically designed for a 100-year life; few mines operate for more than 50 years. The history of structural failures due to material deterioration and environmental effects over these relatively short time periods emphasizes the importance of addressing durability in engineering design. A high-level waste repository will have a design life of 10,000 years, longer than recorded history. The issue of durability becomes more difficult to address over this very long time period, requiring geological evidence for natural materials or their analogs and geochemical modeling

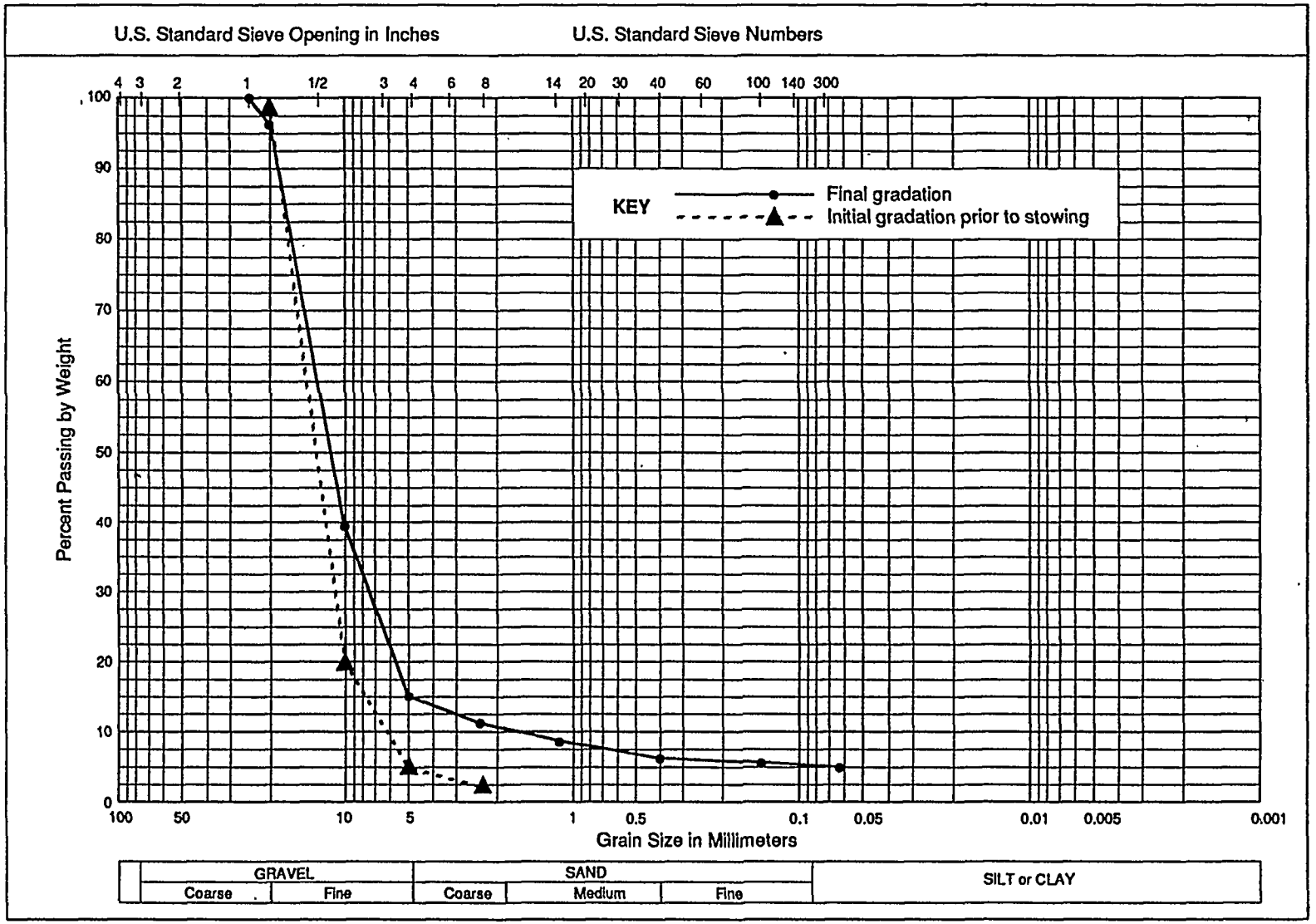
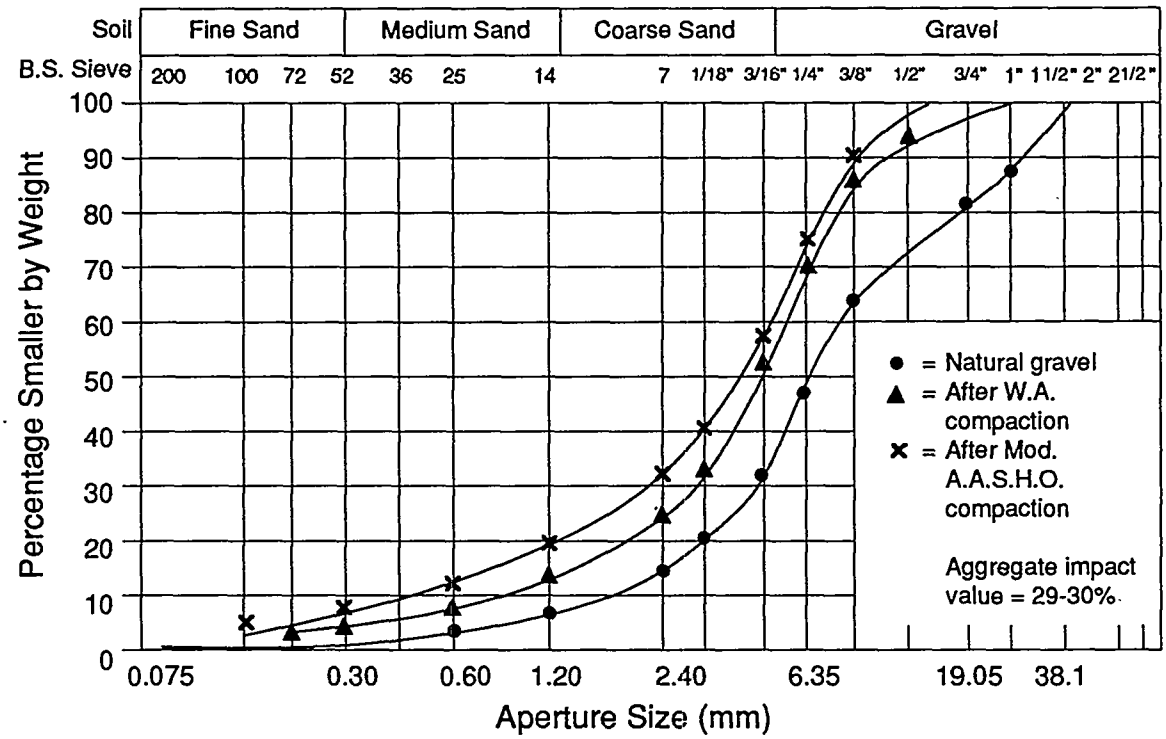


Figure 5-27

Grain Size Distribution of Fill Material as Fed into the System and Initial Gradation Prior to Stowing (after Masullo, 1990)

a)



b)

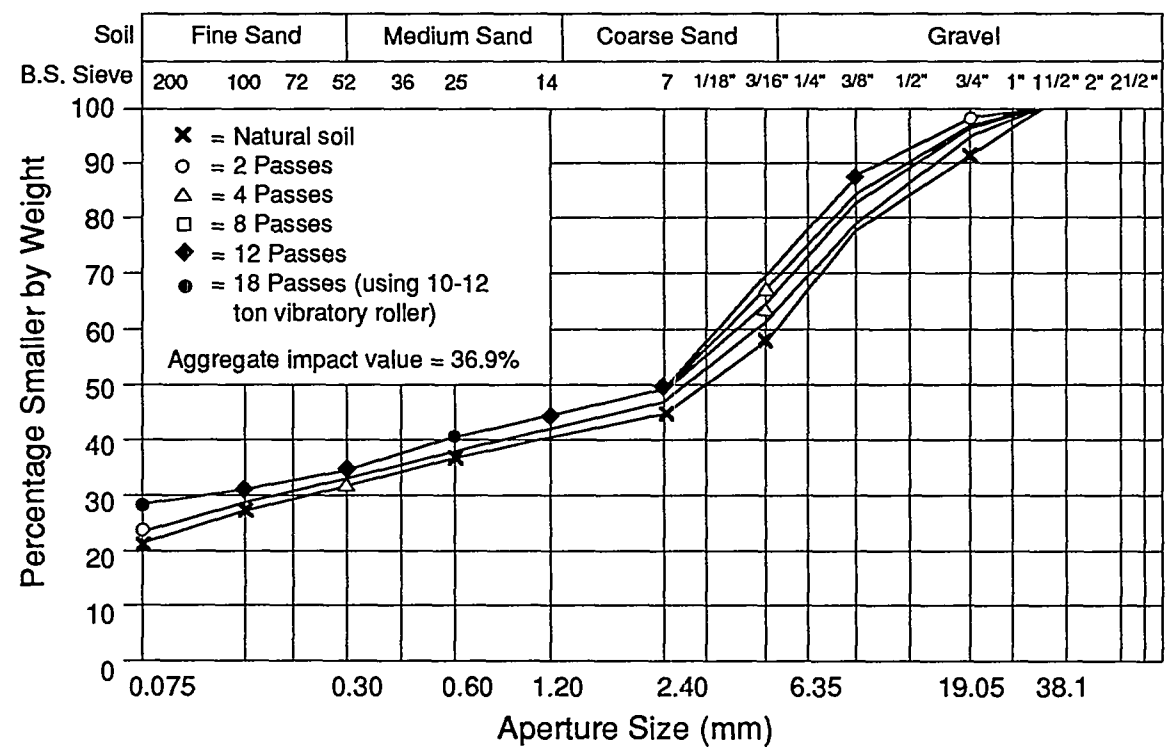


Figure 5-28

Effect of Laboratory and Field Compaction on Degree of Breakage of Lateritic Gravels (after Gidagasu, 1991)

(Hinkebein and Gardiner, 1991) when appropriate case histories do not exist. Fernandez et al. (1987) argue that repository sealing components need to function for only 500 to 1,000 years based on hydrologic considerations, which increases the confidence of durability predictions for these components. Durability requirements must consider the importance of the sealing component to overall repository performance. The use of self-healing seal designs and multiple-barrier concepts is recommended where significant performance is allocated to particular seals.

This discussion is important to this review of available technologies for two reasons: (1) the materials to be placed define the technologies being considered, and (2) the placement method of seals has an important impact on durability.

Environmental Factors. Environmental factors have a significant impact on the durability of engineering materials and structures. In the context of repository sealing, the following environmental conditions are important:

- thermal environment,
- geochemical environment,
- load environment, and
- erosion environment.

Table 5-1 shows some types of deterioration that could affect the performance of sealing components, along with the causative environmental conditions. Deterioration itself can be broadly classified as chemical or mechanical, with the former affecting the sealing materials and the latter affecting the integrity of sealing components made of these materials.

Material Considerations-Chemical Durability. As Gnirk (1987, p. 222) points out, "The most commonly accepted approach to determining long-term seal performance is to establish the long-term physical and chemical stability of the seal material itself. By inference, the seal functions will be achieved as long as the material remains intact or it can be shown that

Table 5-1
Deterioration of Sealing Components

Component	Material(s)	Possible Deterioration Mechanisms	Environmental Factors
General backfill	Coarse crushed tuff	Settlement	Seismic loads
		Degradation	Seismic loads, thermal expansion, chemical weathering
Graded fill components	Coarse and finely crushed tuff, clays	Settlement	Seismic loads
		Piping	Hydraulic gradients
		Shrinkage cracking (clays)	Elevated temperature, desiccation, chemical attack
Bulkheads	Cementitious	Cracking and spalling, creep	Thermal, seismic, in situ loads, shrinkage
		Weathering	Chemical attack
		Erosion	Hydraulic gradients
Grout curtains	Cementitious, clay	Erosion/migration	Hydraulic gradients
		Dissolution	Chemical attack

In a recent study specific to a potential repository in tuff (Fernandez et al., 1987), a material screening was performed (Figure 5-29), leading to the following groups of recommended materials:

- cementitious materials
  - portland cement-based concretes and grouts

- natural earthen materials
  - crushed tuffs,
  - crushed tuff and clay, and
  - crushed tuff with fines removed.

Cementitious Materials. Cementitious materials based on natural pozzolans were used over 2,000 years ago in Roman construction, and existing examples of Roman architecture attest to their durability. Portland cements have only been in use for approximately 150 years, but an enormous body of research findings and practical experience on durability has been developed. The durability of concretes depends greatly on whether an equilibrium can be achieved externally between the concrete and its emplacement environment, as well as internally between the constituents of the concrete.

Failure to achieve such a chemical balance has led to serious durability problems in civil projects (e.g., Legget and Karrow, 1982). Some of the more common problems are:

- sulfate attack,
- alkali-aggregate reaction,
- acid attack, and
- temperature strength reductions.

The strategy adopted in the YMP Repository Sealing Program develops cementitious materials that are similar in bulk chemistry to the tuffs to minimize the potential for deleterious chemical reactions. Furthermore, laboratory analyses conducted within the YMP Repository Sealing Program show that (1) calcium silicate hydrates, such as tobermorite, formed by reactions between silica and calcium hydroxide, are more stable than calcium hydroxide and exist at temperatures as high as 300°C; and (2) ettringite, a common expansive ingredient, is unstable at temperatures above 100°C (Scheetz and Roy, 1989). Buck et al. (1985) also show ettringite to be unstable at high temperatures.

Initial Material Screen

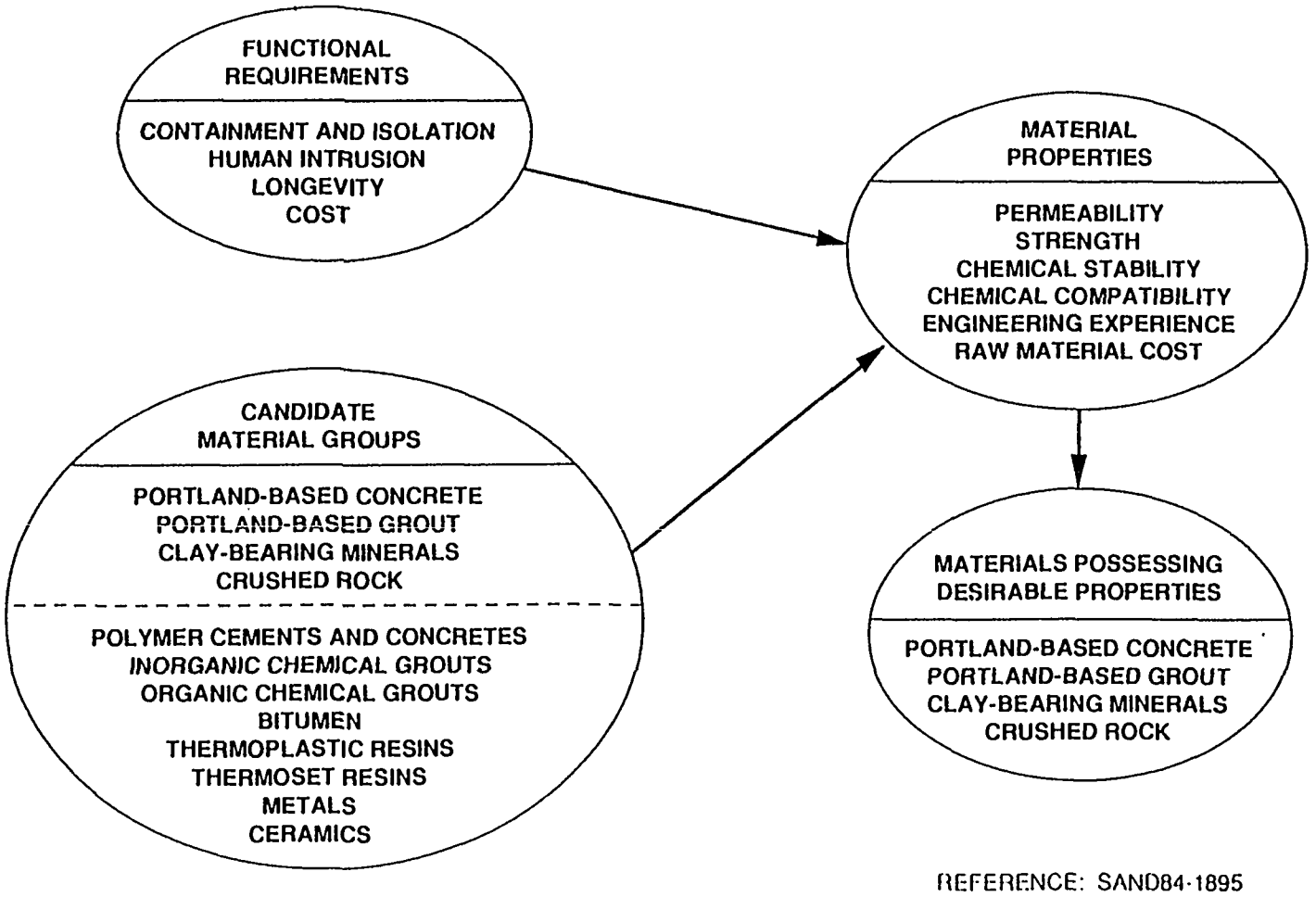


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Figure 5-29

Initial Screen for Repository Sealing Materials (after Fernandez et al., 1987)

Hinkebein and Gardiner (1991) use geochemical modeling to show that if ettringite or excess Portlandite are present, their decomposition can open the concrete structure and cause increased permeability. Accordingly, the YMP is currently considering concrete mixes that minimize the presence of ettringite and contain enough reactive silica to react with the calcium hydroxide to form the more stable calcium silica hydrates (Licastro et al., 1990). Pozzolans are reactive siliceous materials including fly ash and silica fume that may improve the stability and long-term performance of concrete by tying up soluble calcium and producing calcium silica hydrates.

Although work remains to be done, it is likely that durable cementitious mixtures chemically suitable for sealing a tuff repository can be developed. The current state of knowledge is perhaps best described by Gnirk (1987, p. 224):

> *"Although the longevity of cementitious materials under expected repository conditions is uncertain, there is no definitive evidence that the long-term performance is unacceptable. These materials perform most satisfactorily at temperatures below about 100°C, but may degrade under acidic, sulfate-rich, or carbon-dioxide-rich conditions, and under conditions that include thermal cycling. The stability of cementitious materials may be improved under adverse conditions by the use of sulfate-resistant cement, pozzolanic additives, and by producing high strength, low porosity and low permeability materials through high-quality workmanship."*

Earthen Materials. Crushed tuff, clays, and mixtures have desirable characteristics for sealing including durability. Fernandez et al. (1987, pp. 7-15) anticipate that crushed tuff will be geochemically stable under repository conditions. Clay minerals are known to be stable over a wide range of geologic environments over geologic time periods (Grim, 1962). Ran and Daemen (1991) note that the natural occurrence of bentonite minerals in and below the repository horizon reported by Bish et al. (1984a; 1984b) strongly supports the notion that these materials are geochemically stable in this environment.

The mining process results in a dramatic increase in the surface area of minerals, increasing their reactivity in the presence of groundwater. Acid mine drainage is a problem in many mining areas worldwide. It is possible that reactions between groundwater and crushed tuff may occur at an accelerated rate, and more research needs to be done.

Some clay minerals are known to react slowly with groundwater of certain compositions; this is more significant with regard to the potential for altering groundwater chemistry in the vicinity of the waste package rather than concerns about seal durability. Krumhansl (1986) reports the results of experiments performed under the salt repository program, indicating that the separate and combined effects of dehydration, irradiation, and exposure to hydrothermal solutions (brines) can lead to chemical changes in bentonites. However, Krumhansl concluded that the changes could be accommodated in seal design and sealing materials if the environments were correctly anticipated. With further work, suitable earthen materials and mixtures likely can be demonstrated to be chemically durable in the repository environment for the required period.

Sealing Components—Mechanical Durability. In addition to evaluating the sealing materials for chemical stability, the sealing components must also be evaluated for mechanical durability. This includes (1) settlement of backfill; (2) fines migration; (3) thermal and seismic stresses on bulkheads; and (4) dissolution, erosion, and migration of grout curtains.

1. Backfill Settlement

The general backfill is expected to settle naturally after emplacement. The settlement could be greater after the occurrence of one or more seismic events at the site. Fernandez et al. (1992) suggest settlement of less than 10% of the total height could be expected (Figure 5-30). The amount of settlement can be minimized by proper grading and compaction of the fill. Settlement effects are of more concern in shafts than they are in drifts, as the vertical extent is greater. Some size degradation of the fill could also occur by seismic action and thermal cycling, although the effect is expected to be minor. Graded-fill structures are also potentially vulnerable to differential settlement from seismic events.

2. Fines Migration

Deterioration of backfill structures by fines migration may present concern. In engineering practice, fines migration is of concern in dam construction using earthen materials. Three areas of concern are piping, permeability, and segregation of the material. These concerns are typically related to filter design. The intent in dam construction is:

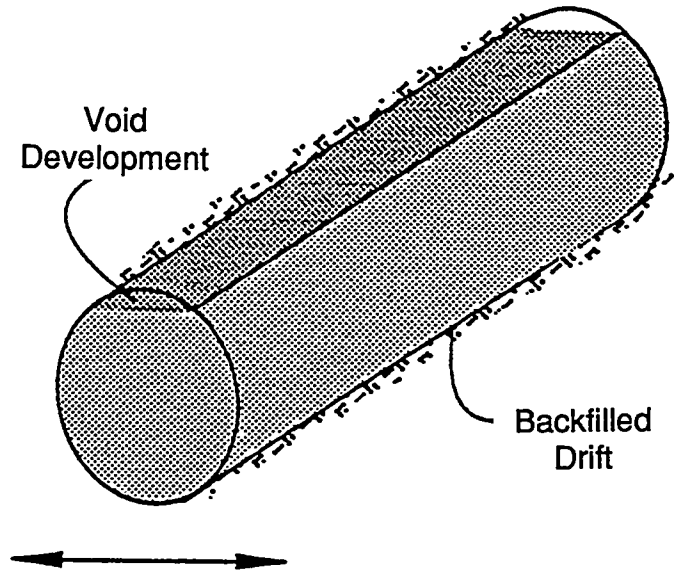


Figure 5-30
Settlement of Granular Materials

- to avoid deleterious movement of the finer particles (piping) that can cause instability of the rock fill;

- to ensure that the fluid moving through the rock fill does not build up excessive seepage or hydrostatic pressures, i.e., the filter has sufficiently high permeability to dissipate the fluids in the rock fill; and

- to avoid segregation between the coarse-sized fraction; the fine-sized fraction of the rock fill segregation would reduce the filter effectiveness.

In a well-graded, sandy gravel filter, which may be similar to repository-graded backfill structures, the larger particles are contained in a matrix of finer-sized particles. It is the finer-sized particles that govern the pore channel sizes through which the water flows, and determine the potential for particle movement and piping. Currently accepted filter criteria for piping prevention are based on the finer-sized fraction of the backfill. The following criterion is generally accepted as the filter for materials having a D_{15} larger than 1.0 mm:

$$D_{15} / D_{85} \leq 5 \quad , \tag{5-1}$$

where D_{15} = particle size in filter for which 15% by weight of the particles are smaller, and
D_{85} = particle size in the base soil for which 85% by weight of the particles are smaller.

This filter criterion is considered conservative by a factor of two and applicable to angular particles of crushed rock and to a filter that may be subjected to vibrations (Sherard et al., 1984a). Sherard also concluded that particles smaller than about 0.15 D_{15} that are carried in water suspensions will generally pass through the voids and out of the filter, and particles greater than about 0.12 D_{15} would be retained near the filter face. Additional criteria mentioned by Sherard et al. (1984b) suggested that for clays, sandy silts, and clays with some sand content and fine-grained clays (D_{85} of 0.03 to 0.1 mm), a sand or gravelly sand filter with average D_{15} not exceeding about 0.5 mm is reasonable and conservative. For fine-grained soils of low cohesion and exceptionally fine soils, a filter having an average D_{15} of 0.3 and 0.2 mm would be acceptable and conservative.

The permeability criterion for drainage to allow dissipation of fluids through the filter is

$$D_{15}\ (\text{filter}) / D_{15}\ (\text{base}) \geq 5 \quad . \tag{5-2}$$

This relationship is roughly equivalent to the hydraulic conductivity of the filter being approximately 25 or more times the hydraulic conductivity of the base soil (Perry, 1987). To avoid segregation of the filter material, the USBR recommends the maximum size of the filter aggregate should be less than 3 in. to minimize the potential for segregation and bridging of the large particles during placement (Cedergren, 1967).

3. Thermal and Seismic Effects

Thermal effects may cause desiccation, shrinkage, and cracking in emplaced clay components. Because of their stiffness and good mechanical coupling with the rock walls, cementitious plugs are more vulnerable to mechanical damage than general fills. Induced thermal stresses can load the plugs and cause creep and cracking.

Earthquake loads can also load cementitious seals. A concrete gravity dam in India was damaged by a 6.5 magnitude earthquake, which caused major cracking and required

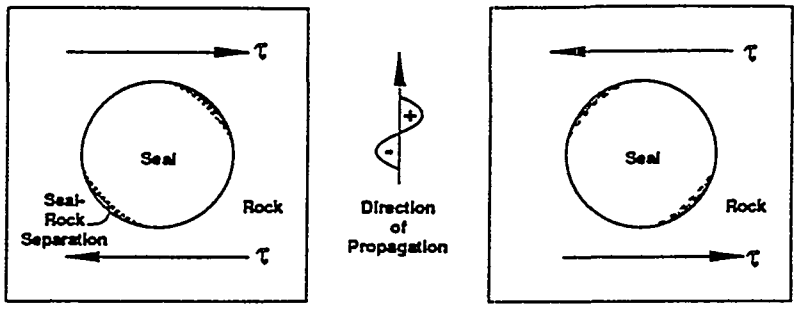
extensive repairs (Chopra, 1973). This free-standing structure would be expected to be more susceptible to damage than a concrete seal underground. Fernandez et al. (1992) performed some preliminary calculations suggesting that seismic loading from a single design basis event at Yucca Mountain may cause minor separation of the plug-rock interface, possibly increasing plug permeability if the separation remains open after the event (Figure 5-31). Preloading the plug radially by contact grouting or expansive cement materials will reduce or eliminate this problem. Although no adverse effects on current plug performance goals from seismic effects were predicted, consideration must be given to seismic effects in plug design.

If leakage occurs in the presence of a high hydraulic gradient, erosion of the cementitious plug will be relatively rapid. Because it is expected that only minor amounts of water will be encountered and the pressure generated will be rapidly dissipated, this mechanism is not likely to lead to plug failure, unless the natural permeability of the rock mass is substantially reduced.

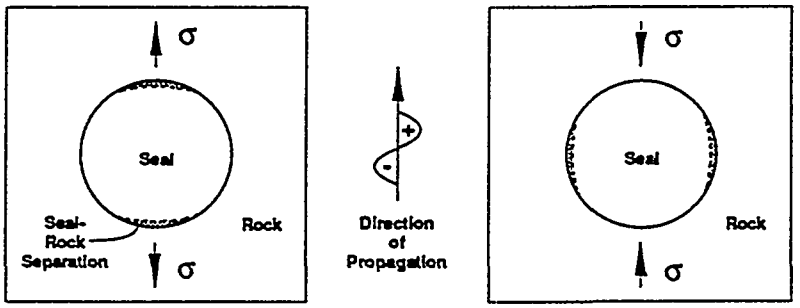
4. Dissolution/Erosion/Migration

Grout curtains are subject to dissolution, erosion, and migration, particularly under conditions of high hydraulic gradients. Houlsby (1990, p. 93) describes several case histories where cutoff grout curtains under dams deteriorated over time. At the Burrinjuck Dam, an old concrete gravity dam that was grouted over the period from 1936 to 1948, very thin mixtures were used (average 12:1). Much of this grout leached into a foundation tunnel under the dam (Figure 5-32a). It was recently estimated that the volume of calcium formations in the tunnel is approximately equal to the amount of grout originally placed, indicating near complete removal of the grout curtain. The mechanism involved dissolution and reprecipitation of calcium; biological attack of the grout from algae was also noted. At the Blowering Dam, an earthfill dam, the cutoff grout curtain has slowly disintegrated, causing a rise in the water table level (Figure 5-32b). Houlsby recommends water/cement ratios of no more than 3:1 to improve durability of grout curtains.

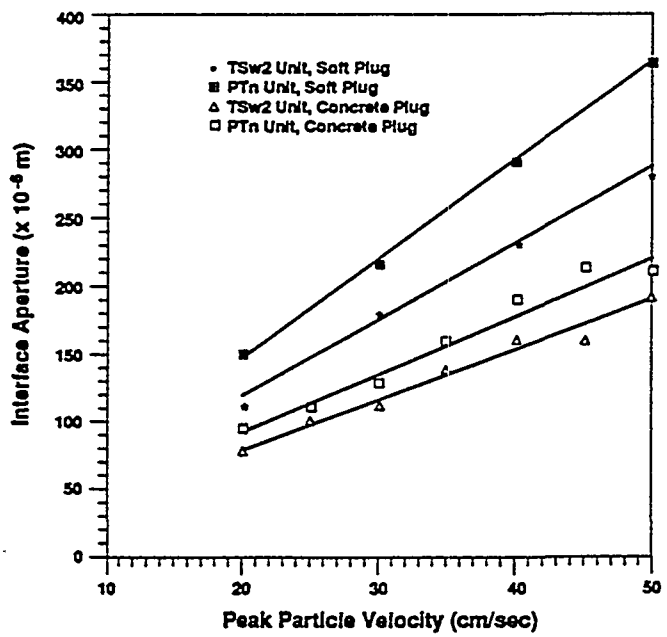
High hydraulic gradients across repository grout curtains would be considered off-normal. However, this information illustrates the effect of the water-to-cement ratio on grout curtain durability in some civil applications.



a) Shear Wave-Induced Deformation



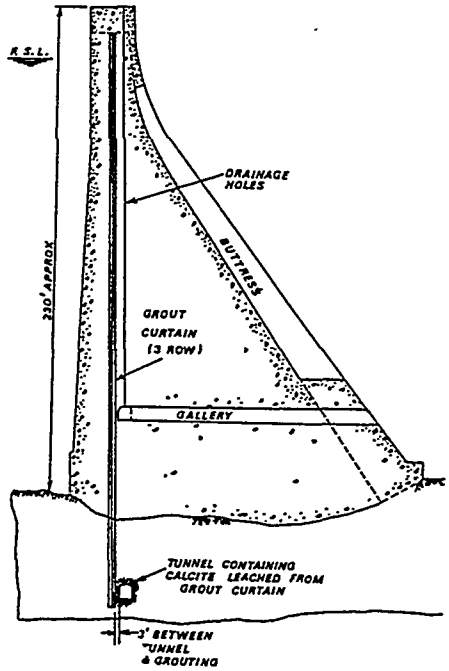
b) Compressional Wave-Induced Deformation



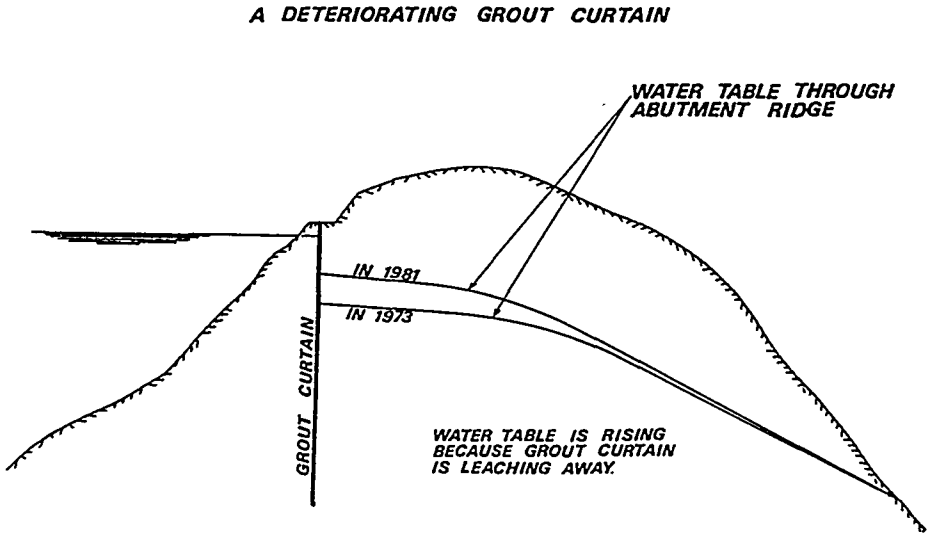
c) Aperture of Interfaces Under Various Loading Conditions in the PTn and TSw2 Units

Figure 5-31

Separation Around Cement Plug Caused by Seismic Loading



a) Burrinjuck Dam Cross Section, Showing Tunnel Into Which Grout Curtain Was Leached (after Houlsby, 1990, p. 93)



b) Blowering Dam Cross Section Showing Rise in Water Table from Dissolving Grout Curtain (after Houlsby, 1990, p. 94)

Figure 5-32

Examples of Cementitious Grout Curtain Dissolution Under High Hydraulic Gradients

6.0 Conclusions

The purpose of this report is to determine whether available technologies exist to seal a potential underground repository at Yucca Mountain. Technologies needed in the YMP Repository Sealing Program fall into three areas: preconditioning of the emplacement area, seal component placement, and seal component durability. These three areas apply to backfill (general and graded fills), bulkhead, and grout placement. Conclusions on the availability of these technologies are summarized below. These conclusions are based on case histories from the civil, mining, and defense industries.

Conclusion 1

Technology for placement of general backfill under moderate temperature conditions (<100°F) in underground openings is currently available and is being routinely used in dozens of underground operations. Crushed tuff of any desired gradation can be prepared in a surface backfill plant and can be transported to the placement area. Mechanical or throwing methods are the preferred technologies for repository backfilling. Limitations for pneumatic stowing include limited horizontal transport distances, dust, and wear inside the feeder and transport pipes. A limitation for mechanical placement is the ability to place backfill tightly against the roof. The technology exists and is routinely used to place and compact backfill on surface to very exacting specifications during civil construction. If very stringent specifications are established for repository backfill, such as very tightly compacted fill against the roof, then new technology may have to be developed because of limited clearance underground.

Discussion

This conclusion applies to the processing, mixing and blending, transportation, and placement of materials. Case histories presented in this report substantiate this conclusion. Technologies evaluated included mechanical emplacement and compaction, throwing (pneumatic and slinger), gravity, and hydraulic methods. The most applicable technologies for the YMP are pneumatic and mechanical compaction. It was noted that the mechanical emplacement method used for the Cannon Mine (Appendix A.1) could readily be applied for backfilling the Yucca Mountain repository. The lower half of the repository drifts could be filled with crushed tuff unloaded from either LHDs or tele-dump trucks. In a second pass, the upper half could be filled. Both mechanical and pneumatic stowing of backfill used at the Billie Mine (Appendix A.2) could be directly applied to backfill placement at

Yucca Mountain. The mechanical emplacement method could be directly applied, with the crushed tuff filled and compacted in lifts. The pneumatic system could also be directly applied, using sized and screened crushed tuff instead of waste rock. One limitation of the pneumatic backfill system is the limited ability to transport fill horizontally (2,000 ft). An additional concern is the potential wear that could occur inside the RALF and transport pipes. Experience obtained at other locations (especially Sullivan Mine [Appendix A.4] and Castlegate Mine [Appendix A.5]) is also applicable to Yucca Mountain. Experience at the Lake Lynn Laboratory (A.9) suggests that backfill can be pneumatically placed using simple feeders with lower maintenance requirements than the RALF. Equipment for remote monitoring of placement is also under development.

Conclusion 2

Technology exists for placement of large-scale bulkheads in underground drifts and shafts under moderate temperature conditions. Quality control methods are available to verify engineering properties during the placement of cementitious seals.

Discussion

Numerous types of plugs have been placed underground, including inundation plugs, ventilation stopings, hydraulic fill containment bulkheads, abandonment bulkheads, consolidation bulkheads, and "conversion" bulkheads. Some plugs, such as the inundation plugs used in the West Driefontein Mine in South Africa, are designed to withstand very large pressures (Garrett and Campbell-Pitt, 1961; Cousens and Garrett, 1969). An experimental plug constructed here withstood pressures of up to 6,800 psi without structural failure, despite the lack of a keyway. Hydraulic fill containment bulkheads, which are used to retain fill during drainage and consolidation of hydraulic fill, are used in the Mount Isa Mine in Australia (Grice, 1989). On a smaller scale, abandoned bulkheads are used in the Eagle Mine (Appendix C.3) to retain mine drainage water and prevent human intrusion.

In civil applications, dam diversion plugs and hydropower tunnel plugs are commonly constructed. Hydropower tunnel plugs can experience high pressure gradients. At numerous dams, including the Foster Reservoir Dam in Oregon (Department of the Army, 1978), tapered, dam-diversion plugs were installed under the dam axis to permanently seal the diversion tunnel prior to filling the reservoir. The Helms Pumped Storage Project (Appendix B.1) is used to illustrate placement of a hydropower tunnel plug.

In the defense industry, considerable amounts of cementitious materials have been placed to withstand extremely high temperatures and pressure resulting from detonations of nuclear weapons. The stemming material (bulkhead) is designed to withstand 1,000 psi and 1000°F loads for very short time periods. Perhaps the most significant contribution of bulkhead placement from defense applications is the quality control used prior to and during plug placement.

Conclusion 3

Technology exists to place grout into a variety of media including fractured rock masses at moderate temperatures. However, grouting operations remain more of an art than a science.

Discussion

Literally thousands of grouting operations have been performed including placement of chemical-, cementitious-, and clay-based grouts. Of primary importance to the YMP Repository Sealing Program are the cementitious- and clay-based grouts. The application of cementitious grouts is extensive in USBR projects, starting with the Hoover Dam in 1932. Most recently, the USBR used cementitious grouts in constructing the New Waddell Dam (Appendix B.2). Aside from the extensive use of cementitious grouts to seal fractured tuff and other rock types here, a computerized system was employed to optimize the placement of the grout. The case history associated with the Helms Pumped Storage Project demonstrated the effective sealing of a shear zone using microfine cement. It is possible that similar shear zones will be encountered at Yucca Mountain, although flows are anticipated to be considerably less due to the presence of the repository in the unsaturated zone. The case history of the Vat Tunnel (Appendix B.3) also illustrates the ability to emplace grout under adverse conditions of high formation water pressure and high volumes of water. While conditions of this nature are not anticipated at Yucca Mountain, this case history infers the comparative ease in grouting a more benign site like Yucca Mountain. Finally, primarily cementitious grouts and microfine cement grouts have been demonstrated extensively at the NTS. Because of the extensive placement of grout, an empirical understanding has been obtained between the properties of the grout and composition of the grout. Again, quality control measures are extensively used to obtain consistent and high-quality grout curtains.

Conclusion 4

Technologies exist to precondition areas where seal components are to be placed. This includes shaft liner removal, blast-damaged zone removal, rockbolt removal, concrete floor removal, and slot cutting and borehole drilling in rock to enhance drainage.

Discussion

There are two aspects of preconditioning considered in this report. The first is the removal of rock to develop keyways and specialty features (drainage holes). Second is the removal of materials introduced into the repository. The first can be achieved through standard drill-and-blast techniques. The largest volume of material potentially requiring removal from the repository includes concrete and rockbolts. Concrete typically will be used for the roadway and the shaft liner. If placed as a continuous pour, it can be removed using hand-held pneumatic breakers, drill and blast methods, hydraulic splitters, impact breakers, and roadheaders. While the technology exists for rockbolt removal, this removal has never been demonstrated in a full-scale operation associated with underground development; nor has the need for rockbolt removal in a repository been demonstrated. Although the equipment for removing ground support components could be relatively easily developed from available technology, the operation may be difficult and hazardous in deteriorated ground conditions, possibly requiring development of special technologies.

Enhancement of drainage can be accomplished through the use of slot cutting and borehole drilling. The drill-and-broach method can be considered available technology. The diamond saw, currently used as a research tool, may need to be converted to a truck-mounted production unit. This may require some developmental efforts. Currently, the need to enhance drainage by slot cutting and borehole drilling has not been demonstrated.

Conclusion 5

Technology that demonstrates the long-term durability and performance of seal components and is specifically relevant to Yucca Mountain does not currently exist.

Discussion

Typically civil engineering structures are designed for a 100-year life, mining structures less than 50 years, and defense seals on the order of seconds. A high-level waste repository, on the other hand, will have a design life of 10,000 years. In Fernandez et al. (1987), it is argued that repository sealing components need function for only 500 to 1,000 years based

on hydrologic considerations. As durability requirements for sealing components are revised, they must be evaluated considering the entire system. The use of self-healing seal designs and multiple barrier concepts are recommended where significant performance is allocated to seals. Chemical and mechanical durability of sealing components must be investigated because these components are part of the engineered barrier system, which will contribute to the overall performance of the repository.

Conclusion 6

Case histories do not currently exist that adequately document sealing component placement or performance under greatly elevated temperatures or radiation environments. In extreme cases, remote control equipment or robotics may be required to place seals.

Discussion

With any repository design there may be specific locations where seals must be placed under adverse environmental conditions. This is more likely with above-boiling design and in-drift emplacement concepts.

Conclusion 7

Specific material properties for sealing materials appropriate to Yucca Mountain are not available.

Discussion

Prior to assessing long-term performance of sealing components, it is essential to understand the as-placed (initial) properties of sealing components. For backfill, properties, such as density, void ratio, porosity, and shear modulus, have been determined from large-scale testing for uncemented and cemented rock fill. The work by Stout and Friel (1982) illustrates a range of properties of relatively low-strength backfill. Al-Hussaini (1981) performed tests on crushed basalt evaluating triaxial strength as a function of varying particle-size distributions. Additionally, some large-scale testing of crushed materials has been performed to obtain design information for earth-fill dam construction. However, the data from these testing programs can only be used in a very general way. It is strongly recommended that large-scale testing be performed on crushed tuff, both welded and nonwelded, to obtain the properties needed for performance and design calculations.

Fernandez et al. (1993) describe two sets of tests to address performance issues associated with backfill placement. These are the surface backfill tests and the underground backfill tests. The surface backfill test would obtain a preliminary understanding of rock fill performance and particle breakage during excavation. Quality control measurements would be made of the moisture content, density, gradation, and settlement of the backfill. The underground backfill tests would require an engineered backfill with specified properties, and evaluation of the coupled response of the rock fill and rock to simulated thermal loads. Heaters would be emplaced around the drift to simulate both far- and near-field thermal loads.

Despite the general availability of technology to place bulkheads, it is recommended that specific tests be performed to develop an understanding of bulkhead performance. Typically, in mining and underground applications, the successful analysis, design, and construction of bulkheads and grout curtains results in the reduction of water inflows for mining operations. In constructing large-scale seal components in an underground nuclear waste repository, these components may need to satisfy water flow or airflow performance objectives. Several large-scale bulkheads and grout placement tests are proposed by Fernandez et al. (1993) that would evaluate the performance of bulkheads and grout curtains under anticipated and extreme conditions. Additionally, quality control procedures for placing these sealing components would be developed.

It is concluded that a review of available technologies as provided by this document is not in itself sufficient to resolve several issues. These issues include long-term durability and performance of sealing components; the as-placed, in situ properties of sealing components; and the placement of sealing components under potentially high-thermal and high-radiation environments, and structurally unstable underground openings due to removal of rockbolts and concrete liners.

7.0 References

Adam, M. E., C. E. Brechtel, and J. F. T. Agapito (1986). "Improved Stopping, Door and Overcast Construction for Oil Shale Mines," USBM Contract J0145020, Mining Research Contract Report. U.S. Department of the Interior, Bureau of Mines, Washington, D.C. (NNA.940505.0033)

Al-Hussaini, M. (1981). "Effect of Particle Size and Strain on the Strength of Crushed Rock, Application of Rock Mechanics to Cut-and-Fill Mining," *Proceedings, Conference on the Application of Rock Mechanics to Cut-and-Fill Mining,* Lulea, Sweden. The Institution of Mining and Metallurgy, London, England. (NNA.940509.0006)

Allan, M. L., and L. E. Kukacka (1993). "In-Situ Containment and Stabilization of Buried Waste Annual Report," *BNL-49709.* U.S. Department of Energy, Washington, D.C. (MOL.19940714.0118)

Auld, F. A. (1983a). "Design of Underground Plugs," *International Journal of Mining Engineering,* October, pp. 189-228. (NNA.890808.0054)

Auld, F. A. (1983b). "Concrete in Underground Works," *Technical Paper No. 105.* The Concrete Society, London, England, 34 pp. (NNA.940509.0008)

Baker, W. H. (1982). "Improved Design and Control of Chemical Grouting," *U.S. Department of Transportation Report FHWA/RD-82/038,* Washington, D.C. (MOL.19940718.0002)

Barrett, J. R., J. E. Stewart, and J. G. Brock (1983). "Cemented Gravel Fill for a Small Underground Mine," *Proceedings of the International Symposium on Mining with Backfill,* Lulea, Sweden. A. A. Balkema, Rotterdam. (MOL.19940718.0014)

Baz-Dresch, J. J. (1987). "High Strength Backfilling at the Cannon Mine," *Proceedings, 3rd Western Conference on Precious Metals, Coal and Environment,* Rapid City, SD. Society of Mining Engineering, Littleton, CO, pp. 7-14. (NNA.940509.0009)

Beck, L. A. (1981). "Sealing an Underground Mine in Pennsylvania," *Mining Congress Journal,* July, Vol. 67, No. 7, pp. 37-44. (NNA.940608.0308)

Berdal, B., B. Bven, and J. Johanson (1985). "Lake Tap The Norwegian Method," *Tunnelling 85.* The Institution of Mining and Metallurgy, London, England, pp. 159-163. (NNA.940509.0010)

Berry, P. (1981). "Geomechanical Investigations for the Design of Cemented Fill," *Applications of Rock Mechanics to Cut-and-Fill Mining.* The Institution of Mining and Metallurgy, London, England. (NNA.940509.0011)

Bish, D. L., D. T. Vaniman, R. S. Rundberg, K. Wolfsberg, W. R. Daniels, and D. E. Broxton (1984a). "Natural Sorptive Barriers in Yucca Mountain, Nevada, for Long-Term Isolation of High-Level Waste," *Radioactive Waste Management. Proceedings of an International Conference, Seattle, IAEA-CN-43/461,* International Atomic Energy Agency, Vienna, pp. 415-432. (NNA.890602.0025)

Bish, D. L., A. E. Orgard, D. T. Vaniman, and L. Benson (1984b). "Mineralogy and Groundwater Geochemistry of Yucca Mountain Tuffs," *Scientific Basis for Nuclear Waste Management VII, Materials Research Society Symposia Proceedings.* Vol. 26, G. L. McVay, Editor, North-Holland, New York. (NNA.891016.0134)

Brechtel, C. E., M. P. Hardy, J. Baz-Dresch, and J. S. Knowlson (1989). "Application of High Strength Backfill at the Cannon Mine," *Proceedings of the 4th International Symposium on Mining and Backfill,* Montreal, Canada. A. A. Balkema, Rotterdam. (NNA.920819.0010)

Brechtel, C. E., D. A. Thies, and J. R. Aggson (1985). "Additives for Strengthening Backfill Materials in Steep Coal Seam Mining," J. F. T. Agapito & Associates, Inc., A Mining Research Contract Report, Contract No. J0245006. U.S. Department of the Interior, Bureau of Mines, Washington, D.C. (NNA.940505.0034)

Bruce, D. A. (1989). "Contemporary Practice in Geotechnical Drilling and Grouting." Keynote Lecture, First Canadian International Grouting Seminar, Toronto, Canada, April 18. (NNA.940505.0035)

Buck, A. D., J. P. Burkes, and T. S. Poole (1985). "Thermal Stability of Certain Hydrated Phases in Systems Made Using Portland Cement," *Miscellaneous Paper SL-85-7*, U.S. Department of the Army, Waterways Experiment Station, Corps of Engineers. (SRX.870819.0204)

Buttiens, E. (1977). "Conversion of Abandoned Collieries in Southern Belgium in a Low-Pressure Gas-Storage, with Description of Special Plugging of the Various Shafts," *RockStore 77. Proceedings of the First International Symposium,* Stockholm, Sweden. (NNA.940509.0012)

Cedergren, H. R. (1967). *Seepage, Drainage and Flow Nets.* John Wiley & Sons, Inc., New York, NY. (MOL.19940714.0110)

Charles, J. A., and K. S. Watts (1980). "The Influence of Confining Pressure on the Shear Strength of Compacted Rockfill," *Geotechnique,* Vol. 30, No. 4, pp. 353-367. (NNA.920817.0136)

Charles, J. A., and M. N. Soares (1984). "Stability of Compacted Rockfill Slopes," *Geotechnique,* Vol. 34, No. 1, pp. 61-70. (NNA.920819.0008)

Chekan, G. J. (1984). "Design of Bulkheads for Controlling Water in Underground Mines," *IC-9020.* U.S. Department of the Interior, Bureau of Mines, Washington, D.C., 36 pp. (NNA940509.0013)

Chekan, G. J., J. K. Richmond, J. M. Listak, and M. J. Sapko (1983). "Investigation of an Explosion-Proof Bulkhead That Acts as a Water Seal in Underground Coal Mines," Internal Report. U.S. Bureau of Mines, Pittsburgh Research Center, Pittsburgh, PA, October. (NNA.940509.0014)

Chopra, A. C. (1973). "Evaluation of Ground Shaking Effects on Concrete Gravity Dams," *Proceedings, Conference Inspection, Maintenance and Rehabilitation of Old Dams,* Pacific Grove, CA. American Society of Civil Engineers, Littleton, CO, pp. 771-785. (NNA.940509.0015)

Church, A. K. (1981). *Excavation Handbook.* McGraw-Hill Book Company, New York, NY, pp. 14-27. (MOL.19940714.0111)

Clarke, W. J. (1987). "Microfine Cement Technology, "*23rd International Cement Seminar,* Atlanta, GA, December 6-9, 13 pp. (NNA.940509.0016)

Code of Federal Regulations, Title 30, Chapter 1, Part 75, Subchapter D, Subparts 329-1, 329-2, 330, and 330-1. (HQX.870301.4730)

Conveyor Equipment Manufacturers Association (CEMA) (1979). *Belt Conveyors for Bulk Materials,* 2nd Edition. Prepared by the Engineering Conference of the Conveyor Equipment Manufacturers Associations, CBI Publishing Company, Inc., Boston, MA, 346 pp. (NNA.890713.0210)

Cousens, R. R. M., and W. S. Garrett (1969). "The Flooding of the West Driefontein Mine," *Journal of the South African Institute of Mining and Metallurgy.* Published jointly by the South African Institute of Mining and Metallurgy, Johannesburg, South Africa, and the 9th Commonwealth Mining and Metallurgical Congress, London, England, Vol. 69, No. 8, pp. 421-463. (NNA.940509.0017)

Cowling, R., A. Voborll, L. T. Isaacs, J. L. Meek, and G. Beer (1989). "Computer Models for Improved Fill Performance," *Proceedings of the 4th International Symposium on Mining and Backfill,* Montreal, Canada. A. A. Balkema, Rotterdam, pp. 165-174. (NNA.940509.0018)

Cummins, A. B., and I. A. Given, Editors (1979). *SME Mining Engineering Handbook.* Society of Mining Engineers of the American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., New York, NY. (NNA.910327.0009)

DeJongh, C. L., and A. N. Morris (1988). "Considerations in the Design and Operation of Backfill Systems for Narrow Tabular Orebodies," *Backfill in South African Mines.* South African Institute of Mining and Metallurgy, Johannesburg, South Africa. (NNA.940509.0019)

Deere, D. U. (1982). "Cement-bentonite Grouting for Dams," *Grouting in Geotechnical Engineering. Proceedings of the American Society of Civil Engineers Conference,* New Orleans, LA, February. (HQZ.870228.5916)

Department of the Army (1978). *Engineering Manual and Design Tunnels and Shafts in Rock. Engineer Manual EM 1110-2-2901,* Washington, D.C. (NNA.900122.0016)

Dight, P. M., and R. Cowling (1979). "Determination of Material Parameters in Cemented Fill," *Proceedings of the 4th International Congress on Rock Mechanics,* Montreaux, Switzerland. A. A. Balkema, Rotterdam. (NNA.940629.0008)

Djahanguiri, F., and F. A. Mahtab (1983). "State of the Art Pneumatic Backfilling and Its Application to a Nuclear Waste Repository in Salt," *Proceedings of the International Symposium on Mining with Backfill,* Lulea, Sweden. A. A. Balkema, Rotterdam. (NNA.940509.0020)

Donaghe, R. T., and M. W. Cohen (1978). "Strength and Deformation Properties of Rockfill," *Technical Report S-78-1.* U.S. Department of the Army, Corps of Engineers, Waterways Experiment Station, Vicksburg, MS. (NNA.921218.0012)

Dravo Engineers, Inc. (Dravo) (1984). "Effect of Variations in the Geologic Data Base on Mining at Yucca Mountain for NNWSI," *SAND84-7125.* Sandia National Laboratories, Albuquerque, NM. (NNA.870406.0467)

Dressel, W., and J. S. Volosin (1985). "Inverted Pyramid-Shaped Plugs for Closing Abandoned Mine Shafts Galina, KS, Demonstration Project," *IC-8998.* U.S. Department of the Interior, Bureau of Mines, 14 pp. (NNA.940505.0038)

Dunn, J. R., G. M. Banino, W. Ernst (1977), "The Physical and Chemical Characteristics of Available Materials for Filling Subsurface Coal Mines," U.S. Bureau of Mines Open File Report, Contract J0155182, USOI, 281 pp. (MOL.19940714.0119)

Estivalet, J. (1984). "Les Serrements Dams les Mines Souterraines: Determination, Techniques, et Exemples," *Ind Miner Tech.*, St. Etienne, France, No. 8, 1984, pp. 641-651. (NNA.940505.0039)

Ewart, F. (1985). *Rock Grouting With Emphasis on Dam Sites.* Springer-Verlag, Berlin, Germany. (MOL.19940714.0112)

Faure, M. (1988). "Experimental Backfilling at Harmony Gold Mine, Backfill in South African Mines," *Backfill in South African Mines.* South African Institute of Mining and Metallurgy, Johannesburg, South Africa. (NNA.940505.0046)

Fernandez, J. A., J. B. Case, and J. R. Tyburski (1993). "Initial Field Testing Definition of Subsurface Sealing and Backfilling Tests in Unsaturated Tuff," *SAND92-0960.* Sandia National Laboratories, Albuquerque, NM. (NNA.930512.0001)

Fernandez, J. A., A. M. Richardson, and M. Lin (1992). "Seismic Considerations in Sealing a Potential High-Level Radioactive Waste Repository," *American Society of Civil Engineers Symposium on Dynamic Analysis and Design Consideration for High-Level Nuclear Waste Repositories,"* August 19-20, San Francisco, CA, pp. 215-237. (NNA.920824.0096)

Fernandez, J. A., P. C. Kelsall, J. B. Case, and D. Meyer (1987). "Technical Basis for Performance Goals, Design Requirements, and Material Recommendations for the NNWSI Repository Sealing Program," *SAND84-1895.* Sandia National Laboratories, Albuquerque, NM. (NNA.870717.0048)

Fernandez, J. A., and M. D. Freshley (1984). "Repository Sealing Concepts for the Nevada Nuclear Waste Storage Investigations Project," *SAND83-1778.* Sandia National Laboratories, Albuquerque, NM. (NNA.870407.0030)

Foster-Miller Associates, Inc. (1974). "Review of Past Research and Development in Permanent Explosion-Proof Bulkheads," Draft Report to the U.S. Department of the Interior, Bureau of Mines, Pittsburgh, PA, August. (NNA.940505.0040)

Garrett, C. R. (1985). "Development of a Roadheader Mining System at American Borate Company," *Proceedings, Rapid Excavation and Tunnelling Conference,* New York. Society of Mining Engineers of the American Institute of Mining Engineers, Littleton, CO. (NNA.940505.0041)

Garrett, W. S., and L. T. Campbell-Pitt (1961). "Design and Construction of Underground Bulkheads and Water Barriers," *Transactions of the Seventh Commonwealth Mining and Metallurgical Congress.* South African Institute of Mining and Metallurgy, Johannesburg, South Africa, Vol. III, pp. 1283-1299. (NNA.940505.0043)

Garrett, W. S., and L. T. Campbell-Pitt (1958). "Tests on an Experimental Underground Bulkhead for High Pressures," *Journal of the South African Institute of Mining and Metallurgy,* Johannesburg, South Africa, October, pp. 123-143. (NNA.940505.0042)

Geochemical Corporation (1990). "Product Specification--Microfine Cement," 162 Spencer Place, Ridgewood, NJ. (NNA.921021.0067)

Gertsch, R., and L. Ozdemir (1991). "Performance Prediction of Mechanical Excavators in Yucca Mountain Welded Tuff, From Linear Cutter Tests," *SAND91-7038.* Sandia National Laboratories, Albuquerque, NM. (NNA.920819.0247)

Gidagasu, M. D. (1991). "Characterization and Use of Tropical Gravels for Pavement Construction in West Africa, "*Geotechnical and Geological Engineering,* August, Vol. 9, pp. 219-260. (NNA.940629.0004)

Gnirk, P. (1987). "State-of-the-Art Evaluation of Repository Sealing Materials and Techniques," *Scientific Basis for Nuclear Waste Management XI, Materials Research Society Symposium Proceedings.* M. J. Apted and R. E. Westerman, Editors, pp. 219-231. (NNA.920819.0011)

Grice, A. G. (1989). "Fill Research at Mount Isa Mines, Ltd.," *Proceedings of the 4th International Symposium on Mining and Backfill,* Montreal, Canada. A. A. Balkema, Rotterdam, pp. 15-27. (NNA.940505.0032)

Grim, R. E. (1962). *Applied Clay Mineralogy.* McGraw-Hill Book Company, Inc., New York, NY, 422 pp. (MOL.19940718.0015)

Hertel, E. A., and E. E. Ryder (1991). "Areal Power Density: A Preliminary Examination of Underground Heat Transfer in a Potential Yucca Mountain Repository and Recommendations for Thermal Design Approaches," *SAND89-1989.* Sandia National Laboratories, Albuquerque, NM. (NNA.911106.0013)

Hinkebein, T., and M. A. Gardiner (1991). "Estimating Geochemical Behavior of Concretes to be Placed at Yucca Mountain," *Proceedings, Second International Conference on High Level Radioactive Waste Management,* Las Vegas, NV, pp. 1404-1409. (NNA.910724.0061)

Hough, B. K. (1969). *Basic Soils Engineering.* Ronald Press Company, New York, NY. 75 pp. (NNA.940629.0006)

Houlsby, A. C. (1990). *Construction and Design of Cement Grouting:* A Guide to Grouting in Rock Foundations. John Wiley & Sons, New York, NY, 442 pp. (NNA.921021.0065)

Hunt, W. G. (1989). "Laboratory Testing on High Density Tailings Backfill Mixes," *Proceedings of the 4th International Symposium on Mining and Backfill,* Montreal, Canada. A. A. Balkema, Rotterdam. (NNA.940505.0044)

Jeppe, C. B. (1946). "Gold Mining on the Witwatersrand," *Transvaal,* Chamber of Mines, Johannesburg, South Africa, 1821 pp. (MOL.19940714.0113)

Kapolyi, L., A. Schneider, and Z. S. Kesseru (1979). "Forecasting the Hazard of Water Inrush and New Possibilities of Fighting Against Them in Constructing Tunnels," *Tunneling Under Difficult Conditions, Proceedings, International Tunnel Symposium,* Tokyo, 1978, pp. 125-135. (NNA.940505.0045)

Karol, R. H. (1983). *Chemical Grouting.* Marcel Dekker Inc., New York, NY, 327 pp. (MOL.19940718.0016)

Kipko, Eh. Ya., Yu. A. Polozov, O. Yu. Lushinkova, V. A. Lagunov, and Yu. I. Suirskiy (1993). *Integrated Grouting and Hydrogeology of Fractured Rocks in the Former USSR.* Society of Mining, Metallurgy, and Exploration, Inc., Littleton, CO. (NNA.940629.0021)

Krumhansl, J. L. (1986). "Observations Regarding the Stability of Bentonite Backfill in a High-Level Waste (HLW) Repository in Rock Salt," *SAND83-1293.* Sandia National Laboratories, Albuquerque, NM. (NNA.940505.0036)

Legget, R. F., and P. F. Karrow (1982). *Handbook of Geology in Civil Engineering.* McGraw Hill Company, New York, NY, pp. 16.1-16.22. (NNA.940505.0037)

Leps, T. M. (1970). "Review of Shearing Strength of Rockhill," *Journal of the Soil Mechanics and Foundation Division,* Vol. 96, No. SM4, pp. 1159-1170. (NNA.921201.0142)

Licastro, P. H., J. A. Fernandez, and D. M. Roy (1990). "Preliminary Laboratory Testing of Cementitious Material for the Yucca Mountain Project Repository Sealing Program," *SAND86-0558.* Sandia National Laboratories, Albuquerque, NM. (NNA.900413.0037)

Loy, L. D. (1974). "Description of New, Innovative and Theoretical Mine Drainage Abatement Techniques," *Fifth Symposium on Coal Mine Drainage Research, Coal and the Environment Technical Conference,* October 22-24, Kentucky Fair and Exposition Center, Louisville, KY. National Coal Association. (NNA.940505.0049)

Lutzens, W. (1992). "Foam Concrete Seals for Abandoned Mine Shafts and Adits." Abandoned Mine Lands Program TN #8. U.S. Department of the Interior, Bureau of Mines, Washington, D.C. (NNA.940505.0050)

Marachi, N. D., C. K. Chan, and H. B. Seed (1972). "Evaluation of Properties of Rockfill Materials," *Journal of the Soil Mechanics and Foundations Division, Proceedings of the American Society of Civil Engineers.* SMl, pp. 95-114. (NNA.921201.0143)

Marsal, R. J. (1973). "Mechanical Properties of Rockfill," *Embankment-Dam Engineering,* John Wiley and Sons, New York, NY, pp. 109-200. (NNA.921021.0090)

Masullo, A. L. (1990). "Above-Ground Demonstration of Pneumatic Stowing Technologies." Prepared for U.S. Department of the Interior, Bureau of Mines, Washington, D.C. Contract No. J0388012. (MOL.19940805.0093)

McKinstry, J. D., and P. M. Laukkanen (1989). "Fill Operating Practices at Isa Mine—1983-1988," *Proceedings of the 4th International Symposium on Mining and Backfill,* Montreal, Canada. A. A. Balkema, Rotterdam, pp. 361-368. (NNA.940629.0009)

Mitchell, D. W. (1971). "Explosion-Proof Bulkheads, Present Practices," *RI-75812.* U.S. Department of the Interior, Bureau of Mines, Washington, D.C. (NNA.940505.0051)

Moreno, O., W. Hustrulid, and A. Harak (1981). "Support Capabilities of Rock Fill An Experimental Study, Application of Rock Mechanics to Cut-and-Fill Mining," *Proceedings, Conference on the Application of Rock Mechanics to Cut-and-Fill Mining,* Lulea, Sweden. The Institute of Mining and Metallurgy, London, England. (NNA.940505.0052)

Nash, W. R. (1984). "Underground Construction Application of Grout." *SME/American Institute of Mining Engineers Preprint No. 84375* for Presentation at the SME-American Institute of Mining Engineers Fall Meeting, Denver, CO. October 24-16, 1984. (NNA.940505.0053)

National Coal Board (1982). *The Treatment of Disused Mine Shafts and Adits.* National Coal Board, London, England, 88 pp. (NNA.940509.0002)

Neindorf, L. B. (1983). "Filling Operating Practice at Mount Isa Mine," *Proceedings of the International Symposium on Mining with Backfill,* Lulea, Sweden. A. A. Balkema, Rotterdam. (NNA.940505.0054)

Nuclear Waste Policy Act of 1982 (NWPA) (1983). Public Law 97-425, 96 Stat 2201, 42 USC 10101-10126, Washington, D.C. (NNA.870813.0017)

Nuclear Waste Technical Review Board (NWTRB) (1992). "Fifth Report to the U.S. Congress and the U.S. Secretary of Energy." U.S. Government Printing Office, June. (NNA.920624.0070)

Onofrei, M., M. Giay, R. Pusch, L. Börgesson, O. Karnland, B. Shenton, and B. Walker (1992). "Sealing Properties of Cement-Based Grout Materials," *STRIPA Project Report No. 92-28.* SKB, Stockholm, Sweden. (MOL.19940718.0018)

Peele, R. (1941). *Mining Engineers Handbook.* Third Edition, John Wiley & Sons, Inc., New York, NY, pp. 13.04-13.10. (NNA.940505.0055)

Perry, E. (1987). "Laboratory Tests on Granular Filter for Embankment Dams," *TR GL-87-22.* U.S. Department of the Army, Waterways Experiment Station, Corps of Engineers, Vicksburg, MS. (NNA.920902.0003)

Pothas, W. J. C. (1988). Hydraulic Backfilling as Developed and Practiced at East Driefontein," *Backfill in South African Mines.* South African Institute of Mining and Metallurgy, Johannesburg, South Africa. (NNA.940505.0056)

Ran, C., and J. Daemen (1991). "Effectiveness of Fracture Sealing with Bentonite Grouting," *NUREG/CR-5686.* Division of Engineering, Office of Nuclear Regulatory Research. U.S. Nuclear Regulatory Commission, Washington, D.C. (NNA.921112.0080)

Reuss, R., and F. J. Olivier (1992). "Mina Maria: A New Mine in Mexico's Old Cananea Mining District," *Preprint No. 92-175.* Society of Mining Engineers of the American Institute of Mining Engineers, Littleton, CO. (MOL.19940718.0023)

Rohlfing, L. (1983). "Slinger Belt Stowing Technique for Cemented Backfill at the Meggan, Mine," *Proceedings of the International Symposium on Mining with Backfill,* Lulea, Sweden. A. A. Balkema, Rotterdam, pp. 189-198. (NNA.940505.0057)

Sandia National Laboratories (SNL) (1987). "Nevada Nuclear Waste Storage Investigations Project, Site Characterization Project," Compiled by H. R. MacDougall, L. W. Scully, and J. R. Tillerson, *SAND84-2641.* Sandia National Laboratories, Albuquerque, NM. (NN1.880902.0014-.0019)

Scheetz, B. E., and D. M. Roy (1989). "Preliminary Survey of the Stability of Silica-Rich Cementitious Mortars 82-22 and 84-12 with Tuff," *LA-11222-MS.* Los Alamos National Laboratory, Los Alamos, NM. (NNA.891023.0022)

Sherard, J. L., L. P. Dunnigan, and T. R. Talbot (1984a). "Basic Properties of Sand and Gravel Filters," *American Society of Civil Engineers Journal of Geotechnical Engineering,* Vol. 110, No. 6, pp. 684-700. (NNA.920819.0004)

Sherard, J. L., L. P. Dunnigan, and J. R. Talbot (1984b). "Filter for Silts and Clays," *American Society of Civil Engineers Journal of Geotechnical Engineering,* Vol. 110, No. 6, pp. 701-718. (NNA.920819.0005)

Sitz, P., and W. Forster (1989). "Design of Sealing System for Shafts, Boreholes, and Access Tunnels Basis for Gas Storage in Abandoned Mines and Rock Caverns," *Storage of Gases in Rock Caverns.* A. A. Balkema, Rotterdam, pp. 193-200. (NNA.940505.0058)

Sitz, P., V. Koeckritz, and T. Oellers (1989). "Shaft Sealing for Nuclear Waste Repositories," *Shaft Engineering.* Institution of Mining and Metallurgy, London, England, June, pp. 345-352. (NNA.940505.0059)

Smith, M. J. (1980). "Engineered Barrier Development for a Nuclear Waste Repository in Basalt: An Integration of Current Knowledge," *RHO-BWI-ST-7.* Report to Rockwell International, Rockwell Hanford Operations, Energy Systems Group, Richland, WA. (HQZ.870227.2749)

Stephan, C. R. (1990a). "Construction of Seals in Underground Coal Mines," *Report No. 06-213-90.* Industrial Safety Division, Bruceton Safety Technology Center, Mine Safety and Health Administration, Pittsburgh, PA, 35 pp. (NNA.940505.0060)

Stephan, C. R. (1990b). "OMEGA 384 Block as a Seal Construction Material," *Report No. 10-318-90.* Industrial Safety Division, Bruceton Safety Technology Center, Mine Safety and Health Administration, Pittsburgh, PA, 15 pp. (NNA.940505.0061)

Stout, K., and L. Friel (1982). "Stopewall Closure and High-Modulus Backfill," *Transactions of the American Institute of Mining, Metallurgical, and Petroleum Engineers.* Society of Mining Engineers of the American Institute of Mining Engineers, Littleton, CO, Vol. 272. (NNA.940505.0062)

Thibodeau, D. (1989). "In Situ Determination of High Density Alluvial Sand Fill," *Proceedings of the 4th International Symposium on Mining and Backfill,* Montreal, Canada. A. A. Balkema, Rotterdam. (NNA.940505.0063)

Thomas, E. G. (1979). "Fill Technology in Underground Metalliferous Mines." Queen's University of Kingston, Department of Mining Engineering, International Academic Services, Ltd., Ontario, Canada. (MOL.19941121.0002)

Thomas, E. G. (1978). "Fill Permeability and Its Significance in Mine Fill Practice," *Mining with Backfill, 12th Canadian Rock Mechanics Symposium.* CIM Special Volume 19, Canadian Institute of Mining and Metallurgy, Montreal, Canada. (NNA.940505.0064)

Thomas, E. G., and R. Cowling (1978). "Pozzolanic Behavior of Ground Isa Mine Slag in Cemented Hydraulic Mine Fill at High Slag/Cement Ratios," *Mining with Backfill, 12th Canadian Rock Mechanics Symposium.* CIM Special Volume 19, Canadian Institute of Mining and Metallurgy, Montreal, Canada. (NNA.940505.0065)

Triebsch, G. F., and M. J. Sapko (1990). "Lake Lynn Laboratory: A State-of-the-Art Mining Research Laboratory," *Proceedings, International Symposium on Unique Underground Structures,* Denver, CO, June 12-15, Chapter 75, pp. 75-1–75-21. (NNA.940505.0066)

U.S. Department of Energy (DOE) (1988). "Site Characterization Plan: Yucca Mountain Site, Nevada Research and Development Area, Nevada," *DOE/RW-0199*, Office of Scientific and Technical Information, Oak Ridge, Tennessee. (HQO.881201.0002)

U.S. Nuclear Regulatory Commission (NRC) (1986). "Disposal of High-Level Radioactive Wastes in Geologic Repositories," Title 10, Code of Federal Regulations, Part 60 (10 CFR Part 60), U.S. Government Printing Office, Washington, D.C. (NNA.891018.0190)

Voss, K. H. (1983). "The State of Development of Pneumatic Stowing in the Hard Coal Industry of the Federal Republic of Germany," *Proceedings of the International Symposium on Mining with Backfill,* Lulea, Sweden. A. A. Balkema, Rotterdam. (MOL.19941121.0001)

Young, W. C. (1989). *Roark's Formulas for Stress and Strain.* McGraw-Hill, New York, NY, 763 pp. (NNA.900614.0538)

Yu, T. R. (1989). "Some Factors Relating to the Stability of Consolidated Rockfill at Kidd, Creek," *Proceedings of the 4th International Symposium on Mining and Backfill,* Montreal, Canada. A. A. Balkema, Rotterdam, pp. 279-286. (NNA.940505.0067)

APPENDIX A.1

Cannon Mine

Site Visit: November 8, 1989

Purpose: To show the effectiveness of a simple mechanical backfilling system applied to a complex underground mining system with specific performance requirements.

Table of Contents

A.1.1 Introduction A.1-1

A.1.1.1 Description A.1-1
A.1.1.2 Objective of Sealing/Backfilling Operation A.1-1
A.1.1.3 Purpose A.1-4

A.1.2 Materials A.1-4

A.1.2.1 Description A.1-4
A.1.2.2 Properties A.1-4

A.1.3 Equipment A.1-5

A.1.4 Seal/Backfill Placement Operations A.1-12

A.1.4.1 Sequence of Operations A.1-12
A.1.4.2 Conditions of Emplacement A.1-12
A.1.4.3 Problems Encountered A.1-13
A.1.4.4 Quality Control A.1-14

A.1.5 Discussion A.1-14

A.1.6 References A.1-15

APPENDIX A.1
Cannon Mine

A.1.1 Introduction

A.1.1.1 Description

The Cannon Mine is a joint venture between Asamera Minerals and Breakwater Resources, and is located in central Washington state (Figure A.1-1). Although the original mineral find was in 1885, the latest round of development began in 1982. The first ore was milled in the Cannon Mine's 2000-TPD flotation plant in 1985. The ore averages approximately 0.25 oz of gold per ton and is mined by a sublevel bench-and-fill method (Figure A.1-2). Extraction follows a primary/secondary stoping sequence, with parallel 24-ft-wide stopes mined out on the first pass, leaving a 24-ft pillar. The stopes are mined with 24-ft-wide by 15-ft-high sill cuts from access drifts located within ore on each 50-ft sublevel. Drop raises are driven between sublevels at the end of each stope block and widened into a slot. The bench is then blasted into the slot and mucked from the bottom of the sublevel. Cemented rock fill is trucked to the upper access of the sublevel and dumped in the empty stope (Figure A.1-3), providing ground support while the intervening pillars (secondary stopes) are mined. The secondary stopes are filled with either uncemented fill or cemented fill to provide confinement to the backfill pillars and to minimize the amount of open void at completion of the mining process.

A.1.1.2 Objective of Sealing/Backfilling Operation

Underground mining with cemented backfill was selected because it enabled nearly 100% ore recovery without surface subsidence. Subsidence was not acceptable at this property because the mining company did not control the surface rights and because of the proximity to the city of Wenatchee, Washington. Maximum resource recovery was desired because of the relatively high gold content of the ore.

To achieve the high ore recovery, the backfill properties were designed to deliver specific performance in three areas:

- High compressive strength to support the full overburden load and to provide stable walls during pillar mining,

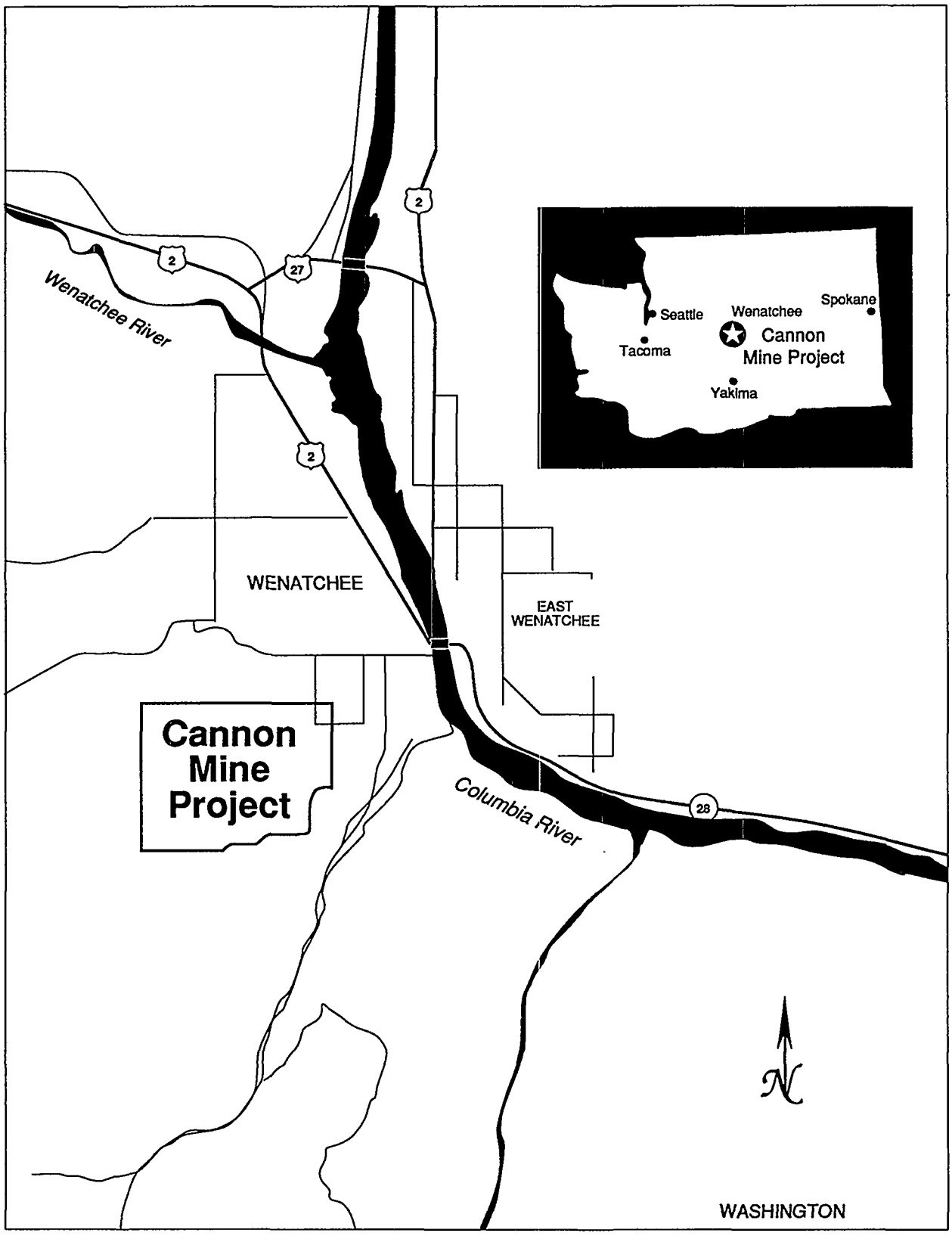


Figure A.1-1
Location of the Cannon Project Showing Proximity of Mine to City of Wenatchee

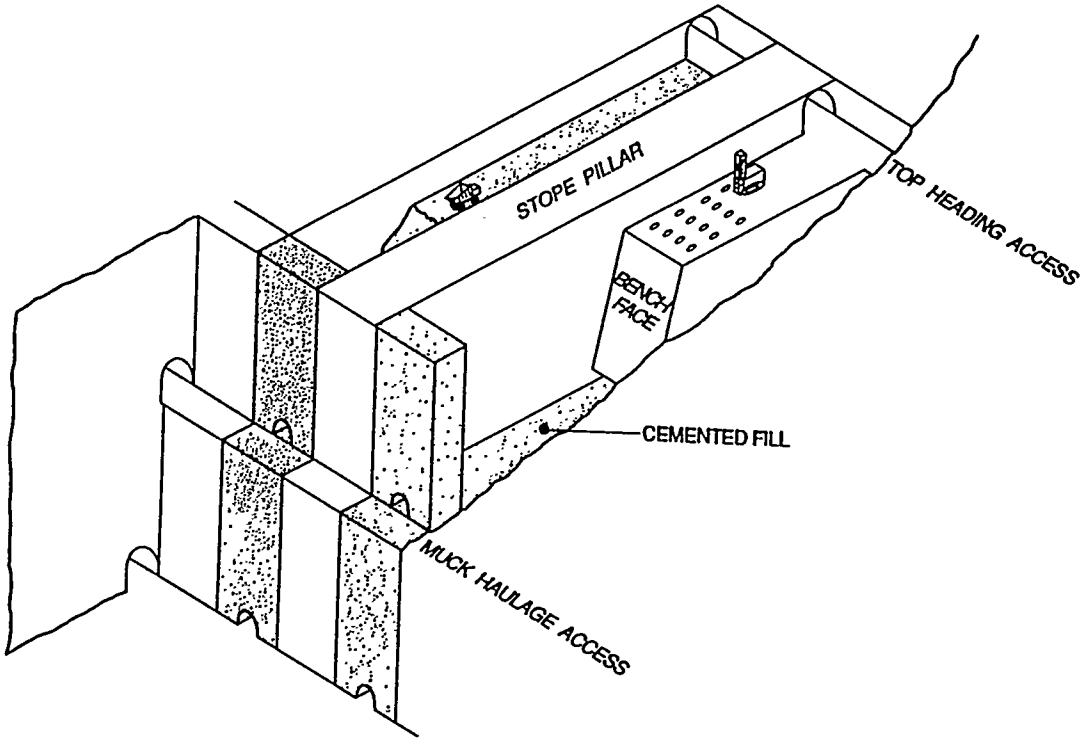


Figure A.1-2
Sublevel Bench-and-Fill Mining Method Used at the Cannon Mine
(after Brechtel et al., 1989)

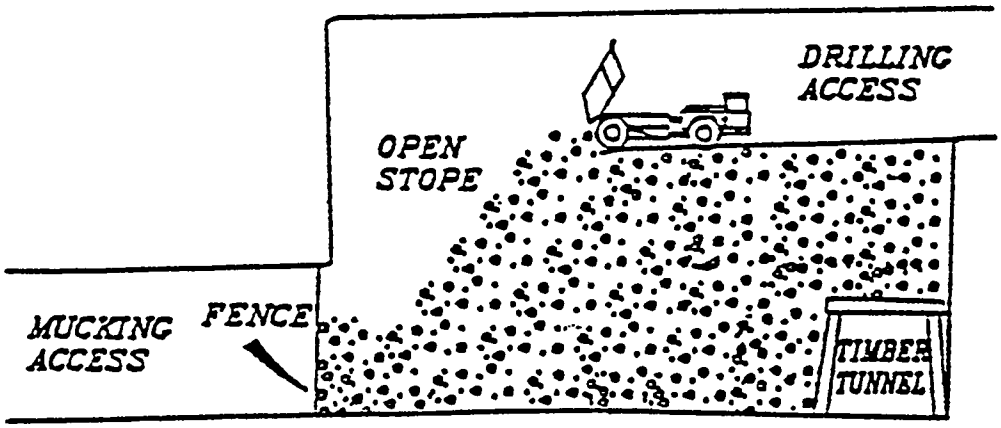


Figure A.1-3
Truck Placement of Backfill at the Cannon Mine (after Baz-Dresch, 1987)

- High strength of fill/rock interfaces to support fully undercut rock pillars hanging between backfill during multilift recovery of the pillars,

- Low slump/water content to support the weight of trucks during backfill placement and to allow tight packing of the backfill at the hanging wall.

A design study, described by Brechtel et al., 1989, was undertaken to evaluate backfill aggregate sources and materials, cement contents, pozzolan additives, and water content. A mixture design was identified that satisfied the performance requirements and minimized the cement content and cost. An extensive tradeoff study was also performed to evaluate various mixing plant and delivery system options.

A.1.1.3 Purpose

The Cannon Mine case history documents the successful use of available and relatively simple mechanical equipment to perform a geometrically complex backfilling job that delivered specific and rigorous performance requirements. Performance of the backfill was verified by monitoring instrumentation and demonstrated by successful recovery of the gold ore. The technology is available and suitable for backfilling operations at the potential Yucca Mountain repository.

A.1.2 Materials

A.1.2.1 Description

Two basic types of materials are used for backfilling: (1) a cemented fill consisting of approximately 55% coarse aggregate, 40% sand, and 5% portland cement; and (2) an uncemented fill consisting of the coarse aggregate component of the cemented fill or mine waste and development rock. The cemented backfill is an integral part of the extraction scheme at the Cannon Mine, making the high rate of pillar recovery possible. Considerable effort was put into designing the backfill mix to deliver the required performance at minimum cost.

A.1.2.2 Properties

The required strength of the backfill was determined by overburden support considerations to be approximately 1200 pounds per square inch (psi). To meet this strength requirement at minimum cost, relatively high-quality coarse and fine materials were used to minimize the cement content. Clean sand was available on the mine property, and "pit-run" material was

used to minimize handling costs. Mined rock was experimented with for coarse aggregate, but the bulk of the coarse aggregate is river gravel purchased commercially from an off-site aggregate pit. The coarse aggregate was screened to remove oversized material and fines.

The mix design was established by testing different coarse/fine aggregate ratios, cement contents, and water/cement ratios. The strength was found to be very sensitive to both cement and coarse aggregate content. A backfill mixture with 5% cement, 55% coarse aggregate, and 40% fine aggregate produced the required mixture strength. Table A.1-1 shows the results of a typical backfill size analysis reported in the literature, while Figure A.1-4 illustrates the effect of the proportion of coarse aggregate to fine aggregate on backfill strength.

A.1.3 Equipment

The backfill system design was developed based on a tradeoff study that addressed aspects of the backfilling process:

- mixing of the backfill,
- transport of the backfill,
- placement of the backfill in the stopes, and
- tightfilling of the hanging wall lift.

Conceptual designs for 11 different backfill systems were developed and evaluated to identify the best approach. The options were described by Baz-Dresch (1987), and consisted of

1. Pumped Backfill, Variation 1. This system would consist of a batch plant and materials storage facilities located on the surface. Backfill would be fed to the underground mine through a borehole. An underground concrete pump would place the material through a pipeline system.

2. Pumped Backfill, Variation 2. This is the same as System 1, with a different location of surface components.

Table A.1-1
Typical Backfill Size Analysis (after Brechtel et al., 1989)

Grain Size	Percent Passing	
Units	Coarse Aggregate	Fine Aggregate
75	100	100
50	90	100
35	70	100
25	40	100
19	20	100
13	10	100
9.5	4	90
4.8	1	87
2	0	74
1	0	48
0.5	0	20
0.1	0	1
0.74	0	1

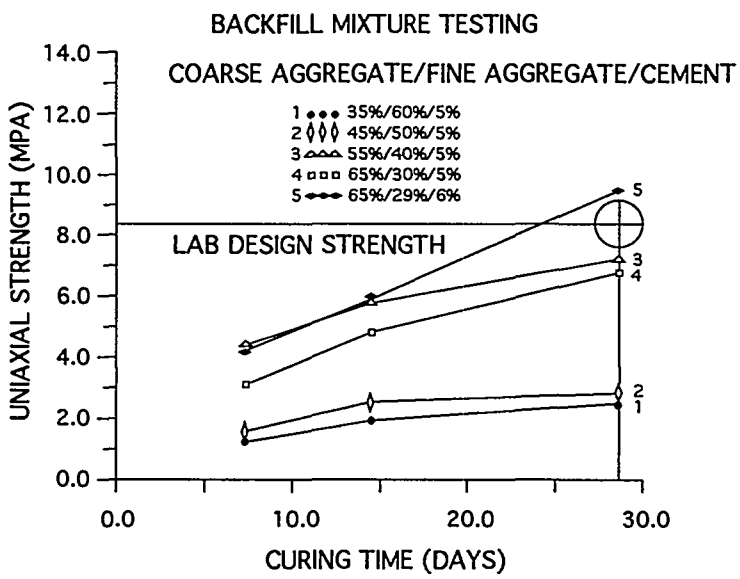


Figure A.1-4
Effect of Coarse Aggregate Content on Backfill Strength (after Brechtel et al., 1989)

3. Pneumatic Placement, Blowers Located on Surface. Materials storage, mixing equipment, and rotary air lock would be located on the surface. Backfill would be pneumatically placed underground via a drill hole and pipeline system.

4. Pneumatic Placement, Two Separate Blow Systems on the Surface. This system would consist of parallel System 3s. Fill could be placed in two locations simultaneously.

5. Truck Haulage, Pneumatic Placement. Sand and aggregate would be blended in surface facilities, transferred underground through a borehole, and trucked to pneumatic stowing equipment located at the stope. In the case of cemented backfill, cement and water would be added after the material had exited the rotary air lock.

6. Truck Placement, Pneumatic Tightfill. Backfill would be mixed in surface facilities, conveyed underground through a borehole, and loaded into trucks for delivery to the stopes. The fill would be dumped by the trucks into the stope void. When tightfilling against the hanging wall was required, the backfill would be pneumatically placed.

7. Truck-Mounted Slinger Placement. Backfill, mixed in surface facilities and conveyed underground through a borehole, would be loaded into trucks equipped with built-in slingers. All backfill would be placed in the stopes through these slinger trucks.

8. Truck Placement, Slinger Tightfill. This system is the same as truck placement with pneumatic tightfill (System 6) except that a slinger instead of a pneumatic blower would be used when tightfilling against the hanging wall.

9. Truck Placement, Pneumatic Tightfill North End; Borehole Placement South End. This system is identical to System 6 except that the stopes at the south end of the orebody (the surface rights in this area are controlled by the company) would be backfilled from the surface through a series of boreholes into each of the stopes.

10. Truck Placement, Slinger Tightfill North End; Borehole Placement South End. This system is identical to System 9 except that slingers instead of pneumatic placement would be used to place the backfill tight against the hanging wall.

11. Truck Placement, Slinger Tightfill, Materials Supplied from On-Site Source. This system is identical to System 8 with the exception that the materials would be supplied from on-site sources.

System 8 was initially selected based on economic and other factors. Relative advantages and disadvantages supporting the selection are listed in Table A.1-2. Slinger placement for tightfilling was abandoned very early, and all tightfilling was performed using specially modified load-haul-dump (LHD) units.

The equipment selected for backfilling at the Cannon Mine consisted of (1) a surface storage facility, (2) an underground mixing facility (pug mill), (3) an underground bin and loading facility, and (4) rubber-tired equipment for fill transport and placement. Figure A.1-5 is a schematic of the backfill plant. The surface facility consisted of a 350-T cement silo with a vibrating feeder, a sand and aggregate dump, and a delivery system for a water-reducing admixture. Backfill components were transferred underground by gravity through boreholes to underground silos. Sand and aggregate were metered from underground silos using vibrating feeders. Cement and water-reducing admixture were fed directly underground from the surface and metered using volumetric vane feeders. Figure A.1-6 shows sand and aggregate vibratory feeders discharging into pug mill.

The backfill constituents were mixed underground in a double-screw pug mill, and the mixed fill was discharged from the mill into a 17-yd^3 bin located above the backfill truck drift.

A batch of fill was stored in the bin until loading and was then dropped through a pair of hydraulically powered clamshell gates into the backfill truck. A diverter gate enabled trucks to load uncemented aggregate only during secondary stope filling.

Selection of trucks was an important part of the design process and was based on dump height, load capacity, maintenance, operating costs, capital costs, number of units required, accessibility of parts and service, production capacity, and equipment delivery time. The most important constraint was dumping height, which was limited to 15 ft; a 30-T,

Table A.1-2

Advantages and Disadvantages of Truck Placement with Slinger Tightfill

Advantages	Disadvantages
Provides great flexibility in reallocating backfill quickly.	Technology for placement cemented backfill with slinger unproven for this material.
Allows placement of course (minus 4-in.) material.	Introduces traffic problems of having multiple trucks on one shift.
Can backfill several stopes at the same time.	Introduces additional ventilation requirements.
Provides observable contact of material with the back.	Potential problems with delivery of concrete through a slinger during the winter due to freezing and material buildup.
Capital and operating costs are lower than for System 1 for a 2800-TPD mining rate.	The reliability of the slinger is an unknown factor.
Does not require major underground pipe maintenance and replacement.	
Allows interchangeability of backfill and production trucks to level peak production or backfill requirements.	

telescoping dump truck was finally selected. Figure A.1-7 shows the dumping procedure for low back heights.

Tightfilling was achieved by jamming the fill against the back with a 4-ft^2 plate mounted on a pair of H-beams bolted to the bucket of an LHD. Cemented fill is placed at the backfill face using either a truck or LHD; the LHD equipped with the push plate then packs the fill to the hanging wall. Although truck-mounted slingers were part of the selected backfill placement system, the mine was unable to devise a simple and economical slinger system that would adapt to their trucks. A bulldozer was also tried, but was unsuccessful.

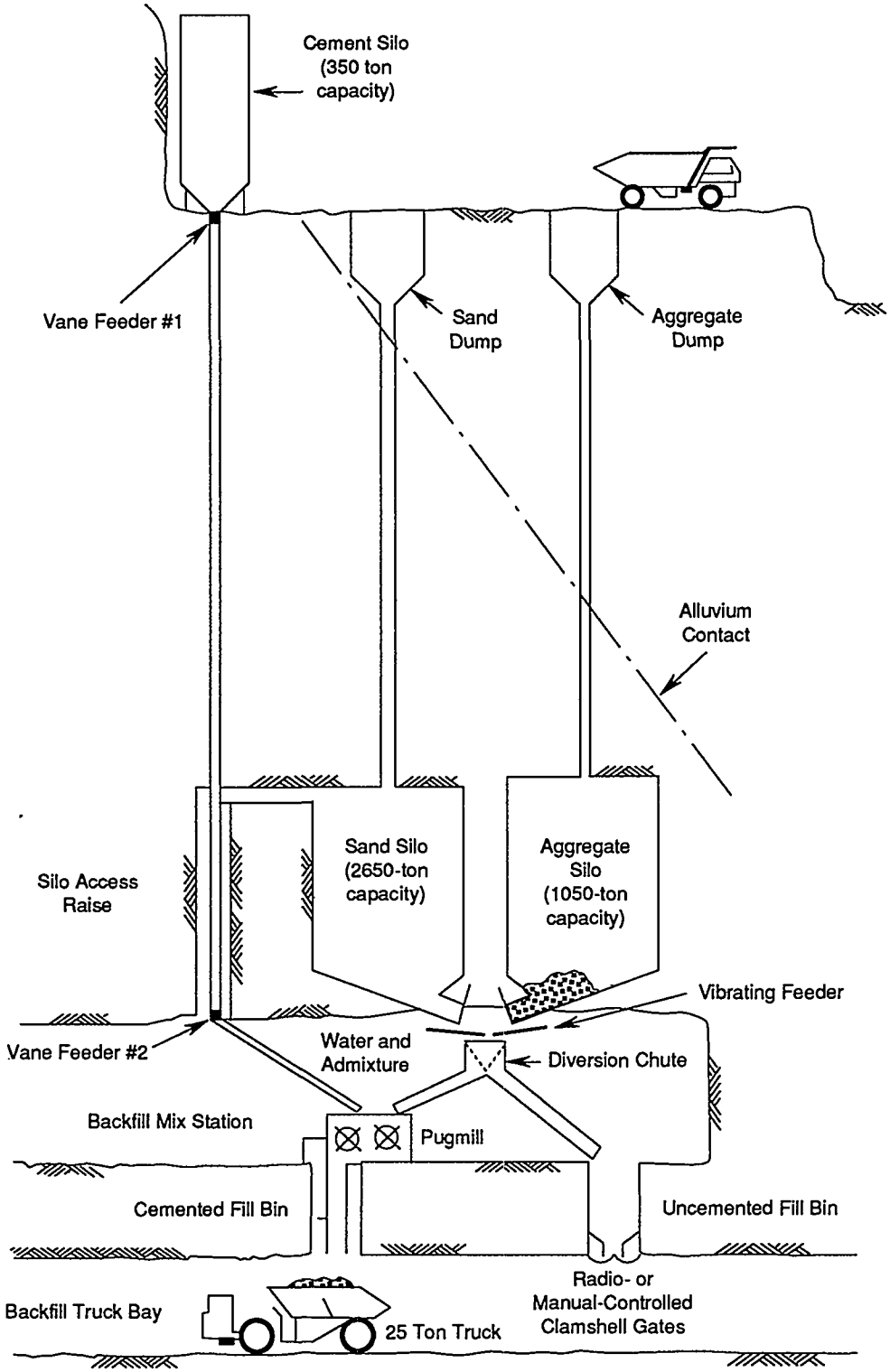


Figure A.1-5
Schematic of Underground Backfill Plant (after Baz-Dresch, 1987)

Figure A.1-6
Sand and Aggregate Vibratory Feeders Discharging Into Pug Mill

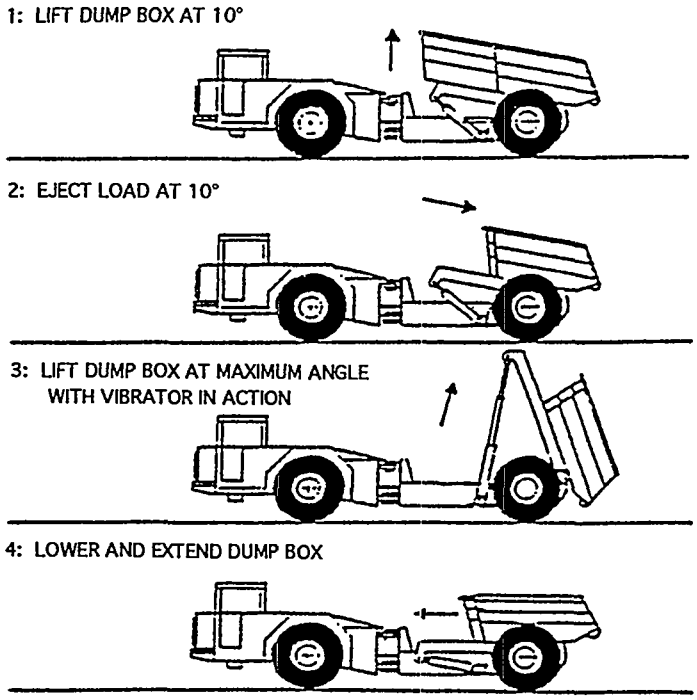


Figure A.1-7
Dumping Procedure for Low Back Heights

A.1.4 Seal/Backfill Placement Operations

A.1.4.1 Sequence of Operations

The sequence of backfilling operations is as follows:

- Backfill components are mixed underground in the underground pug mill.
- Fill is loaded into trucks from the fill bin using manual or radio control.
- Fill is hauled to the top of the stope block and dumped over the edge of the bench into the stope. When the backfill pile reaches the bench level, trucks back out on the fill to dump (Figure A.1-3).
- The modified LHD is used to tightfill to the back in the hanging wall lift.

A.1.4.2 Conditions of Emplacement

The backfill is either a low slump-cemented fill or dry, uncemented fill that allows loaded trucks to travel over it immediately after placement. Headroom is limited during all

placement activities to roughly 15 ft, requiring the use of tele-dump trucks to allow relatively large loads to be tipped. The tightfilling operation has similar headroom and is restricted by the width of the room. The use of articulated LHDs with the small push plate attachment allows maximum range of motion in the restricted area.

Backfill is transported to the multilevel stopes through ramps and accesses used by similar sized trucks and LHDs simultaneously involved in ore production. The backfilling system has operated at rates up to 5,000 tons per day in this relatively constricted environment. Tightfilling rates of up to 100 tons per hour have been achieved using the LHD with push plate.

The tightfilling operation produces an interface with the hanging wall that varies locally from point contact to roughly 4 to 5 in. of gap.

A.1.4.3 Problems Encountered

Several problems were reported during backfilling. The biggest problem was segregation of the coarse aggregate, which occurred as the fill was dumped over the 50-ft stope bench. Zones of uncemented coarse aggregate developed at the toe and along the perimeter of the stope, resulting in locally lower backfill strength. This problem was reduced by changing the maximum aggregate size to 2 in. to reduce separation. With this size of aggregate, considerable remixing can occur as the materials slide down the stope.

Voids occurred in some places, especially when tightfilling areas where the back was too high to jam or the geometry of the drift did not permit efficient jamming. Significant voids at the hanging wall were generally filled by pumping cement grout.

Cement falling down the polyethylene-lined boreholes from the surface storage bins generated static electricity, which caused problems with the electronic controller. This problem was solved by relocating the control cables.

Problems with control of the cement content occurred several times, resulting in large, low-strength zones of backfill. This problem apparently resulted from inconsistent operation of the vane feeders. The operators felt that quality control could be increased by using a weighed system for batching.

A.1.4.4 Quality Control

A test cylinder was taken every day from material in the loaded backfill trucks. An accelerated cure in a hot-water bath was performed on the cylinder, then it was tested. Quick testing enabled adjustments to be made to the backfill plant before too much substandard fill is placed.

The quality of in-place fill is periodically monitored by diamond coring, and some of the cores were tested. Visual inspections are continuously made during secondary mining for voids and segregation. Calibration on the batch metering devices was also periodically checked.

A.1.5 Discussion

The specific properties of the backfill used in the Cannon Mine will probably be different from that required at the Yucca Mountain repository. At Yucca Mountain, it will probably not be necessary to use a cemented rockfill, because there will be no mining between vertical fill walls and no need to support overburden because of the low extraction ratio. However, the mechanical emplacement method used at the Cannon Mine could readily be applied for backfilling the repository. The lower half of the repository drifts could be filled with crushed tuff unloaded from either LHDs or tele-dump trucks. In a second pass, the upper half could be filled. Depending upon the specification on unfilled void at the roof, the crushed tuff may or may not require "jamming" against the back.

In conclusion, the Cannon Mine is successfully using readily available technology to complete extraction and backfilling of a geometrically complex underground orebody. There is no evidence of surface subsidence, in spite of instrumentation that verifies the overburden has settled onto the backfills. There is no evidence to suggest that similar state-of-the-art technology could not be used to backfill the repository at Yucca Mountain; if anything, it would be less complex than the Cannon Mine.

A.1.6 References

Brechtel, C. E., and M. P. Hardy, J. Baz-Dresch and J. S. Knowlson (1989). "Application of High-Strength Backfill at the Cannon Mine," *4th International Symposium on Mining with Backfill*, Montreal, Quebec. (NNA.920819.0010)

Baz-Dresch, J. (1987). "High Strength Backfilling at the Cannon Mine," *Proceedings, Third Western Regional Conference on Precious Metals, Coal and Environment*, Rapid City, South Dakota. (NNA.940509.0009)

APPENDIX A.2

Billie Mine

Site Visit: December 7, 1989

Purpose: To demonstrate the application of pneumatic stowing to a large-scale mining operation where fill transport is primarily vertical.

Table of Contents

A.2.1 Introduction A.2-1

A.2.1.1 Description A.2-1
A.2.1.2 Objective of Sealing/Backfilling Operation A.2-2
A.2.1.3 Purpose A.2-2

A.2.2 Materials A.2-3

A.2.2.1 Description A.2-3
A.2.2.2 Properties A.2-3

A.2.3 Equipment A.2-3

A.2.4 Seal/Backfill Placement Operations A.2-5

A.2.4.1 Sequence of Operations A.2-5
A.2.4.2 Conditions of Emplacement A.2-5
A.2.4.3 Problems Encountered A.2-5
A.2.4.4 Quality Control A.2-5

A.2.5 Discussion A.2-5

A.2.6 References A.2-6

APPENDIX A.2
Billie Mine

A.2.1 Introduction

A.2.1.1 Description

The Billie Mine is owned by National Borate Company, a wholly owned subsidiary of the Owens Corning Fiberglass Company. The mine is an underground borate mineral mine located 15 miles southeast of Furnace Creek, California (Figure A.2-1). The Hanna Mining Company has a management contract with National Borate, originally to develop and operate the mine, but currently to maintain it in a state of shutdown. The primary economic minerals are colemanite and probertite, which are used in the manufacture of fiberglass.

The mine was operated using a sublevel cut-and-fill method, employing continuous miners for development and drill-and-blast for production mining (Figure A.2-2). Stopes were cut along the strike of the orebody using roadheaders. The sills between the levels were

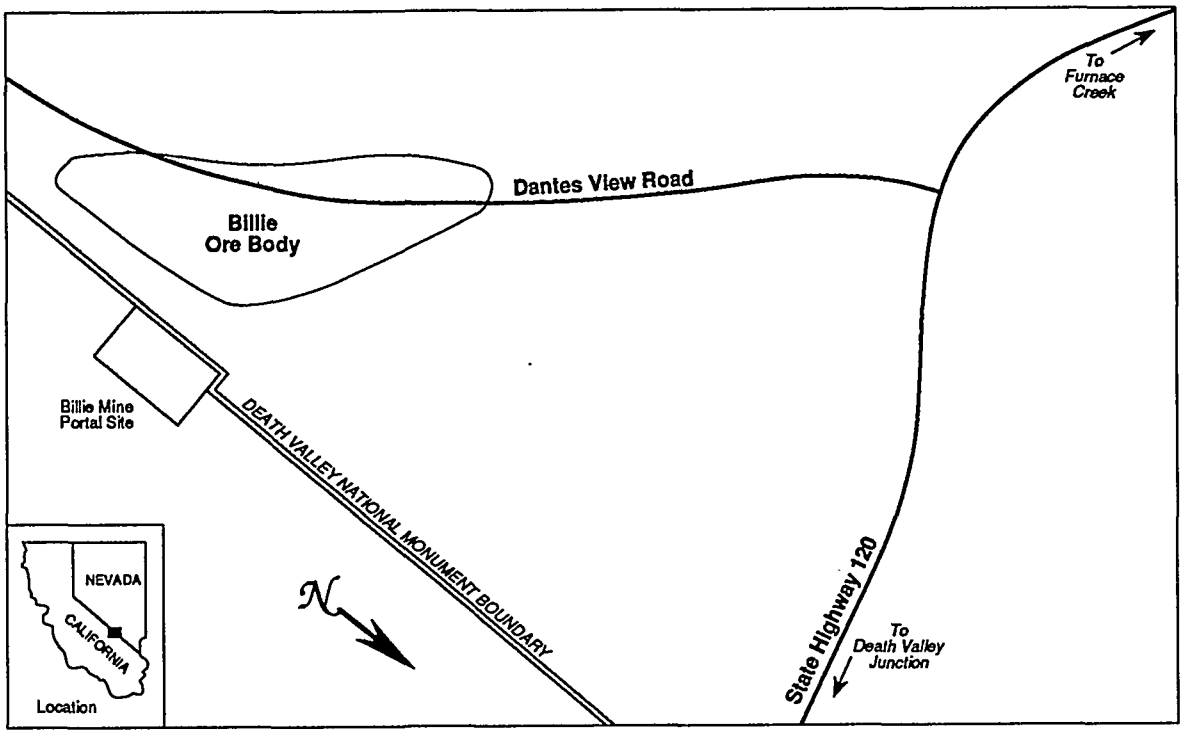


Figure A.2-1
Location of the Billie Mine (after Garrett, 1985)

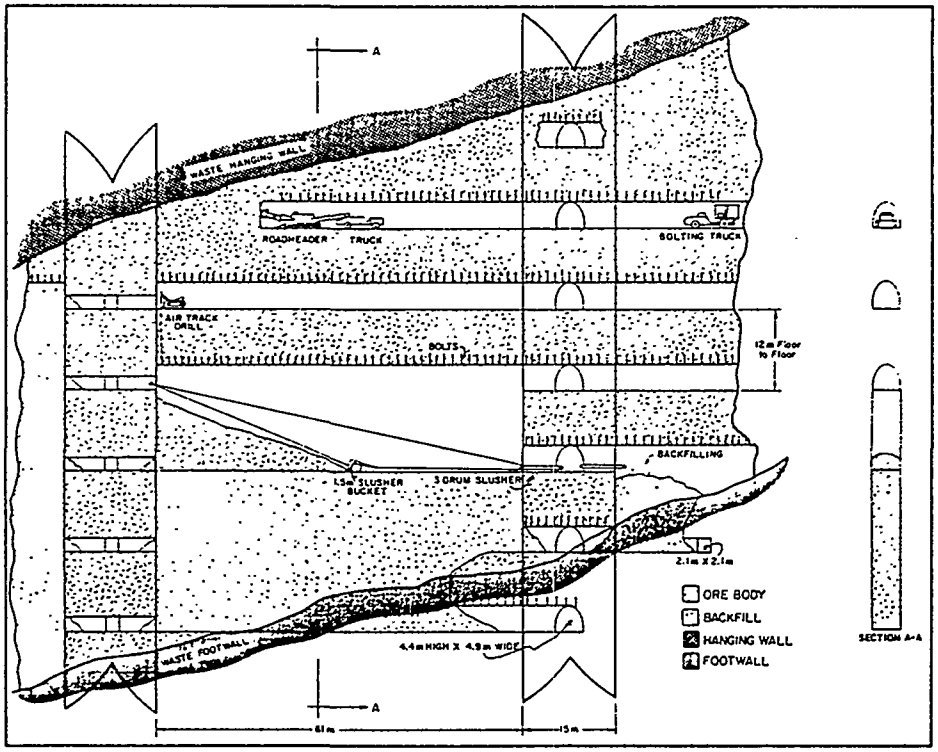


Figure A.2-2

Sublevel Cut-and-Fill Mining Method Used at the Billie Mine (after Garrett, 1985)

extracted by drilling vertical holes with an air-track drill and blasting the stope with ammonium nitrate and fuel oil (ANFO). The stope is mucked from below with a slusher, and transferred to an ore pass using trucks or load-haul-dump (LHD) units, and then hoisted to surface via a skip/cage in a vertical shaft.

A.2.1.2 Objective of Sealing/Backfilling Operation

Backfilling is required at the Billie Mine to prevent subsidence. The ground surface above the orebody is located in a national monument where mine subsidence is not permitted. In addition, stope filling with waste was favored because of the limited surface storage available for waste.

A.2.1.3 Purpose

The Billie Mine was primarily selected as a case history of the application of pneumatic stowing in a large-scale underground mining operation. Pneumatic stowing is a candidate for backfilling at Yucca Mountain because it uses little or no water for fill transport. It can be used to transport fill a limited distance horizontally, as well as vertically. Additionally,

pneumatic stowing can be used to tightfill. Mechanical backfilling of uncemented fill is also practiced at the Billie Mine.

A.2.2 Materials

A.2.2.1 Description

The pneumatic backfill system delivers screened waste from stripping dumps of previous open-pit mines from surface directly into the stopes. The fill is primarily alluvial gravel. The fill is screened to -1 in. Coarse rejects are used for reclamation.

Waste from mine development is used for mechanical backfilling. The ore deposit is essentially a limestone breccia permeated with borate minerals, so much of the waste rock is limestone.

A.2.2.2 Properties

No information on backfill properties was available from the site visit.

A.2.3 Equipment

Two backfill systems are used at the Billie Mine: mechanical and pneumatic. Mechanical stowing is performed with off-the-shelf trucks and LHDs (Figure A.2-3). Pneumatic backfilling is accomplished using a surface backfill plant and a distribution system of pipes leading underground to the stopes.

The backfill plant consists of a 23-ton bin, a blower, a rotary airlock feeder (RALF), and a hydraulic power supply. The blower is a 400-horsepower Gardner Denver positive displacement blower that generates line pressure. The RALF is the key to pneumatic stowing and was supplied by Hanna-Beric. The RALF is a drum that rotates in a close-fitting enclosure. Pockets on the drum transfer the feed from the inlet side to the pressurized air stream. Figure A.2-4 is a schematic of the system. The surface backfill plant is located in a below grade, concrete-lined enclosure. The plant was situated below grade to minimize the visual impact to the monument. An important part of any pneumatic stowing system is the pipe. Material is conveyed vertically down a 10-in. schedule 80 pipe for 900 ft and horizontally via 8-in.-hardened steel pipe up to 700 ft horizontally to the stope. A water ring is placed 50 ft from the pipe exit to add water for dust suppression and increased compaction.

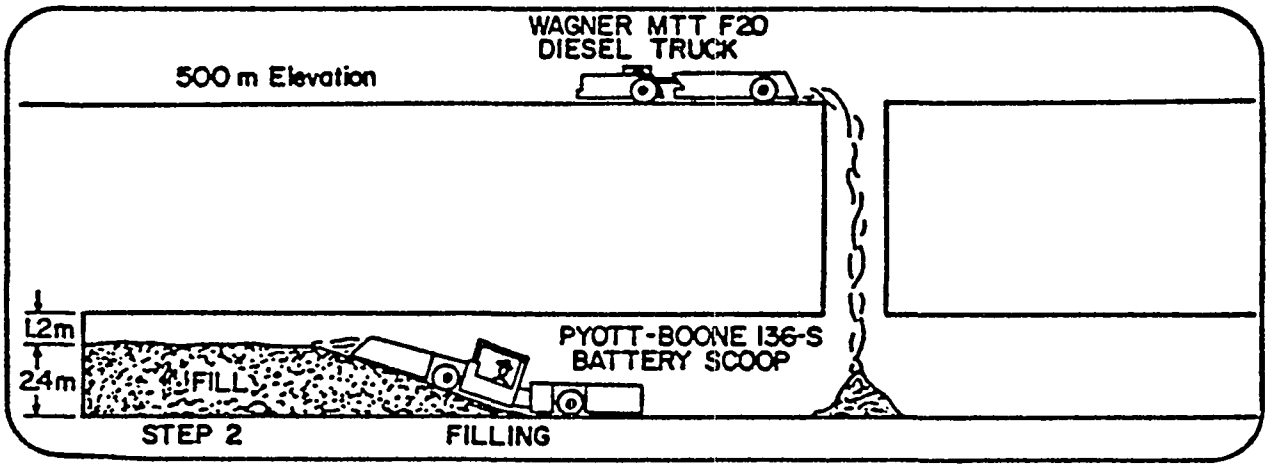


Figure A.2-3
Mechanical Backfilling Method (after Garrett, 1985)

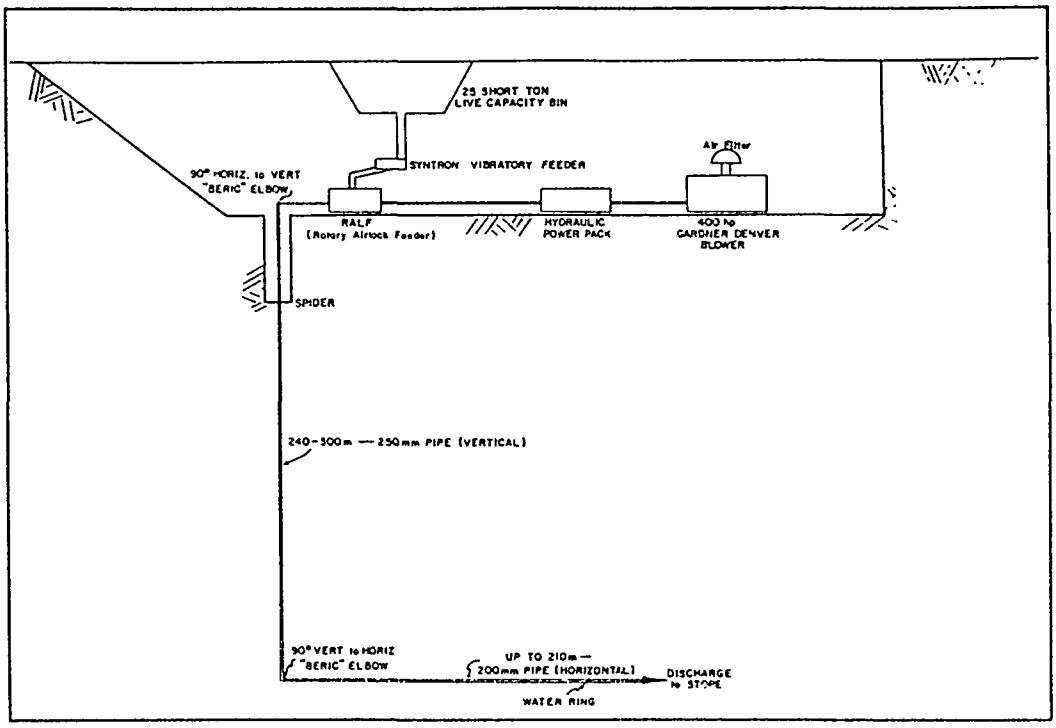


Figure A.2-4
Pneumatic Backfilling System (after Garrett, 1985)

A.2.4 Seal/Backfill Placement Operations

A.2.4.1 Sequence of Operations

For pneumatic backfilling, alluvial material is reclaimed, screened, and transported to the 23-ton bin surface backfill plant. The bin feeds the RALF, which transfers the fill to the pipeline. The fill is conveyed underground via the pipeline and blown into the stopes. Approximately 50 ft^3/sec of air at 15 pounds per square inch (psi) is used to transport the fill underground. Backfilling rates of 100 tons per hour (tph) were achieved.

A.2.4.2 Conditions of Emplacement

The stopes were generally dry, and conditions were good for backfilling.

A.2.4.3 Problems Encountered

Perhaps the most severe problem encountered with the pneumatic filling is pipe wear. The alluvial backfill used is highly abrasive. Pipe wear was measured in the 10-in. vertical backfill line at the 1700 level. Approximately 0.0054 in. of wear occurred per 10,000 tons of fill. This translates to a pipe life of approximately 20,000 tons of fill. The RALF must be rebuilt after 100,000 tons. Pipe wear has been minimized by maintaining very careful alignment of the fill pipe, and by using flanged pipe with special elbows and induction-hardened pipe immediately following the elbows.

A.2.4.4 Quality Control

Quality control of the fill was maintained by visual inspection.

A.2.5 Discussion

Both the mechanical and pneumatic stowing of backfill used at the Billie Mine could be directly applied to the Yucca Mountain repository. The mechanical emplacement method could be directly applied, with the crushed tuff filled and compacted in lifts. The pneumatic system could also be directly applied, using sized and screened crushed tuff instead of waste rock. One limitation of the pneumatic backfill system is a limited ability to transport fill horizontally (2,000 ft). This can be addressed by spacing the vertical fill holes at appropriate intervals. To minimize the problems of sealing these backfill holes during decommissioning of the Yucca Mountain repository, the holes could be positioned at existing exploration hole locations. Alternatively, fill could be mechanically transported underground to a mobile backfill plant for pneumatic stowing only of those areas that need to be tightfilled to the back.

Subsequent to this site visit, communications with mine operators indicated that they were not entirely satisfied with pneumatic stowing because of maintenance problems caused by wear, and they were considering other alternatives, primarily mechanical emplacement.

A.2.6 References

Garrett, Charles R. (1985). "Development of a Roadheader Mining System at American Borate Company," *Proceedings of the Rapid Excavation and Tunneling Conference*, pp. 886-901. (NNA.940505.0041)

J. S. Redpath Corp., Redpath's Pneumatic Conveying System, Sales Brochure.

APPENDIX A.3

Apex Mine

Site Visit: December 8, 1989

Purpose: To demonstrate successful application of pneumatic stowing at a small mine using cemented fill where horizontal transport distances were relatively limited.

Table of Contents

A.3.1 Introduction A.3-1

A.3.1.1 Description A.3-1
A.3.1.2 Objective of Sealing/Backfilling Operation A.3-4
A.3.1.3 Purpose A.3-4

A.3.2 Materials A.3-4

A.3.2.1 Description A.3-4
A.3.2.2 Properties A.3-4

A.3.3 Equipment A.3-4

A.3.4 Seal/Backfill Placement Operations A.3-5

A.3.4.1 Sequence of Operations A.3-5
A.3.4.2 Conditions of Emplacement A.3-5
A.3.4.3 Problems Encountered A.3-6

A.3.5 Discussion A.3-6

APPENDIX A.3
Apex Mine

A.3.1 Introduction

A.3.1.1 Description

The Apex Mine is an underground gallium/germanium mine located near St. George, Utah, which has a long history and was originally discovered in 1880. It was mined intermittently for high-grade copper oxides (malachite and azurite) through the 1950s. In the early 1980s, it was acquired by St. George Mining Company, a subsidiary of Musto Explorations, Vancouver, British Columbia, who worked it for several years. The Apex Mine has recently been acquired by Hecla. The Small Mine Development Company retrofitted the mine and brought it into production under contract with Hecla. Figure A.3-1 shows the location of the Apex Mine.

The primary economic minerals are gallium/germanium, which occur in a complex ore-bearing zone. The ore occurs within a pipe-like structure as a series of iron oxides containing both gallium and germanium, plus significant amounts of silver, zinc, lead, and copper. The zone of oxidation lies near a zone of shattered dolomites, which cause poor ground conditions.

The mine is being developed as a small (100 tons per day) underhand cut-and-fill mine, using conventional equipment for development. The ore zone is of such low strength that rooms wider than 4 ft are unstable and will collapse. The ore is 60 to 70% free digging with a load-haul-dump (LHD) bucket; very little blasting is required.

Figure A.3-2 shows a section of the mine workings. A 15% spiral decline provides access into the ore zone. Crosscuts are driven at 25-ft intervals from this ramp. Initially, rockbolts and wire mesh were employed for ground control, without success. Four-foot-wide openings can be mined and will remain stable if a very thin (0.1- to 0.2-in.) layer of shotcrete is immediately applied.

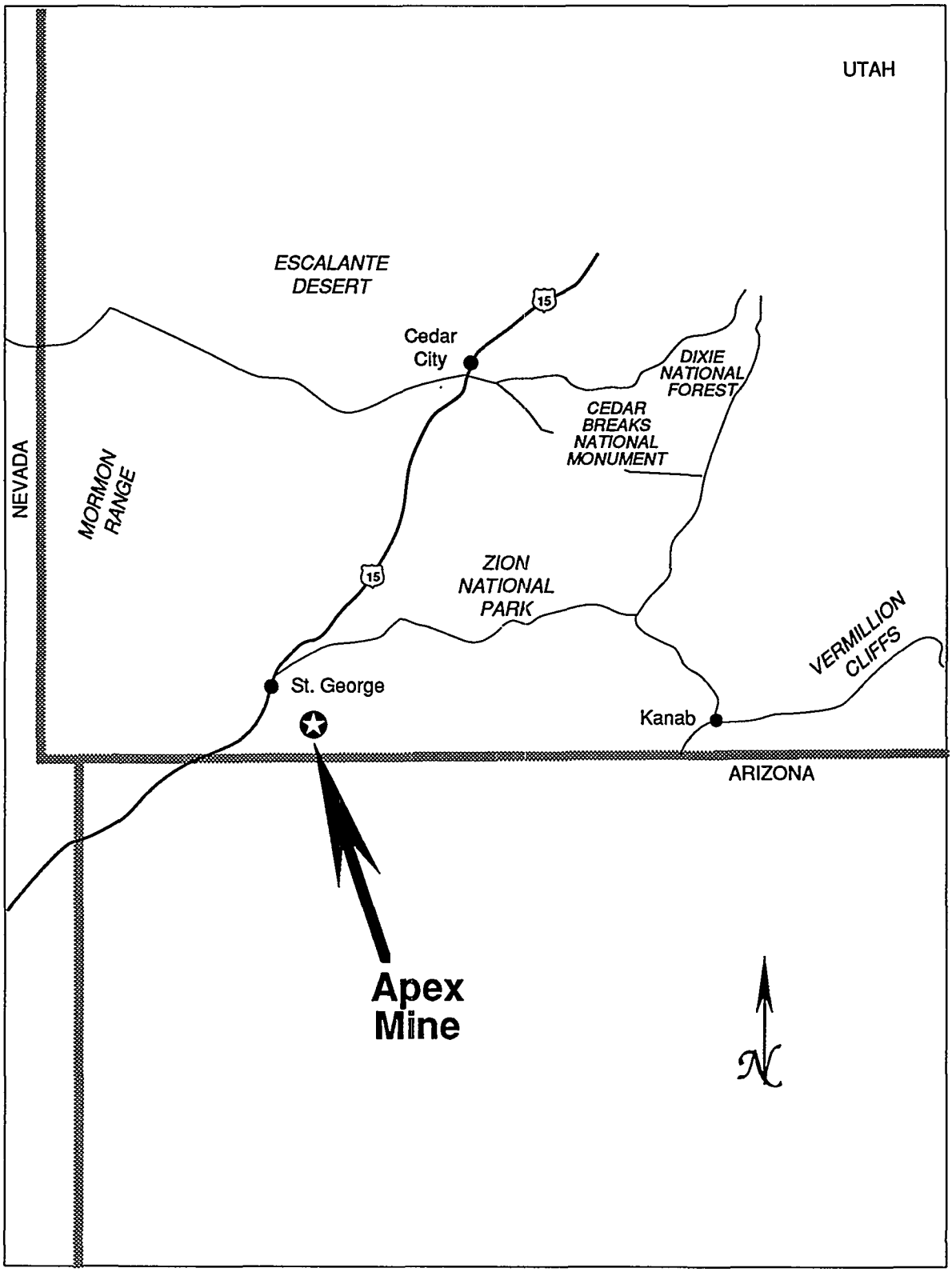
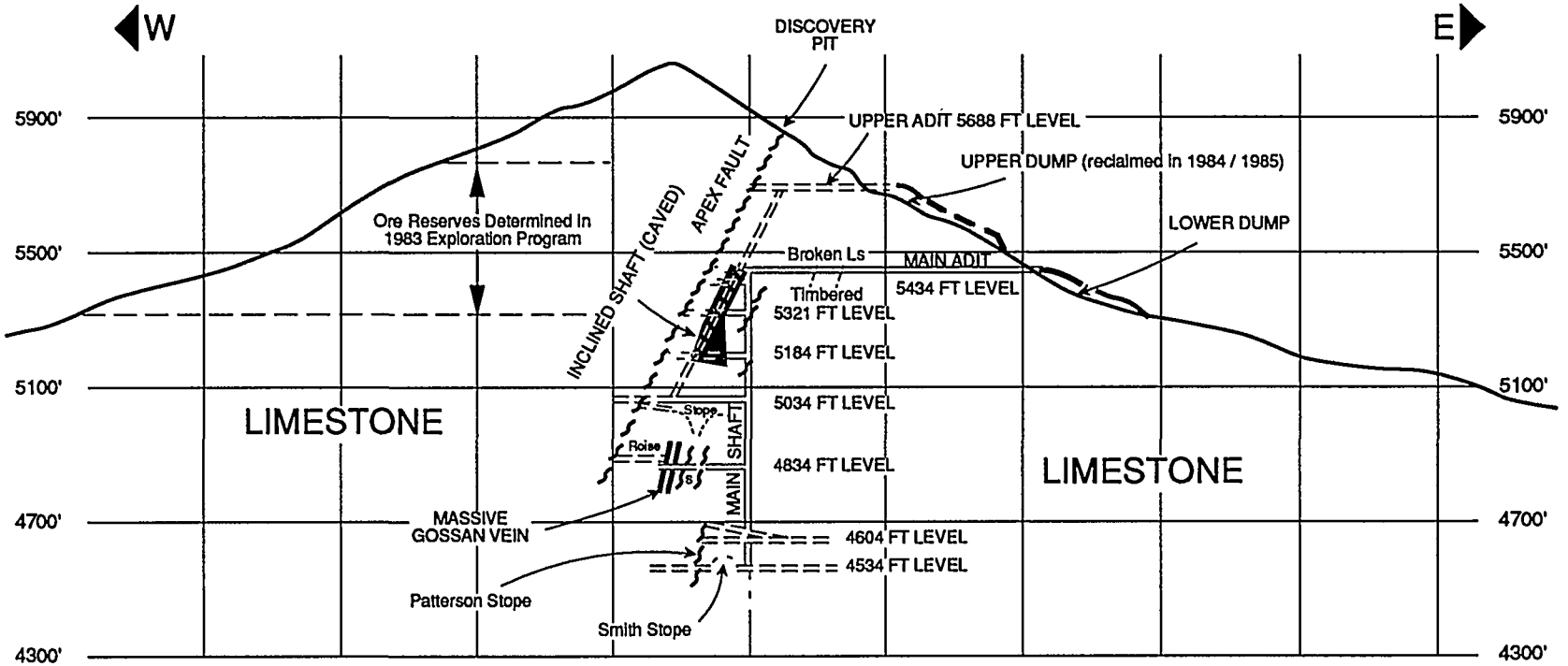


Figure A.3-1
Location of the Apex Mine



ORE ZONE

200 0 200 400 600 800 FEET

MUSTO EXPLORATIONS LTD.		
APEX MINE		
EAST - WEST SECTION OF MAIN SHAFT		
DRAWN L. K.	DATE APRIL 1985	FIGURE :

Figure A.3-2
Section of Apex Mine Workings (1985)

A.3.1.2 Objective of Sealing/Backfilling Operation

Backfilling is required at the Apex Mine for ground control and is an integral part of the mining method. Nearly 100% extraction of the ore zone is possible with backfill. All backfill is pneumatically applied.

A.3.1.3 Purpose

This case history demonstrates the use of pneumatic stowing to enable efficient mining and high-extraction ratios in very poor-quality rock. Pneumatic stowing is being considered for repository backfilling operations, because it requires little water.

A.3.2 Materials

A.3.2.1 Description

Cemented backfill is required to achieve the required performance. Initial mines used gravel from an alluvial deposit. This proved too abrasive, and a switch was made to crushed and sized limestone aggregate. Minus 3/4-in. crushed limestone and 5% portland cement is used.

A.3.2.2 Properties

Sampling of test cylinders has resulted in a 750 pounds per square inch (psi), 20-day strength and a 900-psi, 30-day strength. No other backfill properties are available.

A.3.3 Equipment

All backfilling is performed using pneumatic stowing techniques. After visiting the underground workings, a brief tour of the surface backfill plant was taken. A complete Hanna-Beric backfill plant has been installed on the surface, consisting of a rotary air lock feeder (RALF); a cement silo and infill bin; an electronic control station; and a 150-Kw high-volume, low-pressure compressor. Crushed rock is fed into the pressurized system through the RALF. Immediately after entering the air stream, the rock is blended with cement metered into the air stream via a compression screw feeder.

The 6-in.-diameter fill line was installed with a laser to ensure alignment, and a minimum of bends are used to minimize pressure drops and wear problems. Hanna-Beric-lined elbows are used for important right-angle bends, such as the bend at the portal (Figure A.3-3).

All equipment is available off the shelf.

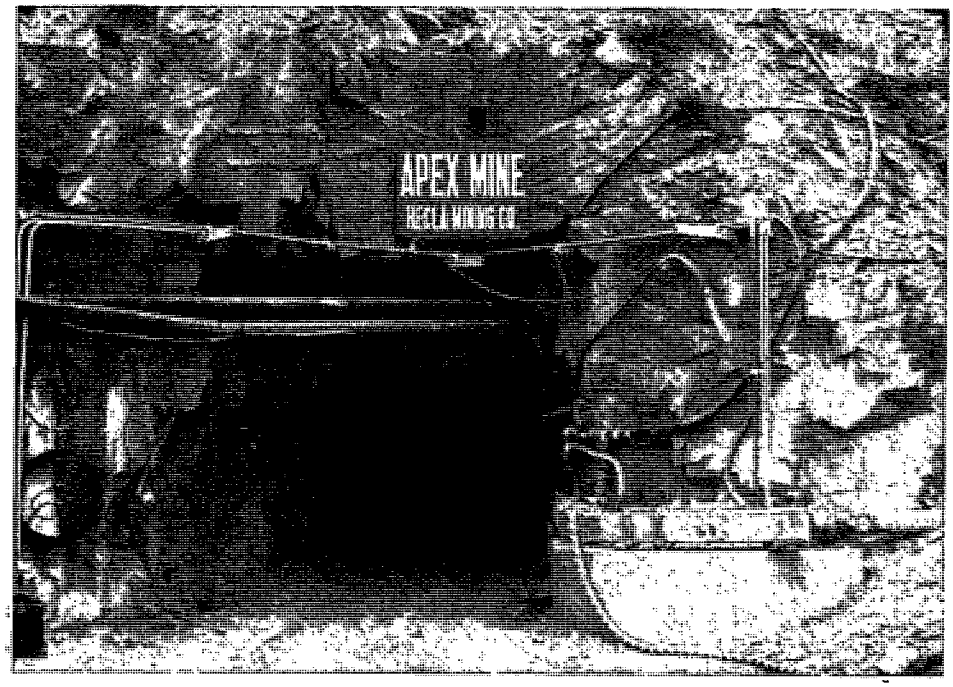


Figure A.3-3
Apex Mine Portal (Note pneumatic fill line at right.)

A.3.4 Seal/Backfill Placement Operations

A.3.4.1 Sequence of Operations

Four-foot-wide, eight-ft-high openings are mined using shotcrete for support. As soon as one cut is completed, it is backfilled, minimizing stand-up time. The system can supply fill at the rate of 35 tons per hour. The operator applies 8 to 10 gal/sec of water at the face, adjusting the amount visually. The mixture is sprayed at 150 ft/sec into each opening.

After 7 days from the time a cut is filled, the adjacent cut can be excavated. When filling this stope, care is taken to achieve a good lateral bond with the adjacent filled stope(s). After 21 days, a drop-cut is excavated beneath the cement cap (Figure A.3-4). With careful backfilling, adequate shear strength will develop between adjacent backfill pours to form a secure back. Because back conditions are improved by the fill, 10- to 12-ft-wide drifts can be excavated and mucked using a 1-yard LHD.

A.3.4.2 Conditions of Emplacement

Ground conditions are very poor at the Apex Mine, as described earlier. The mine is relatively dry.

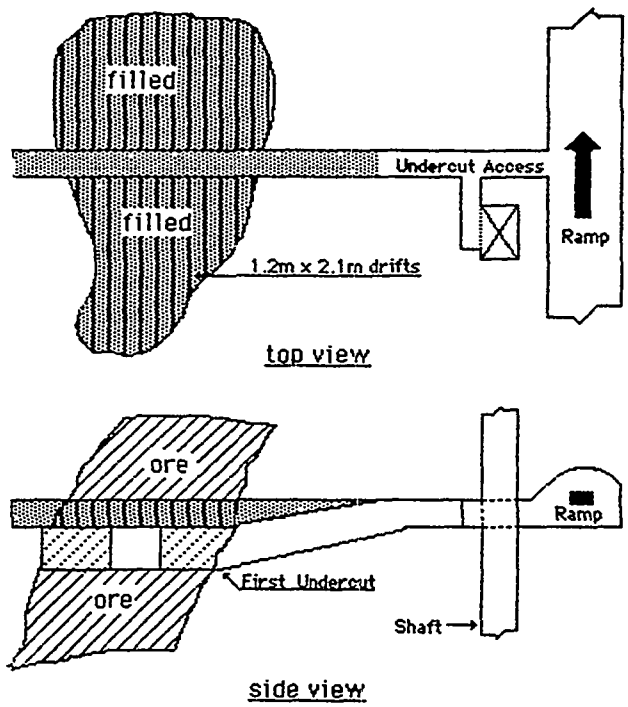


Figure A.3-4
Generalized Apex Method

The fill is transported 800 ft horizontally, then 600 ft vertically down, then 300 ft horizontally to the stope area.

A.3.4.3 Problems Encountered

As with most pneumatic stowing, wear is a problem—especially at the elbows and in the RALF.

A.3.5 Discussion

It is highly unlikely that ground conditions at Yucca Mountain will be as difficult as those encountered at the Apex Mine. However, the pneumatic stowing of backfill used at the Apex Mine could be directly applied to the Yucca Mountain repository to fill drifts during repository decommissioning using sized and screened crushed tuff instead of waste rock. One limitation of the pneumatic backfill system is a limited ability to transport fill horizontally. This can be addressed by spacing the vertical fill holes at appropriate intervals. At the Apex Mine, fill was horizontally transported 800 ft, then down an internal shaft, then

horizontally to the emplacement areas. To minimize the problem of sealing these backfill holes during decommissioning of the Yucca Mountain repository, these holes could be positioned at existing exploration hole locations.

APPENDIX A.4

Sullivan Mine

Case History from Literature

Purpose: To show the successful early application of pneumatic stowing of large open stopes using coarse, uncemented fill.

Table of Contents

A.4.1 Introduction A.4-1

A.4.1.1 Description A.4-1
A.4.1.2 Objective of Sealing/Backfilling Operation A.4-1
A.4.1.3 Purpose A.4-1

A.4.2 Materials A.4-1

A.4.2.1 Description A.4-1
A.4.2.2 Properties A.4-2

A.4.3 Equipment A.4-2

A.4.4 Seal/Backfill Placement Operations A.4-2

A.4.4.1 Sequence of Operations A.4-2
A.4.4.2 Problems Encountered A.4-4
A.4.4.3 Quality Control A.4-5

A.4.5 Discussion A.4-5

A.4.6 References A.4-6

APPENDIX A.4
Sullivan Mine

A.4.1 Introduction

A.4.1.1 Description

The Sullivan Mine is operated by Cominco, Inc., in Kimberly, British Columbia, and has been producing lead and zinc since 1909. The reserves average about 4.5% Pb, 7.1% Zn, and 0.9 oz/ton Ag. The mine is a sublevel open stoping operation with delayed pillar recovery, having a daily production capacity of 10,000 tons. Stopes are large, being 40 to 50 ft wide, 60 to 120 ft high, and as much as 800 ft long.

A.4.1.2 Objective of Sealing/Backfilling Operation

Sublevel stoping is a method of open stoping that produces large primary voids, with stability ensured by leaving ore pillars. Economics typically dictate subsequent pillar recovery, which is best accomplished by introducing backfill into the stopes. If the stopes are steeper than the angle of repose of the fill material, gravity filling can be readily accomplished. However, in the Sullivan Mine, some of the stopes are as flat as 30, requiring mechanical or other powered means of introducing fill to prevent void formation.

Pneumatic stowing was employed at the Sullivan Mine as a means of economically filling the stopes without exposing workers to rockfall hazards.

A.4.1.3 Purpose

The purpose of including this case history is to show the successful early application of pneumatic stowing to backfill large open stopes with coarse, granular uncemented fill. The technology is available off-the-shelf and is suitable for backfilling operations at Yucca Mountain.

A.4.2 Materials

A.4.2.1 Description

The Sullivan Mine orebody contains barren bands of argillite that are broken and drawn with the core. At the mill, a sink/float operation is used to separate the coarser waste particles. The waste from this operation, termed "float waste" or "float," is used as backfill.

The primary rock types found in the float waste are (1) argillites containing up to 50% silica with shear strengths of 25,000 pounds per square inch (psi), and (2) tourmalinized argillite with shear strengths up to 80,000 psi.

A.4.2.2 Properties

Float fill is minus 3 in. and has an angle of repose of 39. No other engineering material properties are available for the Sullivan Mine backfill.

A.4.3 Equipment

Conveyor and rail haulage is used to transport the backfill to the lower levels of the mine. At the stopes, the fill is either placed by gravity or emplaced by throwing methods. Trials were initially made with a belt slinger, but were unsuccessful. Attention was then focused on pneumatic stowing.

Radmark equipment was employed for pneumatic stowing. The Radmark organization—initially a joint venture between Rader Pneumatics of Burnaby, British Columbia, and Markham and Company, Ltd. of Chesterfield, England—developed equipment for the Sullivan operation based on Rader's experience in pneumatic transport of wood products and Markham's experience with stowing in European coal mines.

A Radmark RTL 200 feeder unit was purchased. It was fed by a GR-44 Syntron grizzly feed control unit. A positive-displacement blower with a capacity of 3,550 cubic feet per minute (cfm) at 8 psi supplies compressed air. The blower was powered with a 200-horsepower motor that operated at 800 rpm. Other equipment included pipe, ball joints, bends, and deflectors for the fill line. A two-layer pipe was used that had an inner heavy layer of abrasion-resistant material and a light outer mild-steel container layer. Figure A.4-1 is a schematic cross section through the rotating feeder system.

Although Radmark is no longer open for business, the technology has been transferred to Hanna-Beric, which was recently acquired by J. S. Redpath (Redpath Guill, Inc.).

A.4.4 Seal/Backfill Placement Operations

A.4.4.1 Sequence of Operations

The pneumatic stowing equipment was located underground near the areas to be filled. The blower was located in a main haulage drift, and blowing air was piped to the stower up to

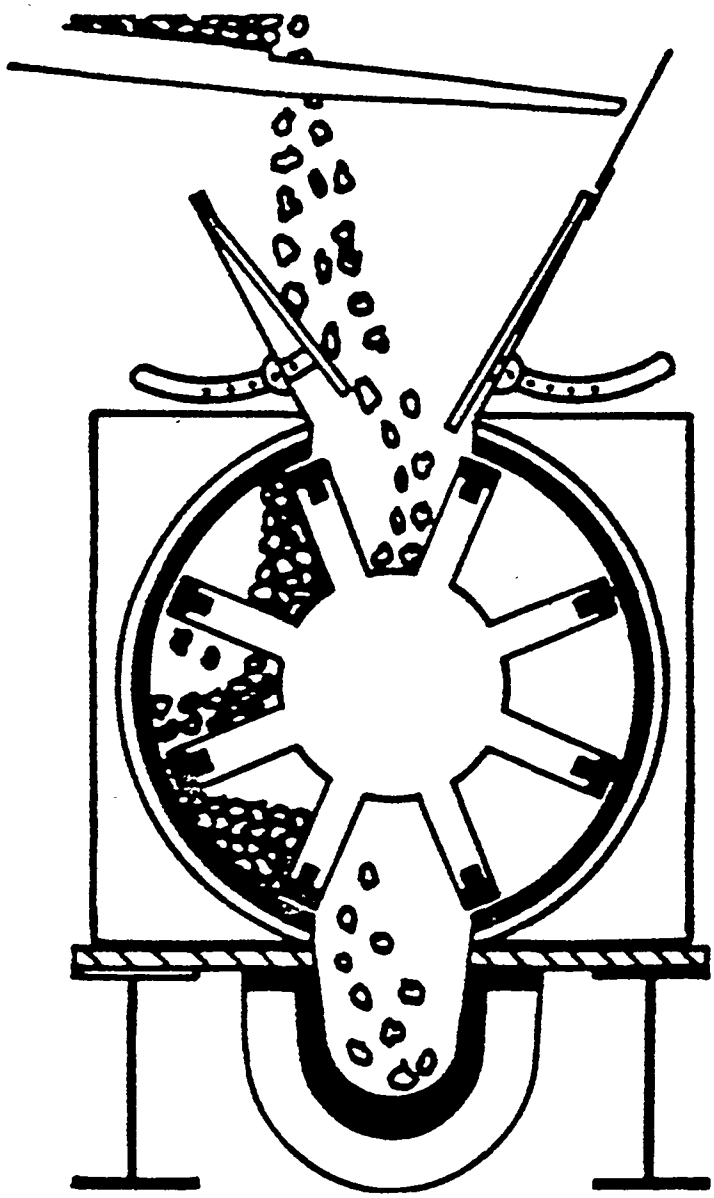


Figure A.4-1
Schematic Cross Section Through Rotating Feeder System

365 ft. In a typical situation (Figure A.4-2), the feeder was located in an access drift along the centerline of the stope to be filled. Fill was supplied via a transfer raise from an overhead haulage dump point.

Filling was done in lifts. Each successive lift was emplaced by extending the fill pipe. Approximately 48,000 tons of backfill were used to fill the 8-35 stope shown in Figure A.4-2.

A.4.4.2 Problems Encountered

Because this was an early trial on pneumatic stowing of abrasive, hard-rock fill, a number of problems were encountered.

Wear was a significant problem, because equipment used previously for coal and wood products was being adapted to transport very hard and abrasive material. Initially, wear on the blade tips and feed-box check pieces limited operation to about 4,000 tons at 100 tons per

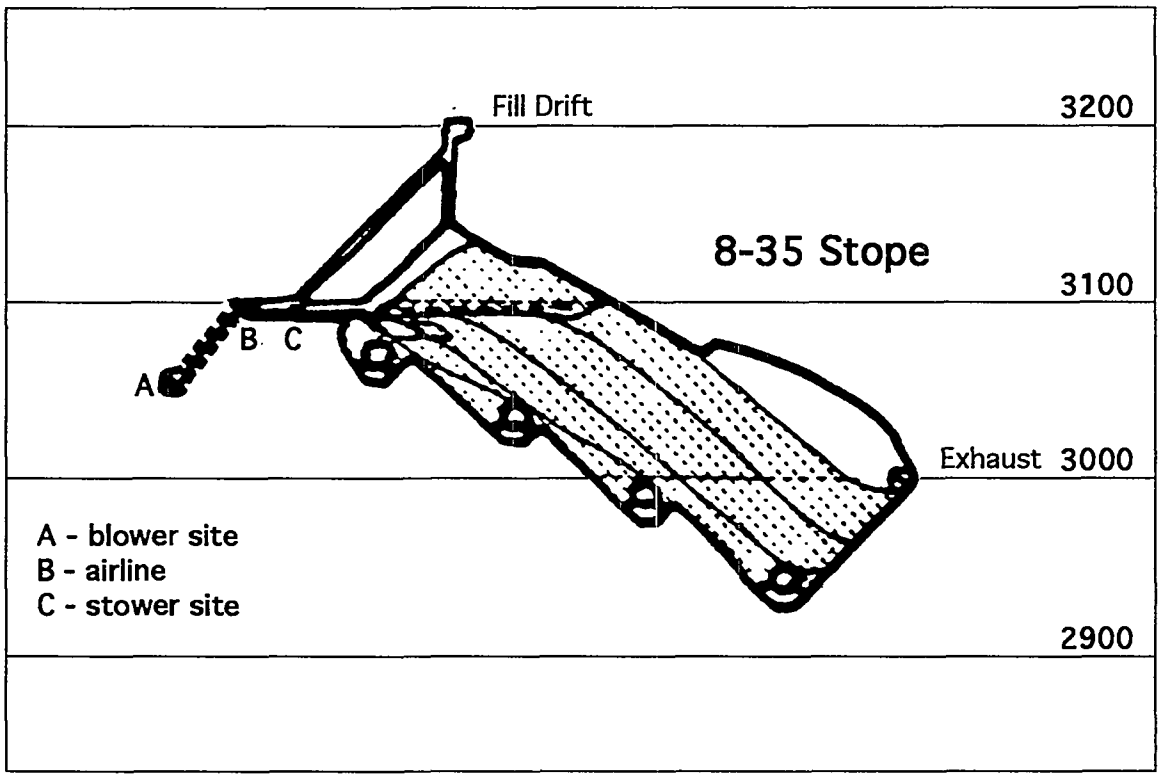


Figure A.4-2
Section Through 8-35 Stope (Elevations in Feet)

hour before rebuilding. A harder facing on the blade tips and replaceable liners for check pieces permitted stowing 100,000 tons or more without feeder rebuild.

Wear was also a problem in the fill pipeline. The air velocity was approximately 90 miles per hour, and in straight sections of pipe, particles up to 1 in. moved with the airstream. Larger pieces (1 to 3 in.) moved along a narrow (4-in.-wide) track, with a rolling action along the bottom of the pipe. This caused wear that could be distributed by rotating the pipe through six different positions at intervals of 50 hrs (or 5,000 tons) per position. Wear was easiest to control on vertical or horizontal pipes; inclined pipes were difficult on curves and deflectors, and wear was more significant. A 90-degree bend would wear through in as little as 5 hrs (or 500 tons). It became necessary to design for frequent replacement of portions of the curves where the rock particles impinged. Also, rock-particle velocity is lowest near the feeder, so a bend should be placed here if possible.

Noise and dust were problems. Noise levels of 109 decibels (dB) were measured at the blower, 103 dB at the feeder, and 107 dB at the end of the blow pipe. This noise level can be reduced by commercially available silencers. Dust is an inherent problem in pneumatic stowing and can only be controlled by ventilation.

A.4.4.3 Quality Control

For the Sullivan Mine application, quality control was not a major concern, as the objective was merely to move float into the voids and fill them as full as economically possible. No special quality control testing was reported.

A large void developed near the hanging wall in the stope shown in Figure A.4-2. This void formed because of the elevation of the fill pipe and the trajectory of the material. It could have been avoided by selecting a lower fill pipe elevation for initial filling.

A.4.5 Discussion

The Sullivan Mine experience is quite applicable to the Yucca Mountain repository. Large voids were filled with coarse, uncemented backfill using (then) prototype equipment. Underground locations for the pneumatic stowing equipment minimized the necessity of providing raises to the ground surface for gravity filling.

A.4.6 References

Reynolds, J. W., 1972. "Pneumatic Backfilling with Crushed Rock at the Sullivan Mine," *Canadian Mining and Metallurgical (CIM) Bulletin*, July, pp. 31-36. (NNA.920902.0004)

APPENDIX A.5

Castlegate Mine

Case History from Personal Communication

Purpose: To demonstrate the technology of using uncemented, pneumatically stowed aggregate to seal mine portals against human intrusion.

Table of Contents

A.5.1 Introduction A.5-1

A.5.1.1 Description A.5-1
A.5.1.2 Objective of Sealing/Backfilling Operation A.5-1
A.5.1.3 Purpose A.5-1

A.5.2 Materials A.5-1

A.5.2.1 Description A.5-1
A.5.2.2 Properties A.5-1

A.5.3 Equipment A.5-1

A.5.4 Seal/Backfill Placement Operations A.5-5

A.5.4.1 Sequence of Operations A.5-5
A.5.4.2 Conditions of Emplacement A.5-5
A.5.4.3 Problems Encountered A.5-5
A.5.4.4 Quality Control A.5-5

A.5.5 Discussion A.5-5

APPENDIX A.5
Castlegate Mine

A.5.1 Introduction

A.5.1.1 Description

The Castlegate Mine is located near the town of Helper, Utah (Figure A.5-1), and was operated for several years by AMAX Coal Company, with a 1987 production of 530,000 tons using longwall and continuous mining methods. The mine was abandoned in 1991.

A.5.1.2 Objective of Sealing/Backfilling Operation

The objective of the sealing operation was to prevent human intrusion into the abandoned mine workings. The seals were not required to impound water; however, tightfilling was desired. The decision was made to use pneumatic stowing techniques.

A.5.1.3 Purpose

This case history demonstrates the technology of using uncemented pneumatically stowed aggregate to seal mine portals against human intrusion.

A.5.2 Materials

A.5.2.1 Description

An unscreened road-base type aggregate was used as fill, with a nominal size of 1.5 in. with fines. The fill was emplaced dry and no cement additives were used.

A.5.2.2 Properties

Because the performance requirements of the fill were minimal, no material testing was performed and material properties are not available.

A.5.3 Equipment

Standard Hanna-Beric-type pneumatic stowing equipment was used. Figure A.5-2 is a diagram of the major components. Table A.5-1 shows the specifications of the equipment.

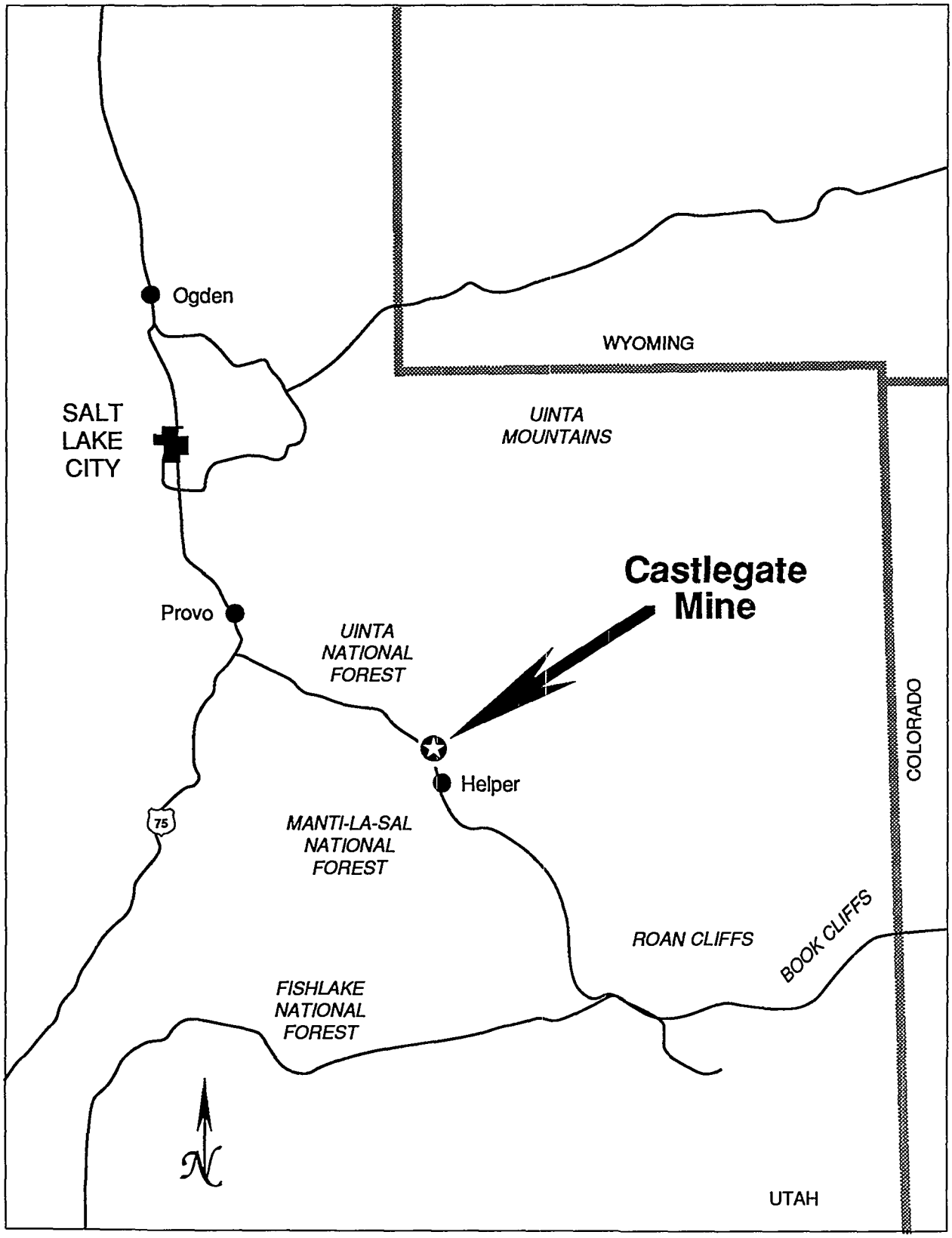


Figure A.5-1
Location of Castlegate Mine near Helper, Utah

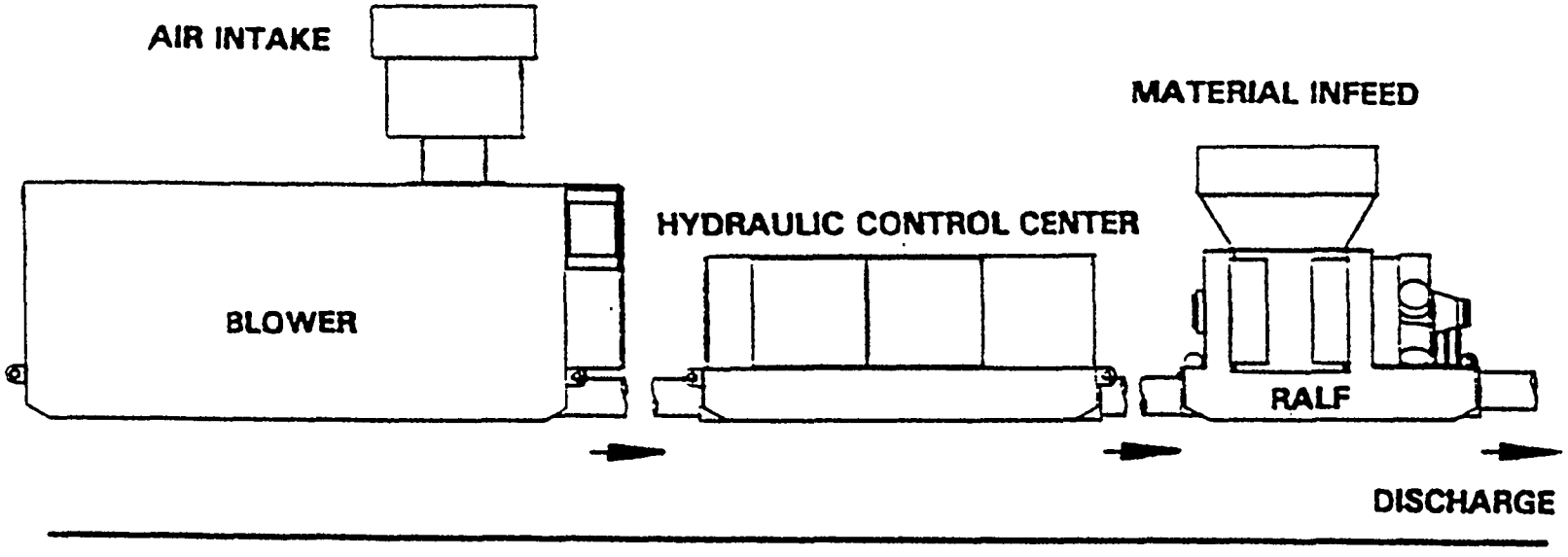


Figure A.5-2
Pneumatic Stowing Equipment

Table A.5-1
Equipment Specifications

MODEL 24D24L			MODEL 164D24L		
BC-400 Blower			**EPP-300 Power Unit**		
Length	130 in.	3302 mm	Length	144 in.	3658 mm
Width	54 in.	1372 mm	Width	68 in.	1727 mm
Height	50 in.	1270 mm	Height	54 in.	1372 mm
Weight	12,000 lbs	5443 kg	Weight	9800 lbs	4455 kg
Power	400 HP	480 VAC	Power	300 HP	480 VAC
Pressure		18 psig	Pressure		20 psig
Hydraulic Control Center					
Length	86 in.	2184 mm			
Width	41 in.	1041 mm	**Hydraulic Control Center**		
Height	39 in.	990 mm	Slave Driven by Power Unit		
Weight	3600 lbs	1633 kg			
Power	40 HP	480 VAC			
RALF (24D24L)			**RALF**		
Length	65 in.	1651 mm	Length	60 in.	1524 mm
Width	34 in.	864 mm	Width	24 in.	508 mm
Height	39 in.	990 mm	Height	29 in.	711.2 mm
Weight	6000 lbs	2722 kg	Weight	2200 lbs	1000 kg
Maximum Material Size	Minus 3 in.		Maximum Material Size	Minus 1.5 in.	

A.5.4 Seal/Backfill Placement Operations

A.5.4.1 Sequence of Operations

A contractor was mobilized (J. S. Redpath from Mesa, Arizona) who was familiar with pneumatic stowing applications. A concrete block stopping was first emplaced by the mine operator. The aggregate was then pneumatically placed against this stopping to a nominal 30-ft thickness. Care was taken to tightfill to the roof. Figure A.5-3 shows the basic filling process. A degree of compaction was attained from the pneumatic stowing process, limiting the amount of future settlement.

A.5.4.2 Conditions of Emplacement

The Castlegate Mine is geologically situated in a bedded sandstone, shale, and coal sequence. At the portals, the roof was typically sandstone, supported by rockbolts, wire mesh, and structural steel with timber lagging. The backfill was placed directly against existing ground support.

A.5.4.3 Problems Encountered

No major problems were reported by the construction contractor. As evidenced in the photos, some dust was generated, but did not become a problem because the equipment was located outside the mine. In an underground situation, water would have to be used to control dust.

A.5.4.4 Quality Control

Quality control was maintained by visual inspection during construction. Figure A.5-4 shows a completed seal in one of the portals, showing the quality of tightfilling.

A.5.5 Discussion

This case history illustrates the dry placement of uncemented crushed-rock backfill using pneumatic stowing to seal an abandoned mine against human intrusion. Similar methods could be used to place general fill or fill structures in an underground repository.

Figure A.5-3
Pipe Installed; Filling in Progress

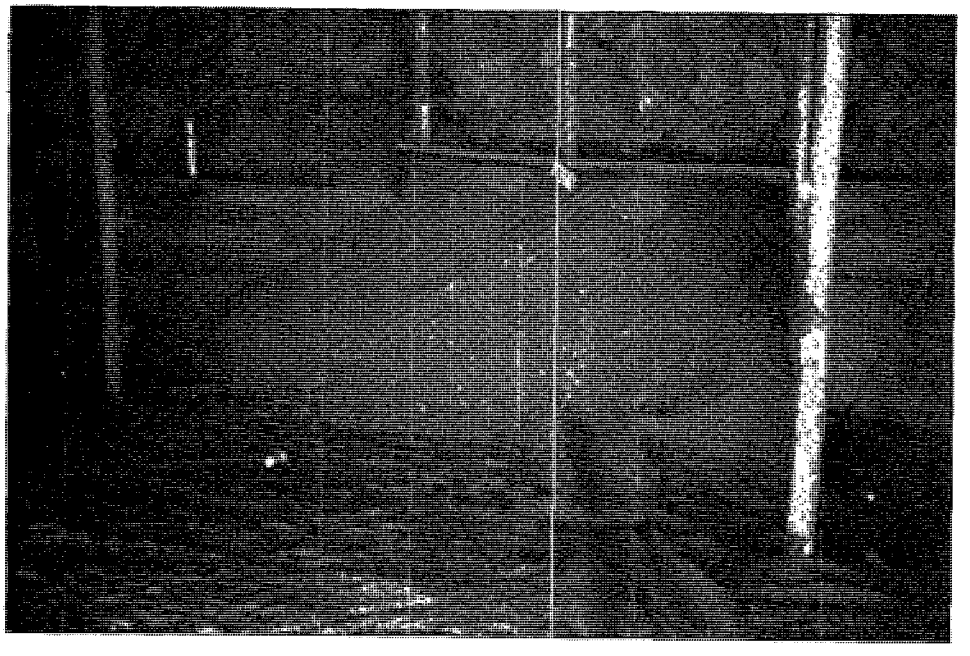


Figure A.5-4
Filling Complete

APPENDIX A.6

Bullfrog Mine

Site Visit: September 3, 1992

Purpose: To demonstrate the use of a very simple system for backfilling mixed stopes using cemented fill.

Table of Contents

A.6.1 Introduction A.6-1

A.6.1.1 Description A.6-1
A.6.1.2 Objective of Sealing/Backfilling Operation A.6-1
A.6.1.3 Purpose A.6-1

A.6.2 Materials A.6-4

A.6.2.1 Description A.6-4
A.6.2.2 Properties A.6-4

A.6.3 Equipment A.6-4

A.6.4 Seal/Backfill Placement Operations A.6-6

A.6.4.1 Sequence of Operations A.6-6
A.6.4.2 Conditions of Emplacement A.6-6
A.6.4.3 Problems Encountered A.6-6
A.6.4.4 Quality Control A.6-6

A.6.5 Discussion A.6-8

APPENDIX A.6
Bullfrog Mine

A.6.1 Introduction

A.6.1.1 Description

The Bullfrog Mine is located near the town of Beatty, Nevada (Figure A.6-1), and has been producing gold and silver using surface mining techniques since 1988. The surface mine has a production capacity of 100,000 tons per day (TPD) and is producing 300,000 oz of gold annually. A large mill with a capacity of 8,000 TPD is located at the site. Originally operated by Bond Gold Bullfrog, Inc., the mine was recently acquired and is presently operated by LAC Minerals, Ltd. The underground operation began in mid-1991 to follow a steeply dipping vein in the pit wall that was too narrow and deep to mine economically using surface methods. All underground excavation is done by drill-and-blast methods with rubber-tired equipment.

The underground mine accesses the vein by means of a ramp in the footwall (Figure A.6-2). The ramp was driven from a portal in the pit wall. Crosscuts from the ramp access the vein. Once in the ore, 4-m by 4-m stopes are driven along the strike and widened 2 m by slabbing. After filling, an adjacent stope can be mined until the full vein width is reached.

A.6.1.2 Objective of Sealing/Backfilling Operation

Cemented backfill is emplaced to enable total extraction of the orebody while minimizing dilution. Open stoping (without backfill) was originally considered, but poor ground conditions caused excessive dilution. Overhand cut-and-fill using cemented backfill was next attempted. The mine is currently using underhand cut-and-fill stoping for most of its production.

A.6.1.3 Purpose

The Bullfrog Mine demonstrates the use of a very simple system for backfilling mined stopes using cemented fill. Although a better fill quality might be achieved using more sophisticated batching and placement procedures, some of the simplicity would be lost. To date, the system has achieved performance requirements at minimum cost. The technology is available and suitable for those backfilling operations at Yucca Mountain where fill performance criteria are not too stringent.

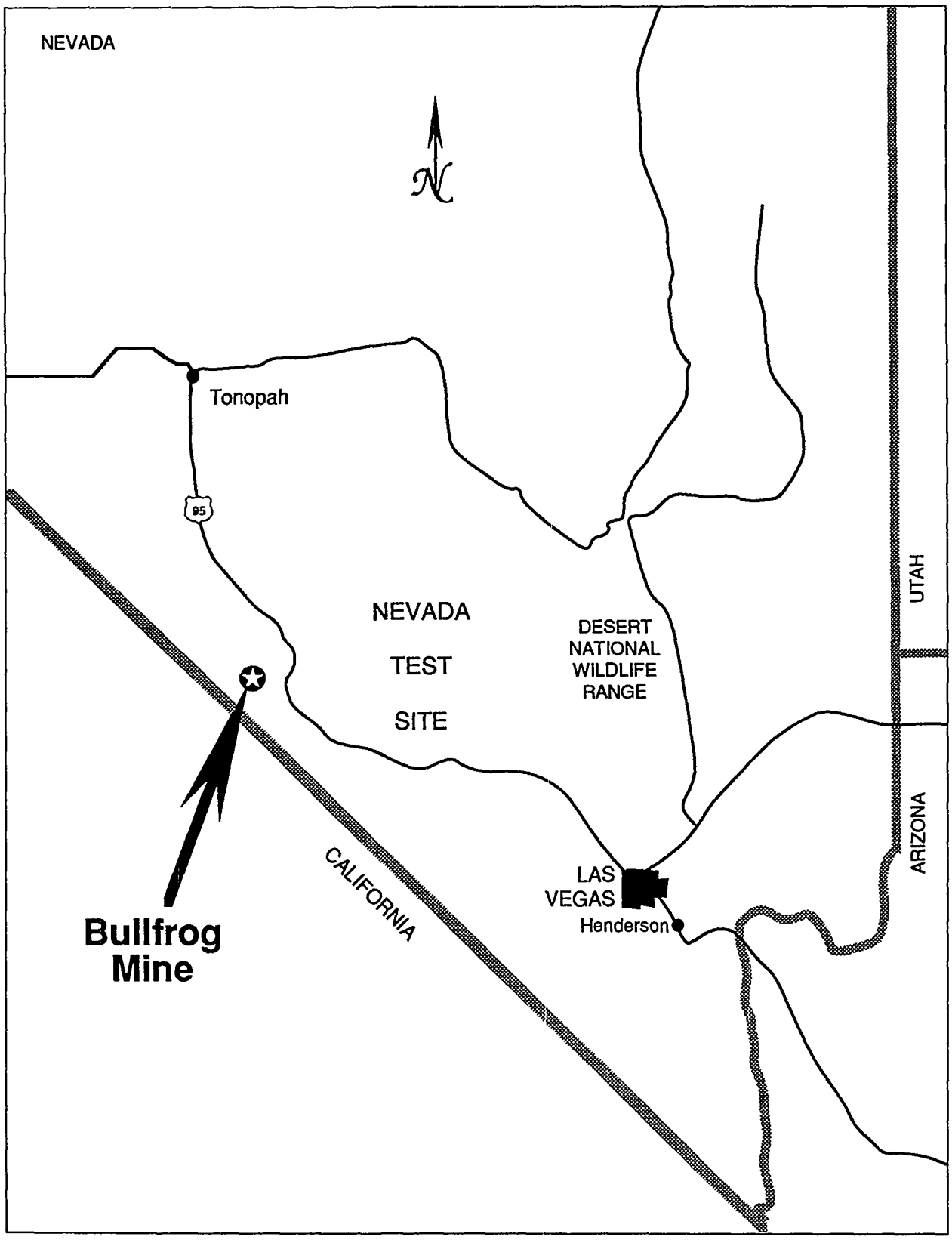


Figure A.6-1
Location of the Bullfrog Mine

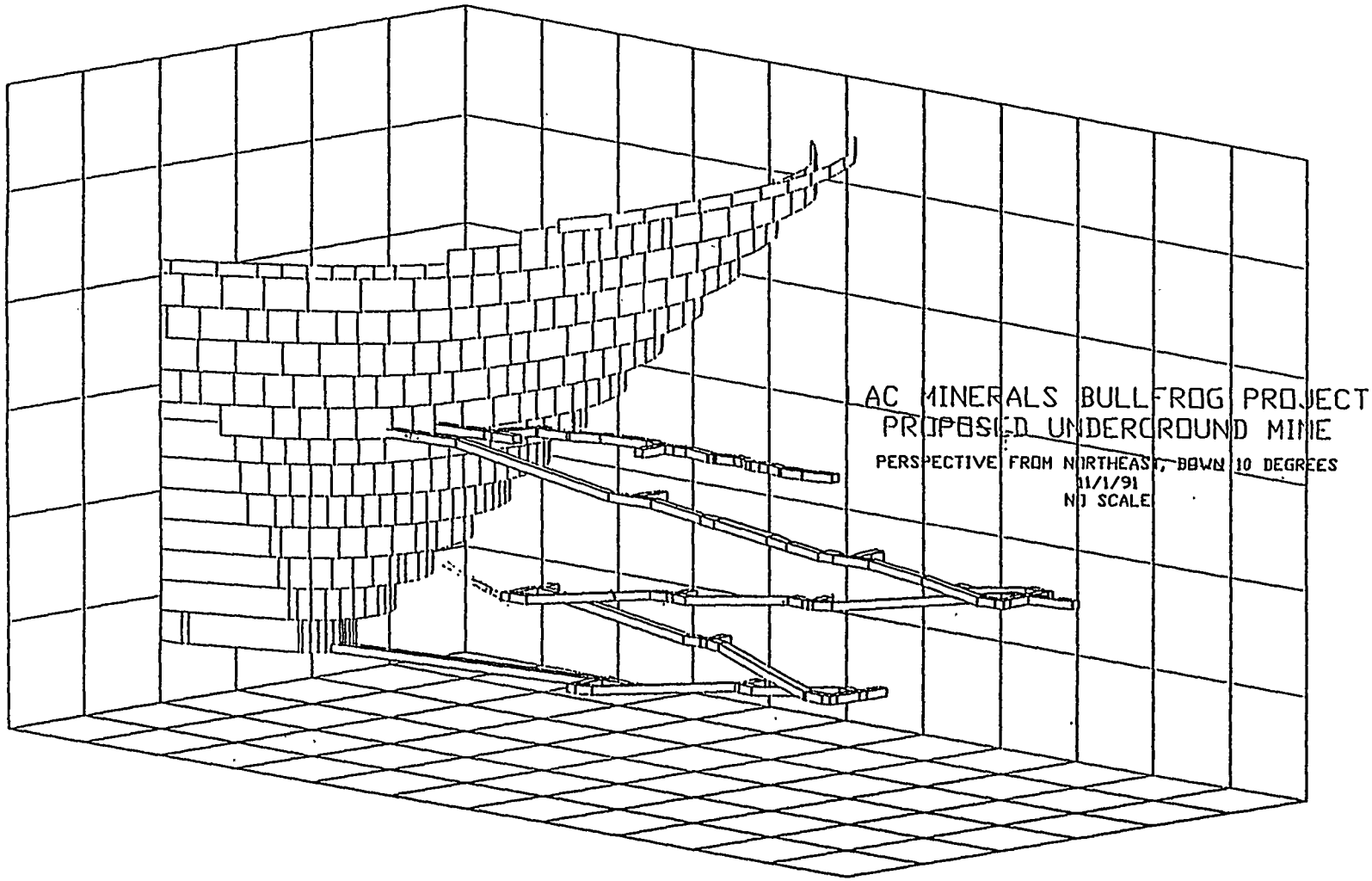


Figure A.6-2
Perspective of Underground Mine

A.6.2 Materials

A.6.2.1 Description

A cemented fill consisting of approximately 95% (by weight) run-of-mine aggregate and 5% cement is used for backfill.

A.6.2.2 Properties

Figure A.6-3 shows a size gradation for the aggregate used at Bullfrog compared with two other sites. The run-of-mine aggregate is approximately 90% (weight) coarse (>3/8 in.). Trial mixes were prepared by screening off the +6 in. rock and adding sand, giving a gradation of 78% coarse and 22% fine. The addition of 5% (by weight) portland cement gave a 28-day strength of 405 pounds per square inch. Other mixes were evaluated using various binder quantities and replacing some of the portland cement with fly ash. Good strengths were obtained with 50% fly ash mixtures and 5% binder.

After consideration of the test results, the mine decided to use run-of-mine aggregate with 5% portland cement binder. Since the time of the visit, the quantity of development muck has been found to be inadequate for production needs, and pit waste was used as a supplemental source of fill. Screening to remove oversize became necessary.

A.6.3 Equipment

Equipment used for preparing and emplacing backfill includes:

- cement silo,
- cement mixer,
- diesel mine trucks (end-dump 15-25 T capacity), and
- load-haul-dump (LHD) unit with tamping plate.

Photos of this equipment are presented during the following discussions of placement operations.

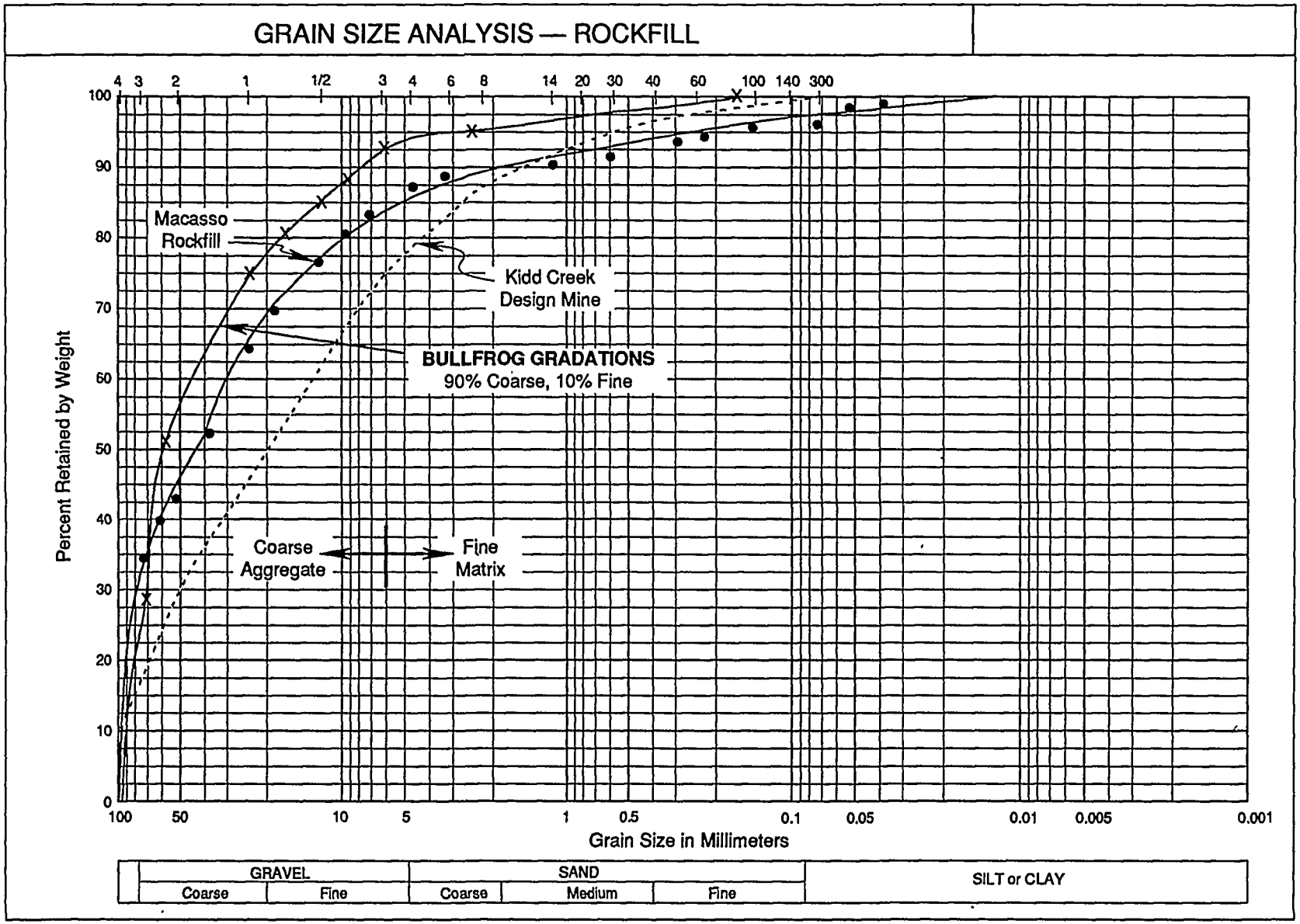


Figure A.6-3

Size Gradation for Bullfrog Run-of-Mine Aggregate Compared with Other Mines

A.6.4 Seal/Backfill Placement Operations

A.6.4.1 Sequence of Operations

The sequence of backfilling operations can be summarized as follows:

- Aggregate is mined.
- A batch of backfill is prepared on surface.
- Fill is transported underground.
- Fill is dumped in drift to be filled.
- Tamping LHD compacts fill.
- Curing takes place without delay to operation.

A mixture of 55% portland cement and 45% water (by weight) is prepared in the mixer and metered into the truck (Figure A.6-4). The fill is mixed during transportation underground; further mixing occurs during fill emplacement.

A.6.4.2 Conditions of Emplacement

The drifts are mostly dry by mining standards. Some hydrothermal water enters through fracture systems, while some perched water is also encountered. The quantities of water have not presented any problems to the backfilling operations so far.

Headroom is limited during all placement activities, but not excessively for underground operations with 4 m of clearance.

A.6.4.3 Problems Encountered

The top of the drift is difficult to reach with the tamping LHD. This problem could be rectified by installing a longer arm on the LHD, but this would cause problems negotiating the corners. The voids near the roof apparently do not cause problems with fill performance.

Due to the method of placements, there are no major problems with fill segregation.

A.6.4.4 Quality Control

Quality control consists of visual inspection of all aspects of the process to ensure the best possible job. There is no systematic plan for sampling or testing of fill.

Although the appearance of the fill is moderate, it is performing adequately.

Figure A.6-4
Cement Being Added to Run-of-Mine Development Muck in Bed of Truck

A.6.5 Discussion

The quality control procedures at the Bullfrog Mine, while adequate for mining purposes, would require tightening for repository purposes. However, the case history illustrates the mechanical placement of rockfill using a simple system.

Since the time of the site visit, stability problems have been encountered during undermining of early backfills, requiring remedial support measures. Quality control procedures have been substantially tightened to achieve a better aggregate size gradation and tighter fill placement. Backfill quality control is very important where underland mining is practiced.

APPENDIX A.7

American Girl Mine

Site Visit: September 13, 1993

Purpose: To define the practical limits of currently available pneumatic stowing in applications requiring long horizontal transport and to demonstrate the superiority of rubber-tired backfilling methods where flexibility and high backfilling rates are important.

Table of Contents

A.7.1 Introduction A.7-1

A.7.1.1 Description A.7-1
A.7.1.2 Objective of Sealing/Backfilling Operation A.7-4
A.7.1.3 Purpose A.7-4

A.7.2 Materials A.7-4

A.7.2.1 Description A.7-4
A.7.2.2 Properties A.7-4

A.7.3 Equipment A.7-4

A.7.4 Sealing Operations A.7-6

A.7.4.1 Sequence of Operations A.7-6
A.7.4.2 Conditions of Emplacement A.7-6
A.7.4.3 Problems Encountered A.7-6
A.7.4.4 Quality Control A.7-7

A.7.5 Discussion A.7-7

A.7.6 References A.7-8

APPENDIX A.7
American Girl Mine

A.7.1 Introduction

A.7.1.1 Description

The American Girl Mine (Towner and Keifer, 1991) is located in the Cargo Muchacho Mountains near the town of Winterhaven in the Imperial Valley of southern California (see Figure A.7-1). The mining district has been producing gold since the early 1900s, and the American Girl orebody was successfully mined by room-and-pillar methods in the 1930s. In 1987, Eastmaque Gold Mines of Canada acquired the property from Newmont and hired a contractor, Small Mine Development, to begin underground development. Approximately 2,500 ft of main decline, 1,500 ft of ventilation decline, and 1,500 ft of level development had been completed by 1990, when the American Girl Mining Joint Venture—an equally divided partnership between Eastmaque Gold Mines and Morrison Knudsen (MK)—was established, with MK being the operating partner. The operations currently include an open pit; an underground mine; and a mill, with a total gold production of 7,500 oz/month. The underground operation produces more than half the total gold from a production of 800 to 1,000 tons/day of ore (average grade 0.32 oz/ton) and waste.

The Cargo Muchacho mountain range is primarily composed of igneous intrusive rocks (gneissic biotite granite) with inclusions of metasedimentary rocks (the Tumco formation). Intense low-angle thrust faulting has created shear zones and quartz veins where gold occurs. The present mining is primarily in the B-zone shear (see Figure A.7-2), dipping at 19 degrees and varying from 7 to 30 ft in thickness. The hanging wall is a sheared gneiss and the footwall, an altered granite. The footwall contact is a detachment fault surface.

A rubber-tired drift-and-fill method was selected for the mine because of the low angle of the ore body and poor ground quality in the shear zone above the ore. Eight- to ten-ft wide cuts are laid out on an apparent dip of 22.5% for gradeability of rubber-tired equipment and are mined using full-face drifting methods. Production drilling is by jacklegs or single boom jumbo with hydraulic hammers. Ammonium nitrate and fuel oil is used for blasting. Bolting is accomplished with jacklegs, and the round is mucked out using 2.5-yd load-haul-dump (LHD) units.

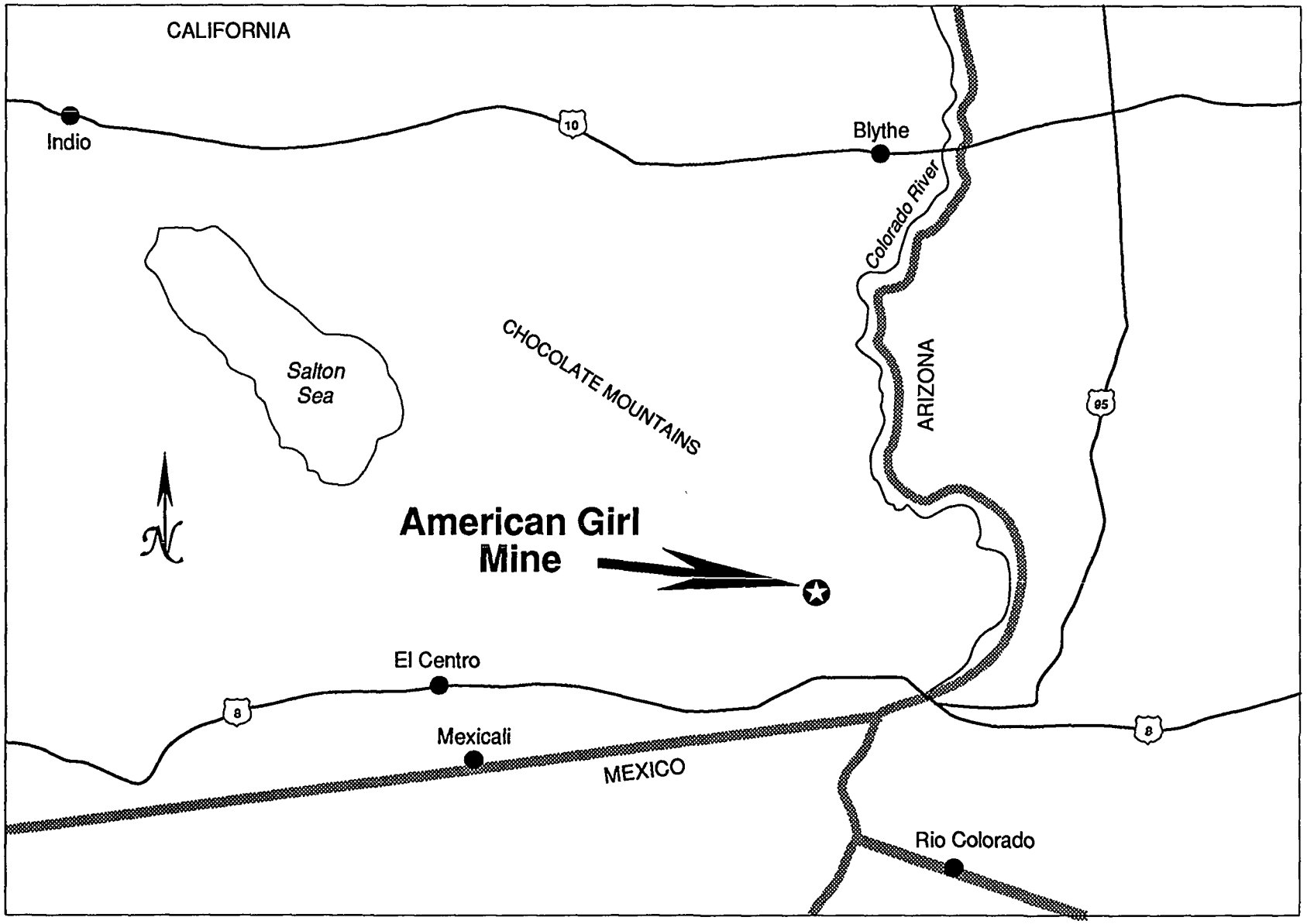


Figure A.7-1
Location of the American Girl Mine

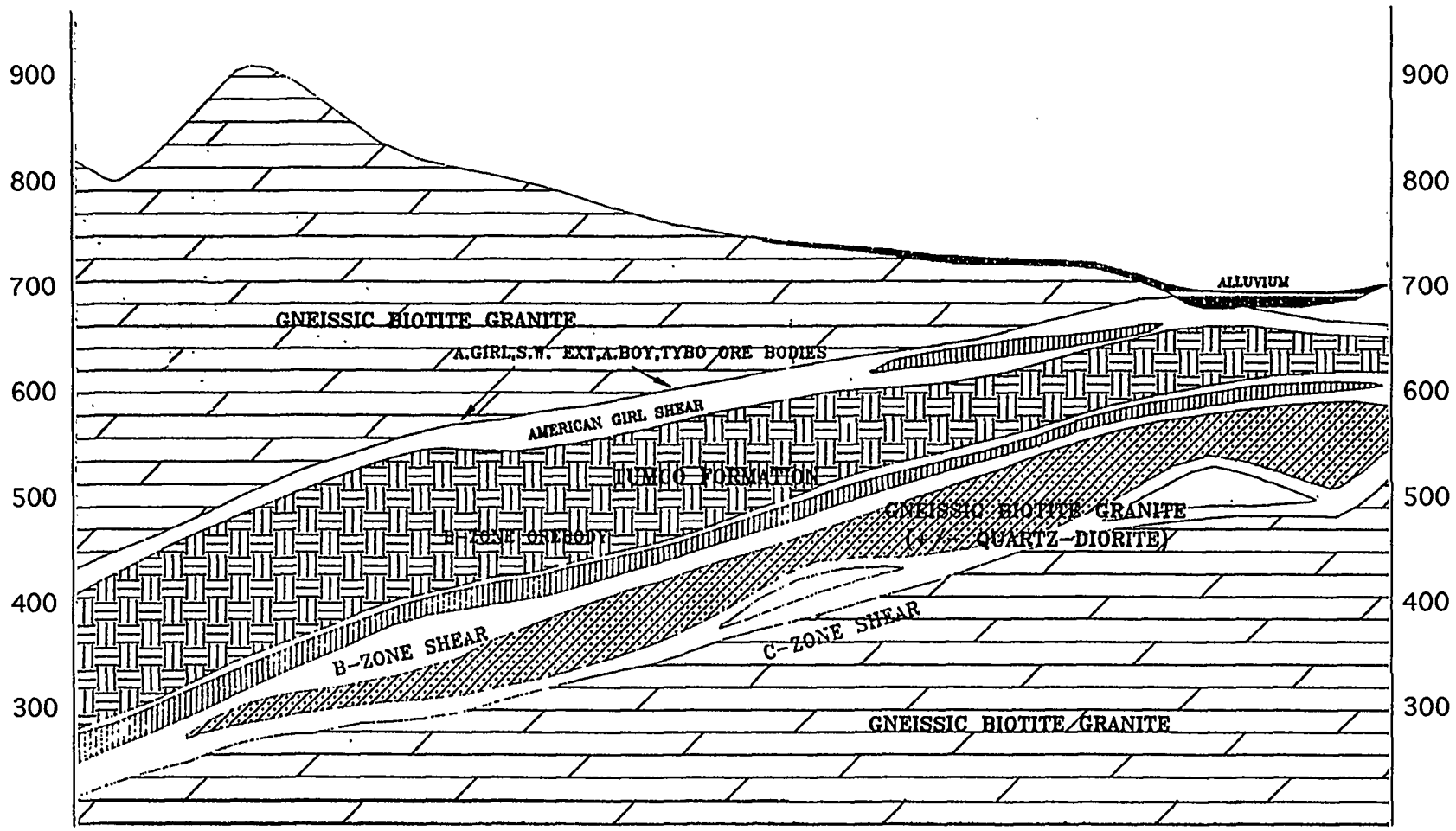


Figure A.7-2
Generalized Cross Section

A.7.1.2 Objective of Sealing/Backfilling Operation

Cemented backfill is emplaced to enable near total extraction of the orebody. The cuts are backfilled in adjacent groups of two or three, making backfill "pillars" to replace extracted ore and maintain stability. Cement is required for strength and cohesion of the pillar sidewalls during secondary extraction (between fill pillars).

A.7.1.3 Purpose

The American Girl Mine case history defines the practical limits of currently available pneumatic stowing technology and demonstrates the superiority of rubber-tired backfilling methods where flexibility and high backfilling rates are important.

A.7.2 Materials

A.7.2.1 Description

A cemented backfill is batched on the surface using a commercial batch plant. A pug mill is used for mixing. Aggregate is alluvial wash material that is mined on site and crushed to -3/4 in. for placed fill and -3/8 in. for pumped. Mine waste was investigated for use as fill, but found to contain excessive clay, which coats the cement particles. The aggregate is mixed with 6 to 7% portland cement (Type II). A 1:1 ratio of water to cement is used. A water-reducing admixture is added to the mix to capture excessive water. The cement and water fraction is increased if fill is to be pumped.

A.7.2.2 Properties

The backfill is designed to have a cylinder strength of 1,000 pounds per square inch (psi). Test results show a range between 450 and 1,400 psi. The emplaced density is approximately 140 lb/cu ft. No other properties are available.

A.7.3 Equipment

Equipment used for preparing and emplacing backfill includes:

- batch plant with pug mill,
- 5-ton backfill trucks, and
- dozer boom LHD (JCI, 12-ft boom, 3-ft-square push plate) (Figure A.7-3).

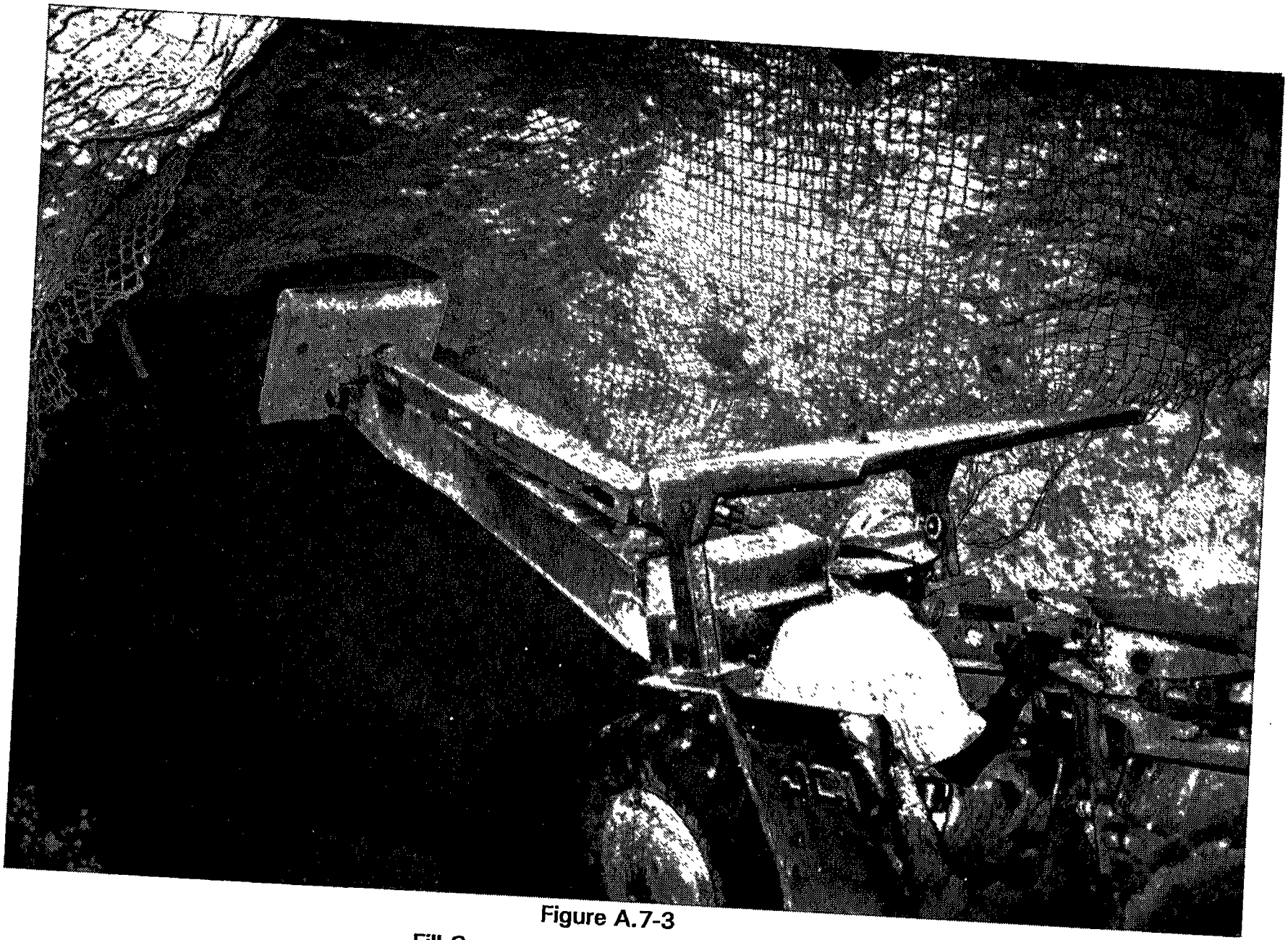


Figure A.7-3
Fill Compaction Using Special Dozer

A.7.4 Sealing Operations

A.7.4.1 Sequence of Operations

In general, each cut is mined downdip at a 22.5% grade. Backfilling proceeds updip. The grade facilitates placement of fill, which is assisted by gravity. The sequence of backfilling operations can be summarized as follows:

- Aggregate is mined and crushed.
- A batch of backfill is prepared on surface.
- Fill is transported underground by truck.
- Fill is dumped at face of cut to be filled.
- Dozer boom LHD compacts fill.
- Cycle is repeated until cut is completely filled.
- If cut is high, fill is placed in two lifts.
- Last load of fill is allowed to set, then trimmed with an LHD to leave a vertical face in the access drift.

In some locations, it is necessary to place fill where gravity worked against the filling operation. In such cases, pumping is necessary. Separate pumping and vent pipes are preplaced in the void, with the vent pipe terminating at the high point.

A.7.4.2 Conditions of Emplacement

The 8 by 8-ft to 10 by 10-ft drifts are mostly dry. The 22.5% downgrade in the cuts greatly facilitates tight placement of the fill to the back.

A.7.4.3 Problems Encountered

Initial mining was conducted using pneumatic stowing. A Hanna-Beric pneumatic backfill system was purchased and installed in 1989. An 8-in. conveying pipeline made of special abrasion-resistant pipe was installed in the decline. Backfilling was ready to commence in June 1990. It soon became evident that there were major problems with the application of state-of-the-art pneumatic stowing at this mine.

- The backfill emplacement rate was inadequate to keep up with mining.
- The system was inflexible and obstructed the access drifts.
- The pipe and rotary air-lock feeder (RALF) experienced high rates of wear.
- The fill quality was poor, with voids and segregation.

After 5 months of intensive effort, it was determined that pneumatic stowing was not workable at this operation. The required horizontal transport distance (2,000 ft) was too long, resulting in back pressure that severely reduced the placement capacity. Terrain did not permit relocation of the backfill plant vertically over the fill areas to alleviate the back pressure. The mine layout necessitated numerous turns in the pipe, which exacerbated the back pressure problem and also led to increased wear. Wear rates were also high because the available fill material was siliceous and very abrasive.

The poor fill quality raised safety concerns in high back areas. Batching was affected by the back pressure and was difficult to control. Tight filling was difficult due to turbulence.

Backfill pipes obstructed the access drifts, restricting access by rubber-tired equipment. The system was prone to scheduled delays (such as dropping a section of pipe on retreat) and unscheduled delays for maintenance.

Consideration was given to relocating the RALF underground, moving it as required to maintain filling progress. The extra effort was not deemed cost-effective compared with mechanical emplacement.

Only 12,000 tons of fill were placed in this 5-month period, approximately one-tenth of the ore production during that period.

A.7.4.4 Quality Control

Backfill quality control is based on two test cylinders taken each day at the batch plant. These are broken at 8 and 28 days. In-place backfill is periodically cored for testing. Quality of the filling process is largely controlled by visual inspection.

A.7.5 Discussion

This case history underscores the practical considerations that impact the application of pneumatic stowing. For successful stowing, a nonabrasive fill is recommended. Stowing equipment should be located either close to the backfilling site or vertically above it; long horizontal runs are to be avoided. Relatively large drifts with few turns are required for effective fill pipe layout. If flexibility and high-production rates are required, pneumatic stowing is not favored.

A.7.6 References

Towner, R. K., and J. W. Keifer, "American Girl Underground Mine," presented at the 97th Annual Northwest Mining Association Convention, Spokane, Washington, December 4, 1991.

APPENDIX A.8

Basalt Waste Isolation Program (BWIP)
Near-Surface Test Facility (NSTF)

Case History from Literature

Purpose: A case study in backfilling underground openings using mechanical methods. The backfilling was performed to decommission stable tunnels with low extraction ratios, similar to the expected condition at a repository. The sequencing of backfilling was largely dictated by access and ventilation considerations.

Table of Contents

A.8.1 Introduction A.8-1

A.8.1.1 Description A.8-1
A.8.1.2 Objective of Sealing/Backfilling Operation A.8-1
A.8.1.3 Purpose A.8-4

A.8.2 Materials A.8-4

A.8.2.1 Description A.8-4
A.8.2.2 Properties A.8-4

A.8.3 Equipment A.8-4

A.8.4 Seal/Backfill Placement Operations A.8-5

A.8.4.1 Sequence of Operations A.8-5
A.8.4.2 Conditions of Emplacement A.8-5
A.8.4.3 Problems Encountered A.8-9
A.8.4.4 Quality Control A.8-9

A.8.5 Discussion A.8-9

A.8.6 References A.8-11

APPENDIX A.8

Basalt Waste Isolation Program (BWIP) Near-Surface Test Facility (NSTF)

A.8.1 Introduction

A.8.1.1 Description

The Basalt Waste Isolation Program (BWIP) was formerly a part of the National Waste Terminal Storage Program of the U.S. Department of Energy. Located in Hanford, Washington, it was one of three candidate sites for the nation's first high-level nuclear waste repository under the Nuclear Waste Policy Act (NWPA) of 1982. In the 1970s, a Near-Surface Test Facility (NSTF) was constructed at the site as a multipurpose field test facility, enabling experiments to be conducted in basalt before access was available at the proposed repository horizon. The NSTF was located at Gable Mountain on the Hanford Site (Figure A.8-1), about 25 mi northwest of Richland, Washington. Figure A.8-2 shows the layout of the NSTF, consisting of three access tunnels connected by test areas, with a total of 3,000 lineal ft of excavation. A number of rock mechanics field experiments were conducted at NSTF through the 1980s.

A.8.1.2 Objective of Sealing/Backfilling Operation

In 1987, the NWPA was amended, resulting in termination of the BWIP. Reclamation of existing disturbances at the Hanford Site was required. Although the environmental assessment for the NSTF (DOE, 1978) did not specifically call for backfilling and sealing the NSTF, the sensitivity of the site as a religious location for traditional local Native American groups and other considerations led to the decision to backfill. The ultimate objective was to restore the site as closely as possible to its original condition. The NSTF reclamation plan (DOE, 1988) called for the following reclamation activities:

- Excavate bench material and place it in the NSTF.
- Install permanent plugs in power drop and exhaust ventilation boreholes.
- Plug exploratory boreholes south of ventilation fan pad.
- Construct cast-in-place concrete curtain wall at tunnel portal.
- Backfill tunnel bench cuts to original profile with bench material.

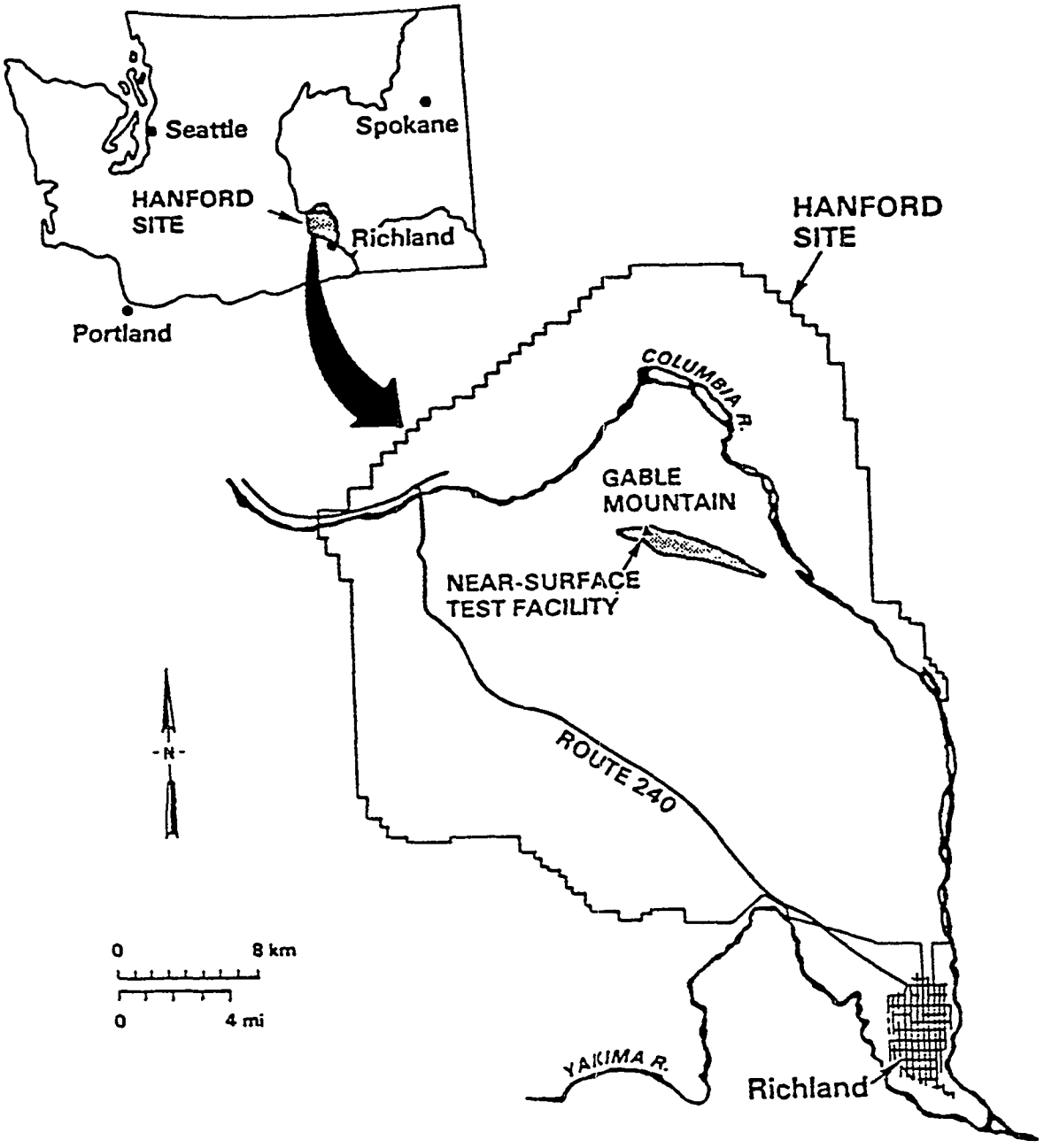


Figure A.8-1
Location of the Near-Surface Test Facility

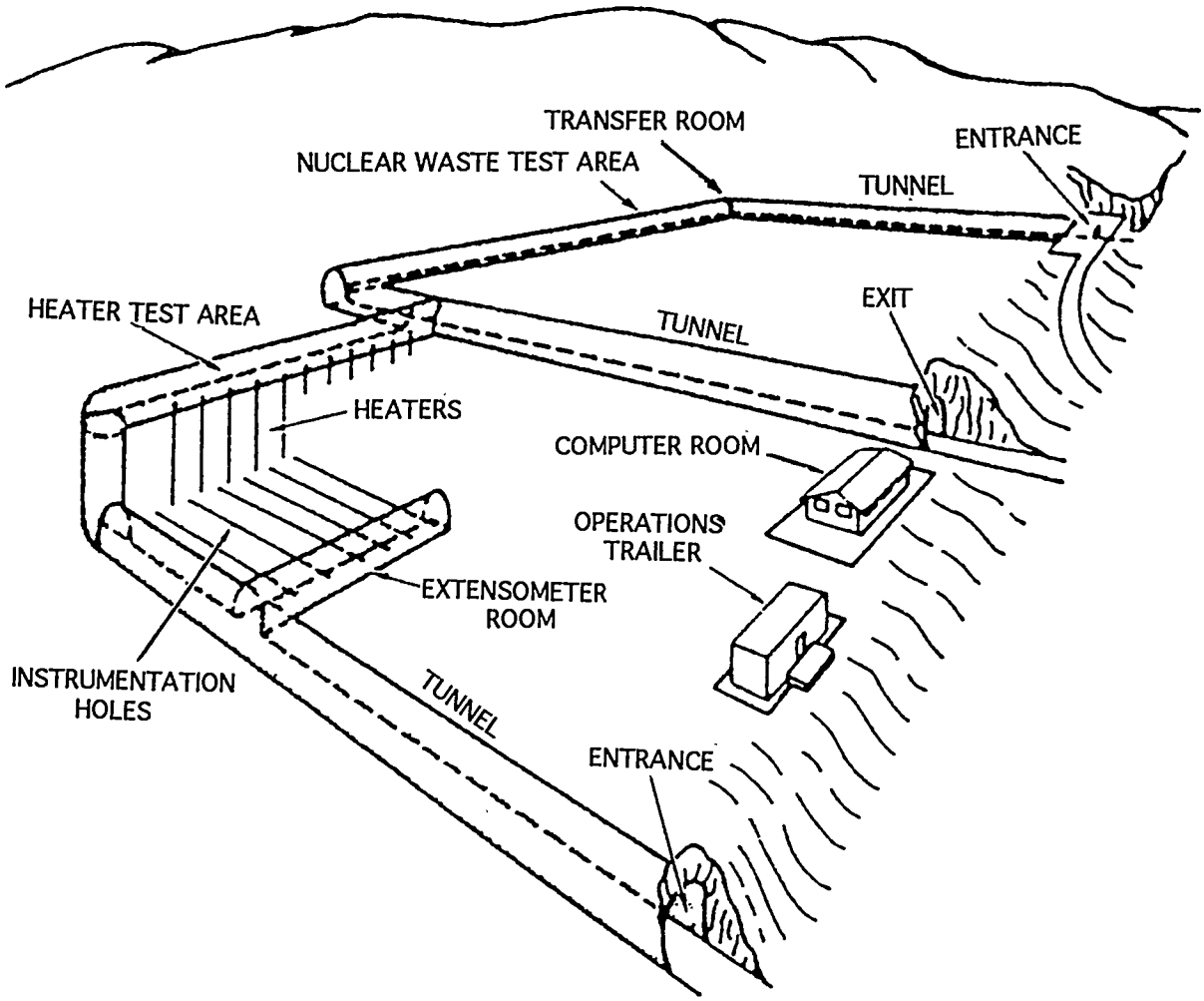


Figure A.8-2
Near-Surface Test Facility Layout

A.8.1.3 Purpose

Backfilling and sealing the NSTF is a case study in backfilling underground openings using mechanical methods. NSTF backfilling was performed to decommission stable tunnels with low extraction ratios, similar to the expected situation at Yucca Mountain. In such cases, the backfilling sequence is largely dictated by access and ventilation requirements. Other backfilling case histories documented in this report used backfill as a means of providing ground support to permit high mineral extraction ratios under difficult ground conditions. Backfilling was sequenced to closely follow mineral extraction as dictated by stability considerations.

A.8.2 Materials

A.8.2.1 Description

During the excavation of the NSTF by drill-and-blast methods, mined material was placed in a bench at the portal elevation to provide a level area for support facilities. This material was ungraded basalt and was covered with a thin layer of crushed stone to make a suitable surface. The volume of the bench was approximately 121,000 loose cubic yards. The scope of the reclamation project called for removal of this bench, with the most expedient place for disposal being the underground excavations and the portal box cuts. Because there were no requirements regarding the in-place fill characteristics, no special treatment was given to the rock prior to its use as fill.

A.8.2.2 Properties

There is no information on the properties of the fill, either loose or as emplaced. A rough calculation considering bench and emplaced volumes gave an in-place compaction of 89%.

A.8.3 Equipment

Mostly standard earth-moving equipment was used for the mechanical backfilling operations. Several sizes of equipment were used to best fill the different excavation sizes. Most of the fill was moved to the remote areas using 10-yd dump trucks. Front-end loaders (988B) were used to excavate the bench material and load the trucks. Load-haul-dump (LHD) units (3-cy) were also used to place backfill, and specially modified D8 bulldozers were used to spread the fill. The modification consisted of a second, smaller blade approximately 10 ft ahead of, slightly higher than, and in tandem with the standard blade. This modification permitted higher placement of fill in openings having an arched roof.

A.8.4 Seal/Backfill Placement Operations

A.8.4.1 Sequence of Operations

Backfilling began on June 28, 1988, and was completed on August 22, 1988. Sequencing was largely dictated by access and ventilation considerations and merits further discussion.

Trucks were loaded from the bench using a front-end loader. Backfill was dumped on the floor of the room to be filled and spread with bulldozers. At least 8 ft of clearance was left on top of the initial lift. Working in this space, LHDs and bulldozers could place an additional lift of fill to within 2 to 3 ft from the back. Tighter fill placement was possible, but not required, and the void space was used to maintain flow-through ventilation. Ramp grades in the fill were maintained less than 15% to permit safe operation of LHDs.

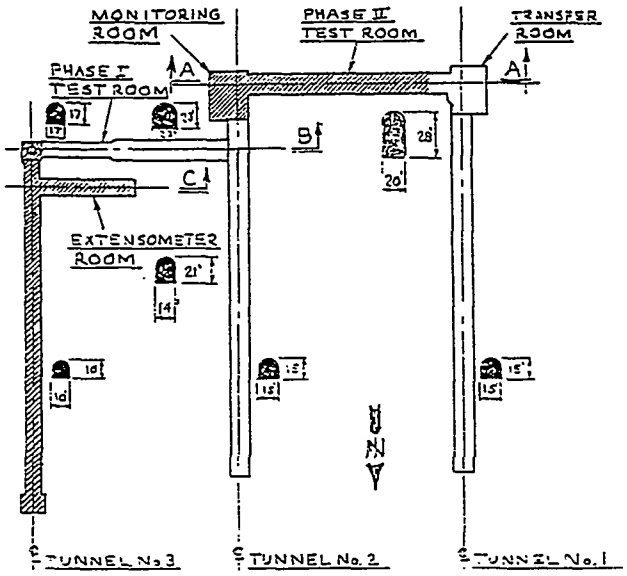
The basic sequence is shown in the plan on Figure A.8-3. Backfill was placed on the retreat, starting from the most remote rooms. When openings required for ventilation were buried, ventilation pipes were left in the fill to maintain the flow.

Figures A.8-4 and A.8-5 are as-built sketches showing the condition of the fill at project completion. Lifts placed by trucks and LHDs can be identified in these sketches.

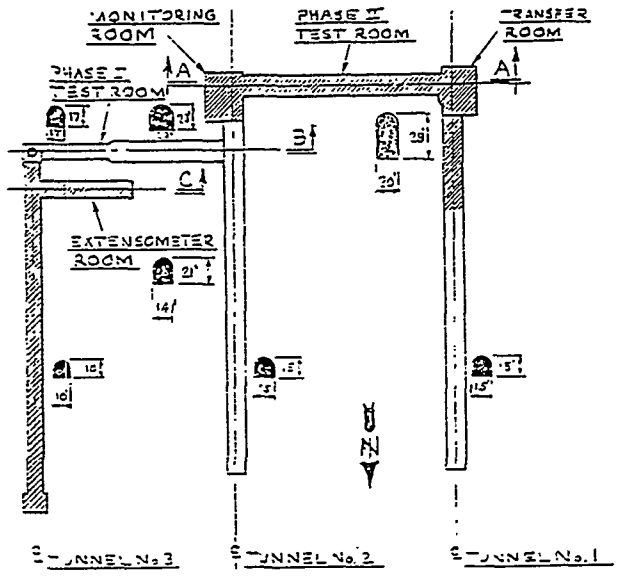
After backfilling was complete, the boreholes and access tunnel portals were sealed. Reinforced concrete plugs having thicknesses from 12 to 28 in. were used to seal the portals. The electrical and ventilation drops were sealed with grout from surface. The underground facilities were considered sealed on September 12, 1988. Following this, the box cuts were filled and other earthwork performed to bring the mountain to its original contour.

A.8.4.2 Conditions of Emplacement

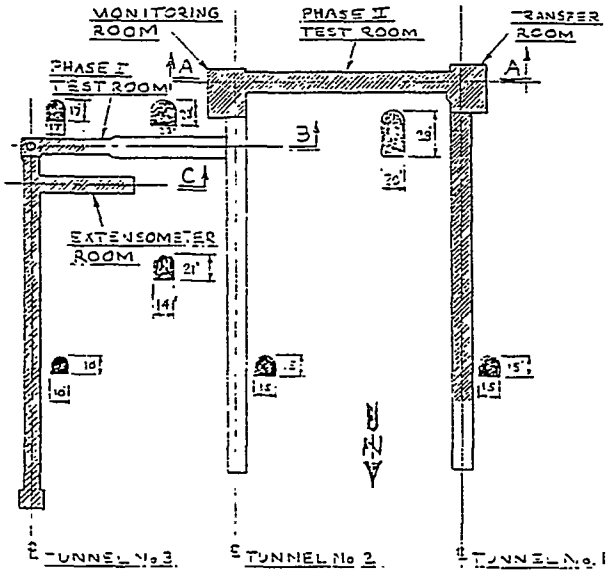
The NSTF is located in the entablature of the Pomona Member of the Saddle Mountains Basalt, approximately 45 m beneath the surface of Gable Mountain. The access tunnels penetrate the Elephant Mountain Member, the Rattlesnake Ridge interbed, and the entablature of the Pomona Member. The rock is highly jointed, with the predominant features being the vertically oriented polygonal cooling columns. The intact rock had a high compressive strength (>300 megapascals). Conditions were generally dry. The rock mass, although highly jointed, was classified as "good rock" using various engineering rock-mass classifications.



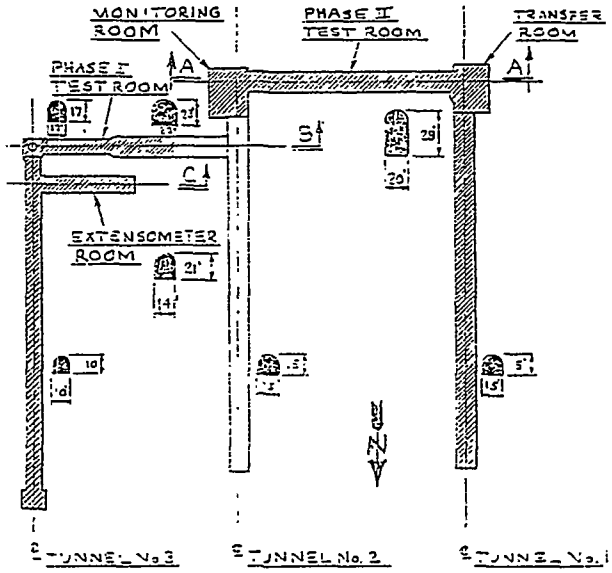
a) 8-1-88



b) 8-5-88



c) 8-10-88



d) 8-12-88

Figure A.8-3
Plan View of Sequence for Backfilling NSTF
(Dates shown are actual) (after DOE, 1989)

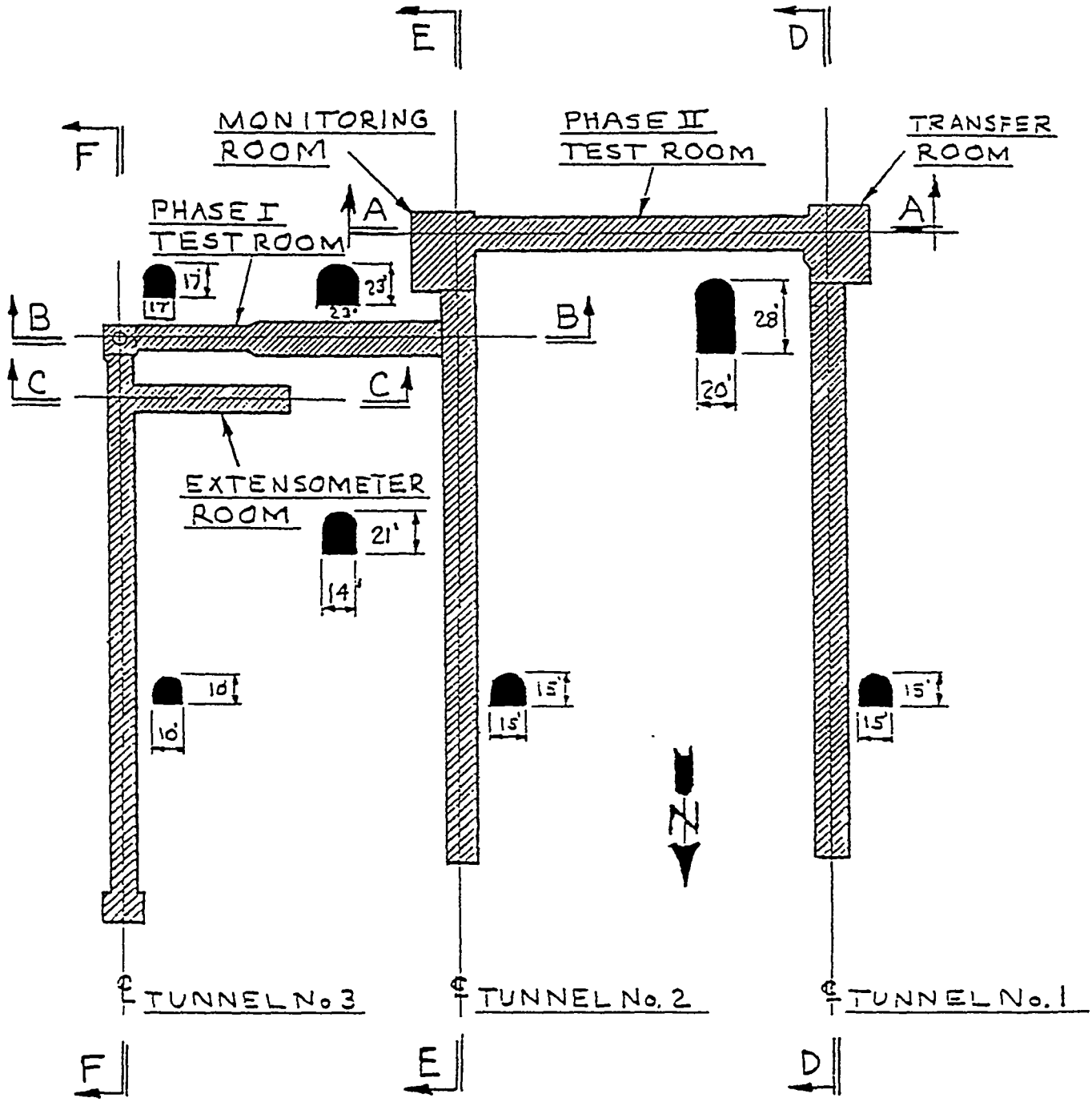


Figure A.8-4
As-Built Plan View of NSTF (8/22/88)
Showing Location of Cross Sections (DOE, 1989)

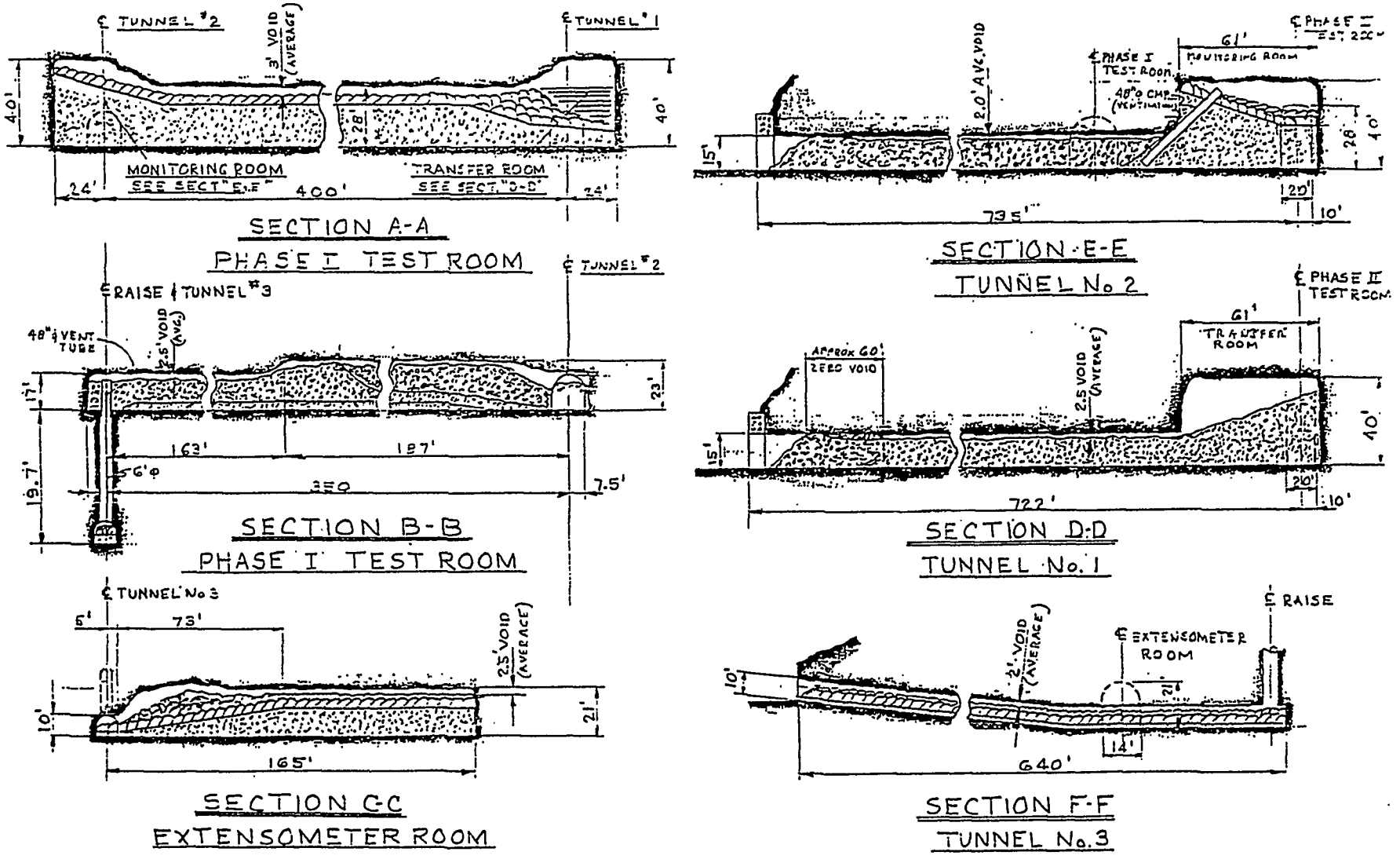


Figure A.8-5

As-Built Sections Through NSTF, Backfilling Complete

A.8.4.3 Problems Encountered

The openings were not originally designed with backfilling in mind, and there were a few locations where achieving a high fill ratio was difficult because of large differences in the height of the back in adjoining rooms. Several of these areas are evident in Figure A.8-5. Because tightfilling was not required, this was not a major problem. Table A.8-1 shows the average height of the void remaining in each of the rooms. Fill ratios were generally maintained at better than 80%, except in the Transfer Room where the high back created a problem. Analysis showed that even the lowest fill ratios were adequate to prevent discernible surface subsidence. Few other major problems were encountered, as evidenced by the rapid progress of the filling.

A.8.4.4 Quality Control

As-built drawings of the backfilling sequence were prepared. The average void space above the fill in each area of the NSTF was carefully monitored. In general, the fill height equaled or exceeded that called for in the reclamation plan. Because there were no other requirements for fill performance, quality control requirements were minimized.

A.8.5 Discussion

Backfilling at the Yucca Mountain repository will be basically similar to that performed in this case study. However, because the performance criteria for the backfill are expected to be much more stringent, improvements will be required in the areas of fill preparation, emplacement techniques, and quality control. Specifically, it is likely that an average void of less than 2.5 ft above the fill will be specified. The fill will probably have specific permeability and compaction specifications.

More complete filling could have been achieved in the NSTF if required. Gravity emplacement of fill into the high rooms was considered, but was determined to be unnecessary to prevent subsidence. Alternately, fill could have been pneumatically emplaced via angled boreholes drilled from adjacent underground openings.

If future backfilling can be considered in the design of an underground facility, measures can be taken to facilitate the process. Where possible, an even back without abrupt brows is preferred.

Table A.8-1
Backfill Volumes—NSTF Tunnels (DOE, 1989)

Tunnel or Room	Calculated Volume (yd^3)	Est. Qty.(1) of Material to be Placed (yd^3)	Calculated Qty. (2) of Material Actually Placed (yd^3)	Percent Filled	Avg. Void (ft)
Tunnel No. 1	6,600	5,366	6,021	90.4	2.5
Tunnel No. 2	7,094	5,788	6,412	90.4	2.5
Tunnel No. 3	2,703	1,811	2,365	87.5	2.0
Monitoring Room	3,022	2,228	2,487	82.3	n/a
Transfer Room	2,889	1,431	1,511	52.3	n/a
Phase I Test Room	6,769	5,560	5,984	88.4	2.5
Phase II Test Room	8,533	7,395	7,678	90.0	3.0
Extensometer Room	2,316	1,712	1,864	80.5	2.5
	39,986	31,291	(3) 34,322	85.8	

Notes:

(1) The estimated quantity of material to be placed as tunnel backfill was based on an average void space of 4.0 ft from top of fill to crown of tunnel.

(2) Calculated quantities are based upon actual average void space after backfill was placed.

(3) The surveyed quantity of material excavated from the bench was 30,900 (bank) cubic yards. The level of compaction achieved in the tunnels is calculated to be:

$$\left[1 - \frac{34{,}322 - 30{,}900}{30{,}000}\right] \times 100 = 89\%$$

A.8.6 References

DOE (1978). "Environmental Assessment, National Waste Terminal Storage Program, Near Surface Test Facility, Hanford Reservation, Washington," DOE/EA-0052, U.S. Department of Energy, Washington, D.C., December. (SRX.821006.0051)

DOE (1988). "Basalt Waste Isolation Project, Near Surface Test Facility, Reclamation Plan," U.S. Department of Energy, March 23, Rev. 2. (HQX.880325.0050)

DOE (1989). "Basalt Waste Isolation Project, Reclamation Report," proposed by Morrison Knudson Co. for the U.S. Department of Energy, January. (NNA.940629.0022)

APPENDIX A.9

USBM Lake Lynn Laboratory
Fairchance, Pennsylvania

Site Visit: November 5, 1993

Purpose: To demonstrate state-of-the-art in pneumatic backfilling for abandoned mine openings. Crushed limestone was emplaced in a simulated underground coal mine using innovative equipment with the potential to improve the maintenance problems associated with pneumatic stowing equipment. Interesting prototype equipment is being developed for remote monitoring of progress of backfilling using a laser rangefinder and TV camera.

Table of Contents

A.9.1 Introduction A.9-1

A.9.2 Pneumatic Stowing Equipment Innovations A.9-4

A.9.3 Previous Research at SAIL A.9-5

A.9.4 Status of Current Research A.9-9

A.9.5 Relevance to Yucca Mountain Project A.9-10

A.9.6 Research Products A.9-10

APPENDIX A.9
USBM Lake Lynn Laboratory
Fairchance, Pennsylvania

A.9.1 Introduction

The Pittsburgh Research Center of the U.S. Bureau of Mines (USBM) has been conducting research in pneumatic stowing for a number of years as part of the Abandoned Mine Lands Reclamation Research Project. They have recently reported several innovations with the potential to improve the application of pneumatic stowing to sealing underground mine openings. Additionally, they have several projects involving development of explosion-proof seals for coal mining applications.

The Lake Lynn Laboratory is a multipurpose facility located in a remote location approximately 70 miles southeast of Pittsburgh, Pennsylvania, at the site of an abandoned limestone mine and quarry (Figure A.9-1). The federal government leases the 80-acre site from the Martin Marietta Corporation. The site has both surface and underground test facilities (Figure A.9-2). Surface facilities of interest to the Yucca Mountain Project include a full-scale Subsidence Abatement Investigation Laboratory (SAIL) where pneumatic stowing research is conducted, while underground facilities include a system of drifts for testing explosion-proof seals for coal mine use and a grouted vertical pipe from the surface for remote construction of pumpable concrete seals.

The SAIL (Figure A.9-3) consists of a three-story, 30- by 30-ft steel tower with two elevated working platforms and a simulated rectangular mine entry. The tower is used to support a vertical borehole, simulating the arrangement for blind backfilling of underground mine openings to prevent subsidence. A bridge provides access to the tower and supports a 24-in. conveyor belt, a 6-in. conveying pipeline, and utility conduits. Bulk granular material can be delivered to the top of the borehole via either pneumatic pipeline or conveyor. The simulated mine opening is located beneath the bottom of the borehole and is constructed of interlocking blocks with a corrugated metal roof. The blocks allow flexibility in opening dimensions, while the roof can be readily removed to evaluate the quality of the filling and to take samples.

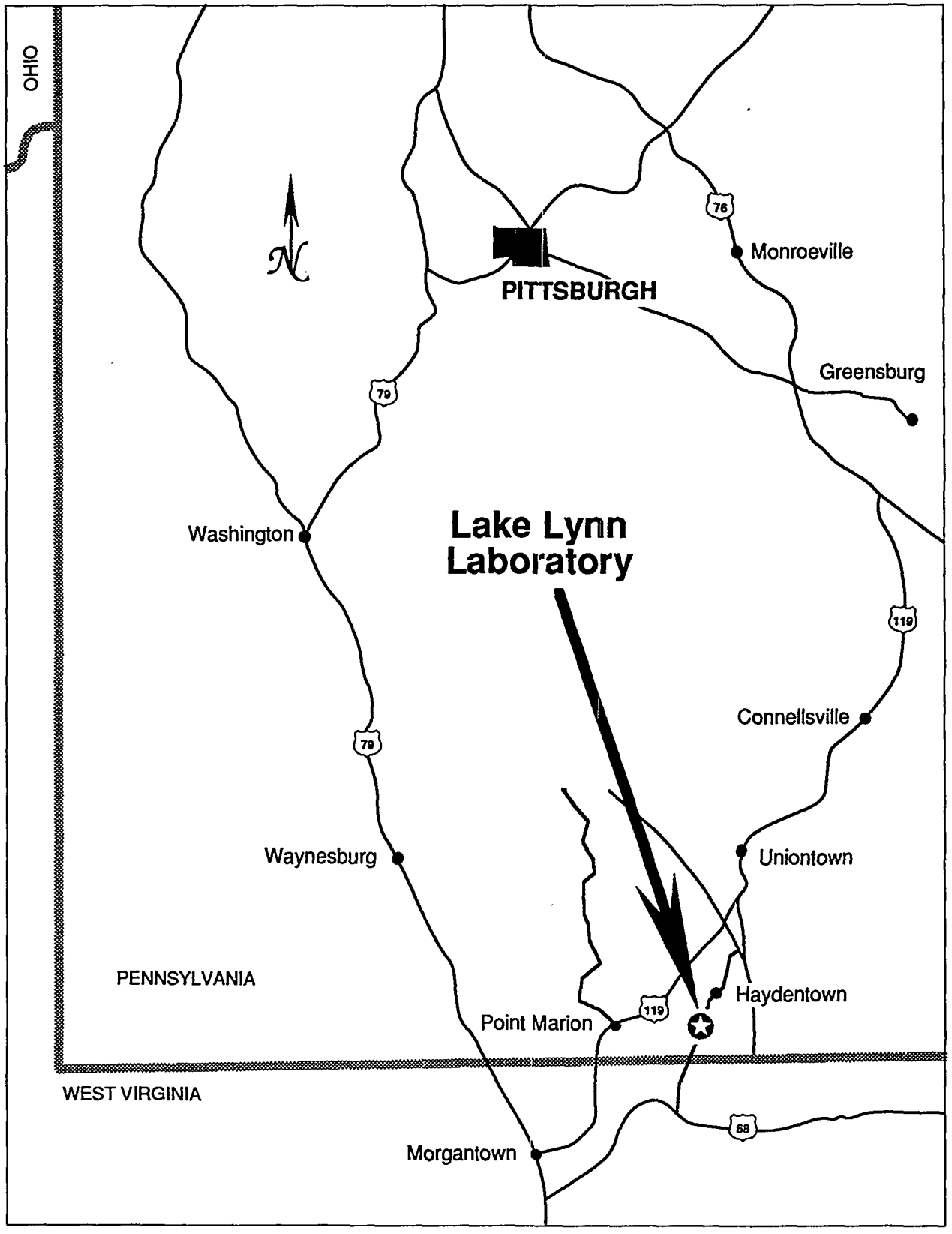


Figure A.9-1
Location of Lake Lynn Laboratory

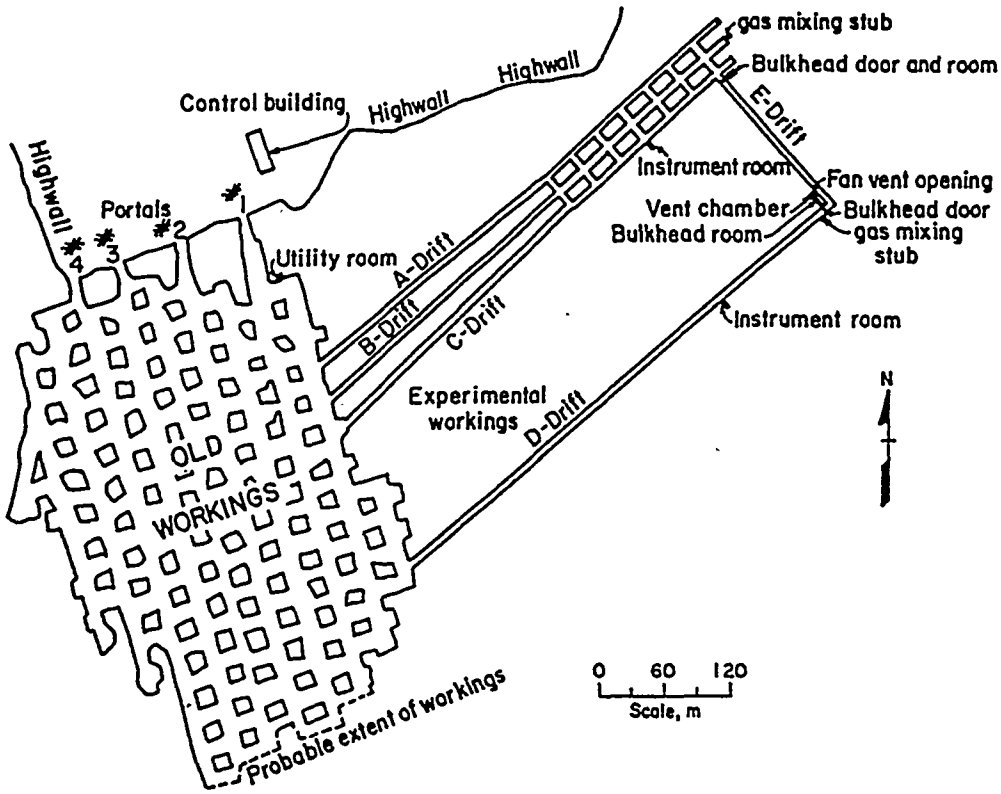


Figure A.9-2a
Plan View of Underground Mine Workings

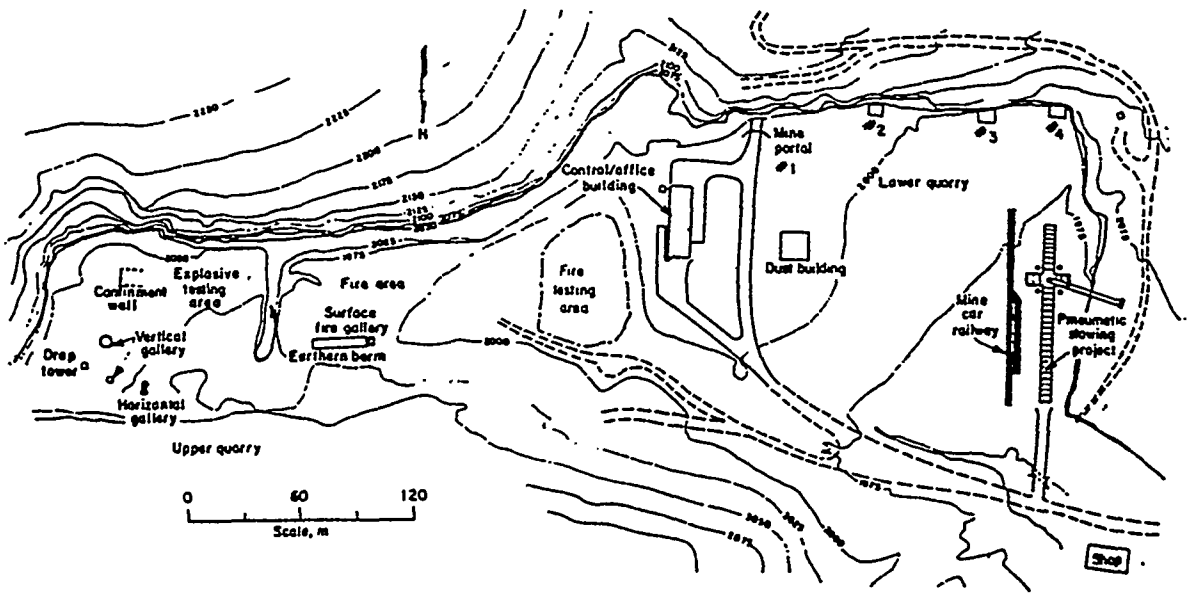


Figure A.9-2b
Plan View of Surface Research Areas

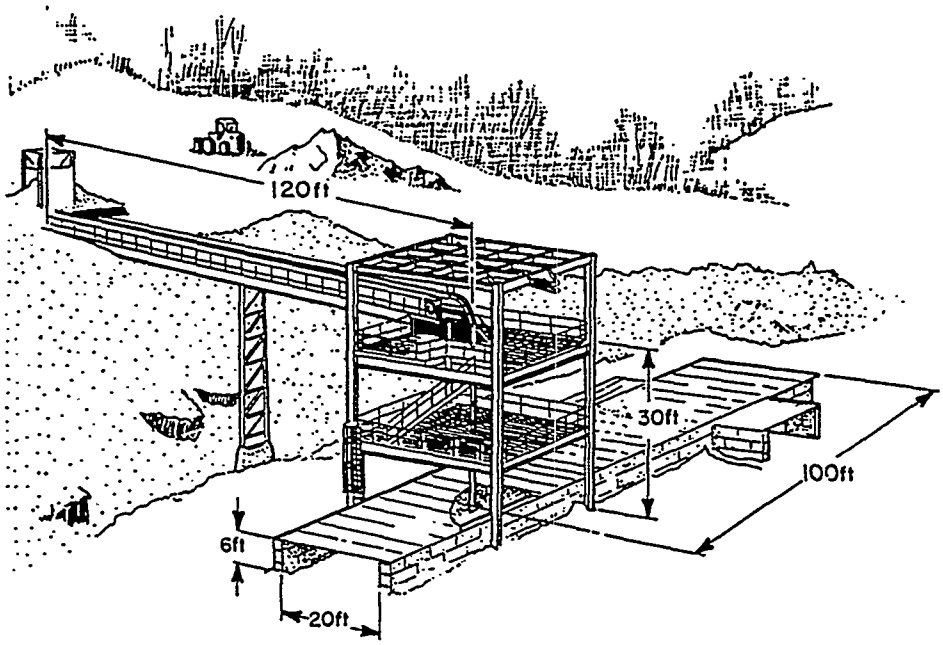


Figure A.9-3
Subsidence Abatement Investigation Laboratory (SAIL)

A.9.2 Pneumatic Stowing Equipment Innovations

Because the research emphasis at Lake Lynn is on blind backfilling of abandoned mine openings, equipment innovations were sought to enhance the utility of conventional pneumatic stowing equipment for this purpose. To maximize the amount of drift that can be filled from each borehole, high exit velocities are required—especially as the stowing trajectory is constrained by the mine roof. The biggest problem with blind backfilling is converting the downward momentum of fill in the downpipe into a horizontal velocity suitable for effective stowing. The massive wear-resistant elbows used in mining applications do not fit well down boreholes, so the USBM has developed a number of devices for this purpose, including simple articulated mechanical devices that, although subject to wear, are able to place fill in over 200 linear ft of limited height mine opening from a single borehole. Development is currently concentrated on a high-efficiency pneumatic ejector to eliminate the wear problem; this device uses a supersonic jet of air to accelerate the fill horizontally. Several prototypes have been developed and successfully tested. An improved nozzle with a laser rangefinder to evaluate the backfilling quality without underground access is currently under development.

Abrasion is always a concern with pneumatic stowing applications, and the rotary air-lock feeder (RALF) has been a high-maintenance item. The USBM has developed two simple low-maintenance pipefeeders to replace the RALF for mine sealing applications. The simplest is the pneumatic pipefeeder (Figures A.9-4 and A.9-5), which has no moving parts. A supersonic jet of air is used to accelerate the feed to pipeline velocity. This inexpensive pipefeeder has been demonstrated to inject stowing material up to 1.5 in. topsize into a 6-in.-diameter pipeline at rates up to 45 tons per hour (TPH) over a short distance (approximately 100 ft) and 25 TPH over a lateral distance of several hundred feet. The second device is the pneumatic screw-pipefeeder (Figure A.9-6), which has the ability to seal higher back pressures and can therefore drive longer sections of pipe. A conservative estimate of the achievable flow rate for the system tested is 18 TPH over a distance of 500 ft. A pneumatic booster has been tested and can be used to extend the pipeline length. The screw-pipefeeder also permits accurate metering of fill for cement fill applications. It is believed that these devices can deliver fill at up to 1,000 ft, although sufficient pipe was not available to demonstrate this during the brief contract period.

A.9.3 Previous Research at SAIL

The objectives of the research conducted at SAIL to date have been to design and evaluate the performance of methods for mitigating abandoned coal mine subsidence. Two in-house and six contract research projects have been completed at SAIL since 1988, and results were presented as open-file reports, published in the USBM's series, or presented in technical journals. Below are brief descriptions of the projects completed at SAIL.

Contract No. J0388011 involved development of a pneumatic pipefeeder for subsidence abatement. The goal of this project was to design an inexpensive, low-wear system capable of towing backfill material of up to 1.5-in. topsize through a 6-in.-diameter pipeline at rates up to 45 TPH. Work on this project consisted of designing the system and testing the prototype at the SAIL.

Contract No. J0388012 involved design of two different collapsible elbow/nozzle assemblies to place fill in over 300 linear ft of a mine opening from a single borehole. Work on this project consisted of designing the two assemblies and performing initial tests at the SAIL.

Contract No. J0388015 involved evaluation of an existing pneumatic stowing system known as the Remote Air-Jet Pneumatic Stowing System. The Air-Jet stowing system had been

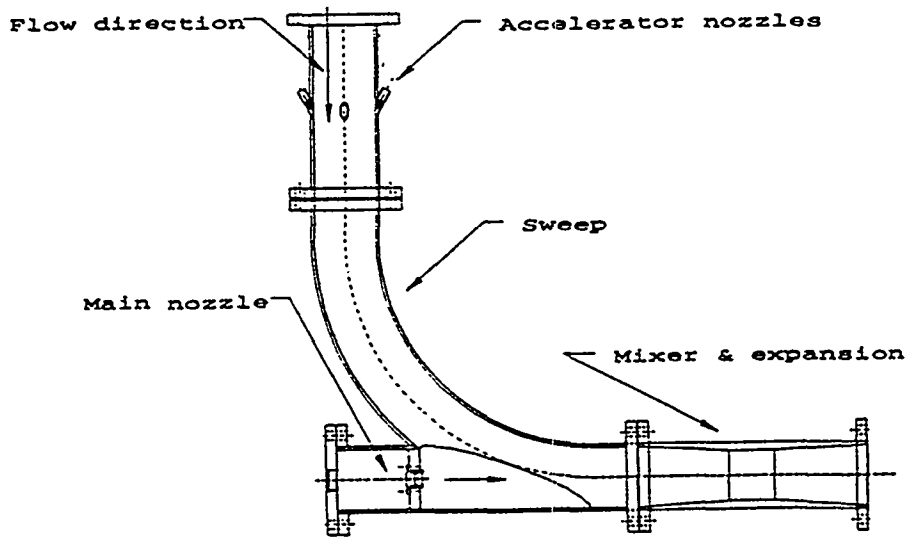


Figure A.9-4
Pneumatic Pipefeeder Assembly

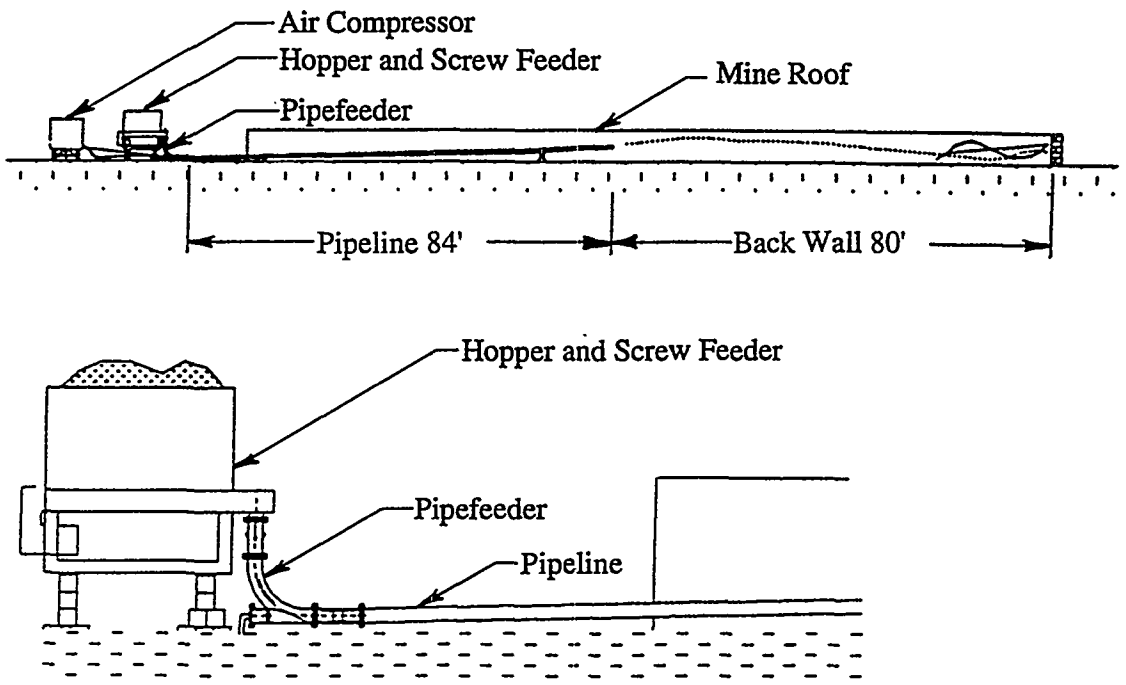


Figure A.9-5
Pipefeeder Test Set-up

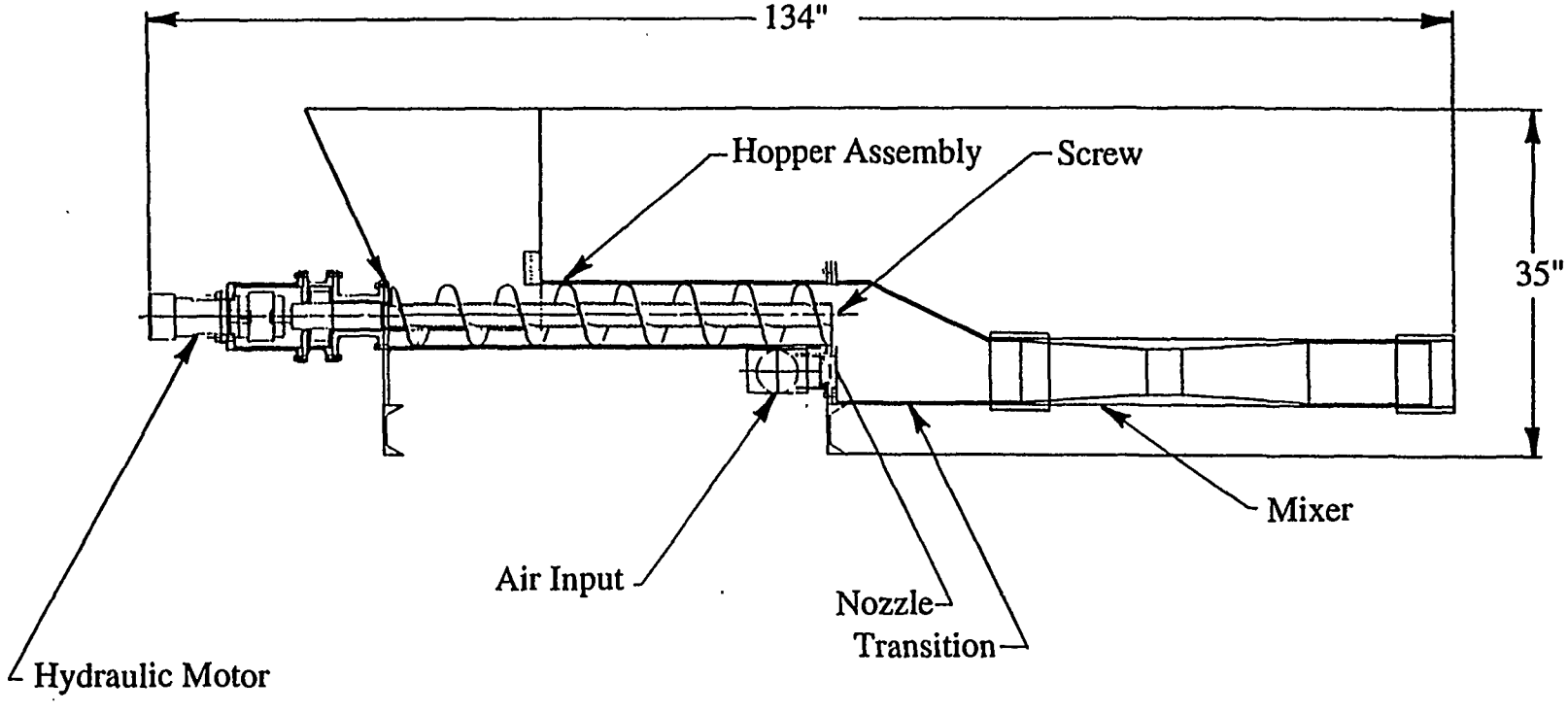


Figure A.9-6
Screw Pipefeeder Assembly

used successfully in previous backfilling efforts by the contractor, and was directed toward improving the performance of the system. Work consisted of designing modifications to the equipment and testing the system at the SAIL.

Work under Contract No. J0309012 included developing a system that eliminated specialized material feeder systems and to improve material discharge performance at the exit nozzle for pneumatic backfill stowing through a borehole. The High-Efficiency Ejector, developed under this contract, eliminated the abrasion problems associated with earlier borehole pneumatic stowing systems.

The in-house project titled "Development of a Remote-Control Elbow" was initiated to improve the performance of remote pneumatic stowing by providing a means to determine the orientation of a pneumatic elbow/nozzle assembly in an abandoned mine opening, to observe from the nozzle location the placement of backfill material, and to provide the capability of remotely adjusting the trajectory of the material discharged from the nozzle. The nozzle assembly, using a supersonic air-jet, was fabricated and tested at the SAIL. The instrumentation assembly housing was also designed and fabricated; a laser rangefinder, compass, video camera with high-intensity lamps, and water sensor were installed into the housing, and field trials of the assembly were conducted at the SAIL. A Report of Investigation summarizing the work performed under this project is currently being prepared.

Contract J0319005 developed a pneumatic concrete placement device for shaft and adit closure. The primary objectives of this work were to design, fabricate, and evaluate a screw-fed pipefeeder and to mix cement with wet gravel at the delivery end of a pipeline to construct shaft and adit seals. The system was developed and tested, and two adit seals were successfully constructed at the SAIL.

The in-house project titled "Fabrication and Performance Evaluation of Grouted Point-Support Columns in Flooded Environments" evaluated construction processes and methodologies to determine their effect on overall support characteristics of grout columns fabricated using sodium silicate technology in flooded abandoned underground mines. The primary objectives of this research effort were to determine optimal grout/sodium-silicate ratios, optimal grout/silicate mixing strategies, and effects of internal calcium silicate deposition to obtain the highest possible column strength and most efficient column fabrication methodologies. Columns with varying grout-sodium-silicate mixtures were constructed at the SAIL, and their engineering properties were established using in-house

laboratories. A Report of Investigation summarizing the work performed under this project is being prepared.

A.9.4 Status of Current Research

Work in fiscal year (FY) 1993 at the SAIL consists of establishing one new contract and one new in-house project, as well as continuing work on a contract initiated in FY92. The following tasks are being accomplished in FY93.

Contract J0329004 was initiated in late FY92 to develop two devices: First, an economical, low-wear, co-rotating, twin-screw, pneumatic pipefeeder will be designed and constructed; this pipefeeder will provide an alternative to current air-lock feeders for blowing backfill material long distances into a mine opening. Second, a novel mechanical device to construct point support columns will be designed and constructed. This device will allow low-slump concrete to be placed in an annular ring in a mine opening to build a cylindrical wall from floor to roof remotely through a borehole. The void in the center of this cylinder will be pneumatically filled with dry material.

Contract H0339008 has been established to improve the concrete placement device designed in Contract J0329004. This work involves improving the system that places low-slump concrete in an annular ring in a mine opening to build a cylindrical wall from the mine floor to roof remotely through a borehole. The improvement consists of a system that introduces fiber-reinforcement into the concrete mix, thereby increasing the overall strength of the cylindrical wall.

The in-house project titled "Development of an Improved Pneumatic Backfill System Using Laser-Radar Technology" was initiated to improve the concept of monitoring the progression of backfill placement through a borehole. The previous system developed by the USBM and tested at the SAIL used a video camera and a laser-rangefinder, to quantitatively and qualitatively monitor backfill progression from the borehole location. This new system uses a laser-radar imaging device that quickly generates both two- and three-dimensional maps of the backfill pile. This system eliminates the need for the laser-rangefinder, and the video camera and associated lighting system. By using the laser-radar system, the instrumentation housing attached to the pneumatic backfilling device will be smaller and will provide a much more accurate assessment of the backfilling process than was possible with the previous generation device.

A.9.5 Relevance to Yucca Mountain Project

Most of the work on explosion-proof seals is not directly applicable to Yucca Mountain. Also, it is unlikely that blind backfilling of the type considered for abandoned underground coal mines will be applied to Yucca Mountain. However, several aspects of this research are of potential interest to the Yucca Mountain Project.

- The pneumatic pipefeeder technology is interesting and bears further investigation. The RALF is one aspect of pneumatic stowing that mining industry users have been dissatisfied with. The concept of the pneumatic booster to increase horizontal transport capabilities also merits attention.

- A relatively simple laboratory, similar to SAIL, could be constructed at another location that would allow emplacement and testing of tuff backfills. A simulated circular opening could be fabricated from a concrete culvert with a removable crown. Methods for placing backfill could be evaluated and the placed fill sampled and tested to determine mechanical and thermal properties. A similar proposal of this nature has already been presented in Fernandez et al. (1993).

- Work on the remote control nozzle may be of interest if some portions of the Yucca Mountain backfilling must be done remotely, with quality control provided by laser-rangefinder, video cameras, and similar remote sensing devices.

A.9.6 Research Products

<u>FY89</u>

Walker, J. S. and M. A. Trevits. "Overview of the Bureau of Mines Program for the Evaluation of Remote Pneumatic Stowing Technology," Presented at 11th Association of Abandoned Mine Land Programs, Williamsburg, VA, October 16-19, 1989.

<u>FY90</u>

Burnett, M. "Development of an Inexpensive Pneumatic Pipefeeder," Contract Final Report, Contract No. J0388011, Burnett Engineering, April 1990.

Masullo, A. L. "Above-Ground Demonstration of Pneumatic Stowing Technologies," Contract Final Report, Contract No. J0388012, ICF/SRW Associates, June 1990.

Hanson, L. C. "Design and Evaluation of a Remote Air-Jet Pneumatic Stowing System," Contract Final Report, Contract No. J0388015, L. C. Hanson Company, June 1990.

Walker, J. S. and M. A. Trevits. "New Developments in Pneumatic Stowing for Subsidence Abatement in Abandoned Mine Lands," Presented at the 33rd Annual Meeting of the Association of Engineering Geologists, Subsidence Symposium, Pittsburgh, PA, October 1990.

Walker, J. S. and M. A. Trevits. "New Developments in Pneumatic Stowing for Subsidence Abatement in Abandoned Mine Lands," Presented at the 12th Association of Abandoned Mine Land Programs Conference, Breckenridge, CO, September 16-20, 1990.

FY91

Abandoned Mine Lands Program TN #1, Subsidence Abatement Investigation Laboratory (SAIL). Technology News, No. 380, Bureau of Mines, April 1991.

Walker, J. S. and E. S. Weiss. "Remote Pumpable Mine Fire Seals," Presented at the 13th Association of Abandoned Mine Land Programs Conference, Lake Ozark, MO, October 7-10, 1991.

Burnett, M. "Development of a High Efficiency Ejector System," Final Contract Report, Contract No. J0309012, Burnett Engineering, October 1991.

FY92

"Pneumatic Concrete Placement for Shaft and Adit Closures," Final Contract Report, Contract No. J0319005, Burnett Engineering, May 1992.

FY93

Dyni, R. C. "An Instrumented Pneumatic Backfilling System," BuMines Report of Investigation, in review.

Dyni, R. C. and M. Burnett. "A Field Demonstration of Two Pneumatic Backfilling Devices," American Society of Civil Engineers, Civil Engineering Magazine, New York, NY, in review.

Dyni, R. C., M. Burnett, D. Philbin and F. McCall. "Field Demonstration of Pneumatic Backfilling Technologies," BuMines Report of Investigation, in review.

Dyni, R. C. "Fabrication and Performance Evaluation of Grouted Point-Support Columns in Flooded Environments," BuMines Report of Investigation, in review.

Dyni, R. C. "Development of an Instrumented Pneumatic Backfilling Device," to be presented at Conference on Mine Subsidence in Urban and Developed Areas, Wyoming Dept. of Environmental Quality, Rock Springs, WY, September 1993.

Burnett, M. and R. C. Dyni. "Field Demonstration of Two New Pneumatic Backfilling Systems," to be presented at Conference on Mine Subsidence in Urban and Developed Areas, Wyoming Dept. of Environmental Quality, Rock Springs, WY, September 1993.

APPENDIX B.1

Helms Pumped Storage

Case History from Literature

Purpose: To demonstrate the effective use of microfine cement to seal an isolated shear zone subject to high water pressure.

Table of Contents

B.1.1 Introduction B.1-1

B.1.1.1 Description B.1-1
B.1.1.2 Objective of Sealing/Backfilling Operation B.1-1
B.1.1.3 Purpose B.1-1

B.1.2 Materials B.1-5

B.1.2.1 Description B.1-5
B.1.2.2 Properties B.1-6

B.1.3 Equipment B.1-6

B.1.4 Seal/Backfill Placement Operations B.1-7

B.1.4.1 Sequence of Operations B.1-7
B.1.4.2 Conditions of Emplacement B.1-8
B.1.4.3 Problems Encountered B.1-8
B.1.4.4 Quality Control B.1-8

B.1.5 Discussion B.1-9

B.1.6 References B.1-11

APPENDIX B.1
Helms Pumped Storage

B.1.1 Introduction

B.1.1.1 Description

The Helms Pumped Storage Project (see Moller et al., 1983) of Pacific Gas & Electric Company is located in the Sierra Nevada Mountains of southern California, 50 miles northeast of Fresno (Figure B.1-1). Using the difference in elevation between two large reservoirs, the project pumps water uphill during off-peak electricity demand periods for later use to generate up to 1,125 MW of electricity during peak demand periods.

The underground workings of the project include approximately 20,000 ft of 27-ft-diameter pressure tunnel; 3,700 ft of access tunnel; 1,600 ft of high-pressure steel-lined tunnel; two large underground chambers; three deep vertical shafts; and one inclined shaft which are shown in profile in Figure B.1-2. The project is situated in granitic formations with tight, cubic jointing. Major geologic features are infrequent. However, during construction an undetected shear zone was discovered (Figure B.1-3), which caused both a high-pressure penstock and a dry tailrace access tunnel. There were concerns that this feature would act as a conduit for water.

B.1.1.2 Objective of Sealing/Backfilling Operation

The objective of the sealing operation was to prevent water from leaking out of the pressure tunnel into the dry access tunnel. An additional concern was to minimize pore pressure buildup in joints and possible hydraulic jacking of existing blocks of rock into the underground complex. Specific objectives were:

- to reduce groundwater seepage to the extent practical, and
- to limit pore pressure in the rock to 500 pounds per square inch (psi) downstream from the plug.

B.1.1.3 Purpose

This case history demonstrates the effective use of grout, especially microfine cement, to seal an isolated shear zone. This case history represents the first large-scale application of microfine cement for grouting in the United States.

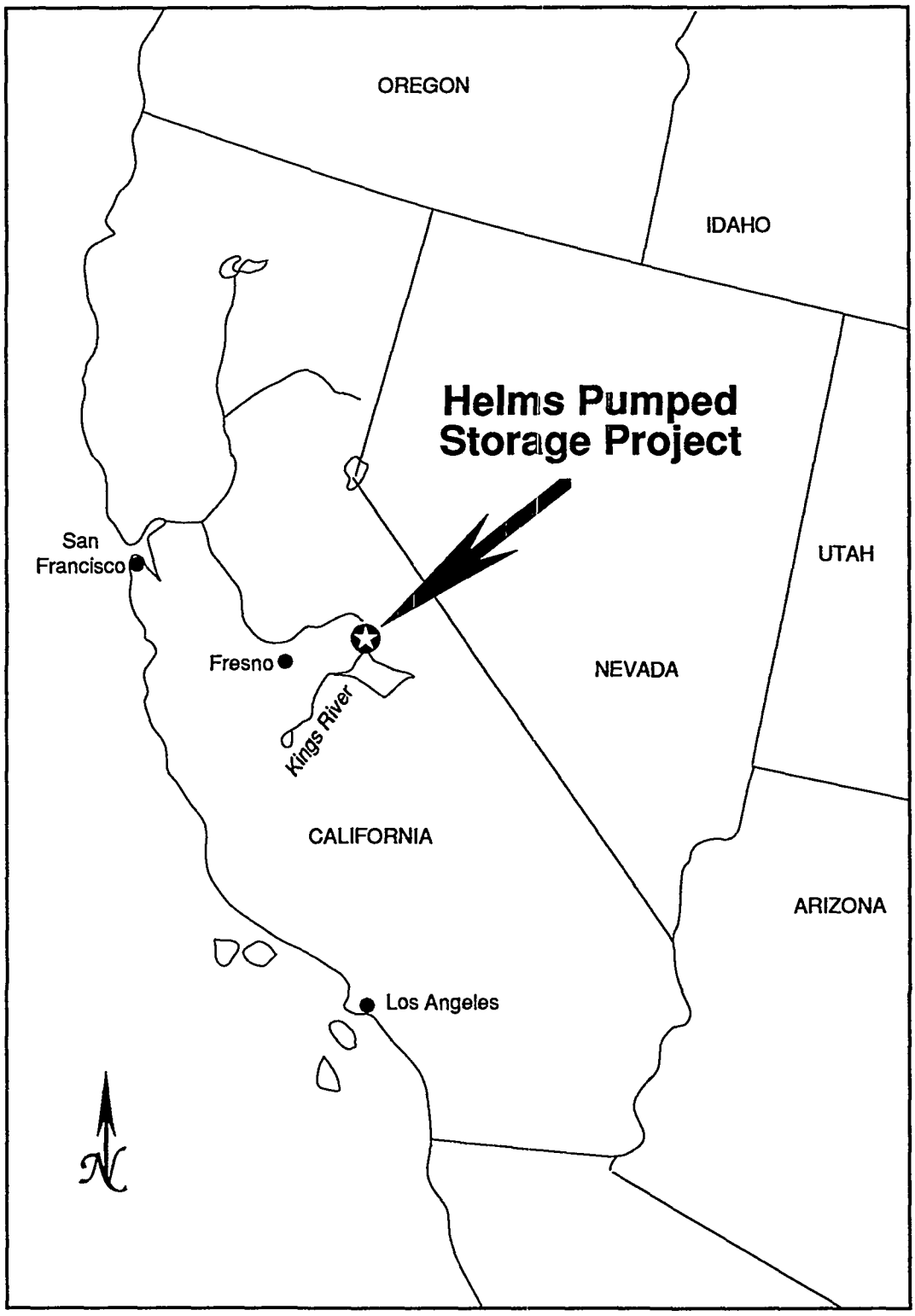


Figure B.1-1
Location of Project

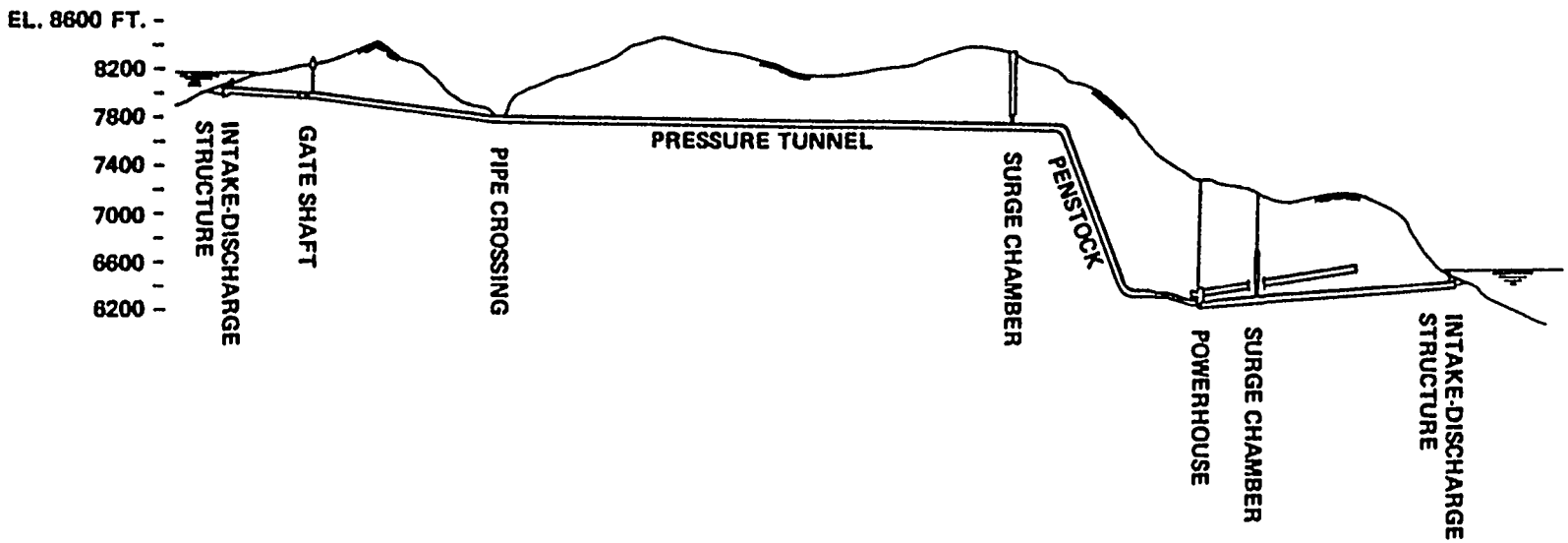


Figure B.1-2
Profile of Underground Workings

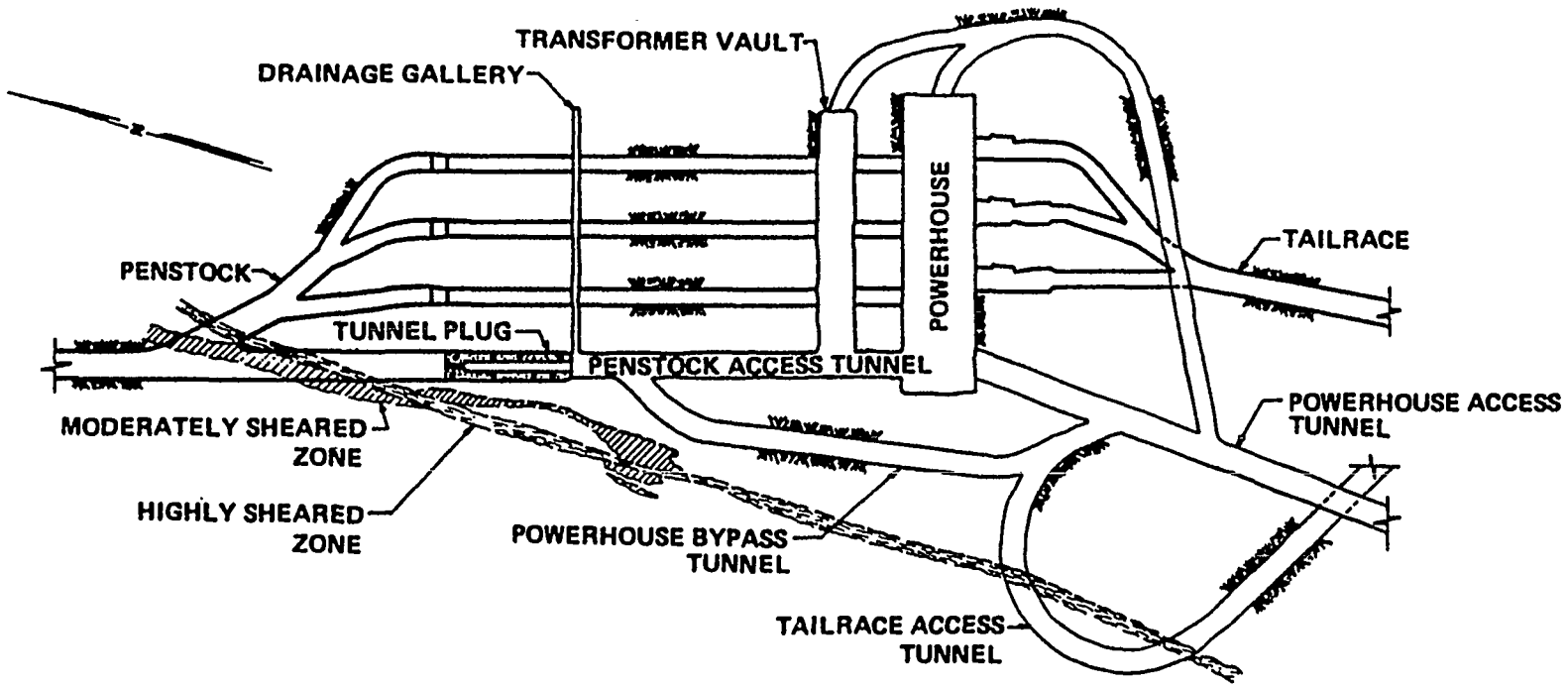


Figure B.1-3
Plan View of Powerhouse Complex and Shear Zone

B.1.2 Materials

B.1.2.1 Description

Grouting specifications called for materials having low viscosity, high permeability, controllable set time, low toxicity, high bonding strength to rock, resistance to degradation, and high compressive strength.

Portland-based materials were selected, and some areas were grouted using a conventional Type II portland cement. Where lower viscosity and superior penetrating capability was required, ultrafine cement grouts were selected. This was the first major application of ultrafine cement grouts in the United States, and availability of a suitably fine cement (Blaine fineness of 8,000 cm^2/g) was limited. For deeper, higher pressure grout stages, an ultrafine cement (ALOFIX MC) was imported from Japan. Its chemical composition is shown in Table B.1-1. Other stages were grouted using a domestically produced, reground, Type III portland cement.

Table B.1-1
Chemical Composition of Ultrafine ALOFIX MC Cement

Ignition Loss	0.30%
SiO_2	29.00%
Al_2O_3	13.20%
Fe_2O_3	1.20%
CaO	49.25%
MgO	5.60%
SO_3	1.20%
TOTAL	99.70%

B.1.2.2 Properties

Table B.1-2 shows a comparison of the properties of the ultrafine and reground Type III cements. It is evident that the ultrafine cement has a finer 50% grain size, a lower viscosity, and a higher Blaine fineness. Although information was not available for ultrafine ALOFIX MC cement, Figure B.1-4 shows the size gradation curve for a very similar microfine cement (MC-500).

B.1.3 Equipment

Conventional grouting equipment was used. There were some space limitations, as all microfine grouting was accessed via a 32-in.-diameter hole. Grout holes were drilled using rotary percussion, jack-leg drills with 2-in.-diameter bits. Drilling air and water were piped from the surface approximately 0.75 mile to the underground site.

Grout was mixed in three, air-powered 75-g colloidal shear mixers. Four, multistage air-powered grout pumps with a combined capacity of 32 gpm at 1,000 psi were used to supply grout to the holes. Inflatable packers isolated the zones to be grouted. All equipment used is available off the shelf.

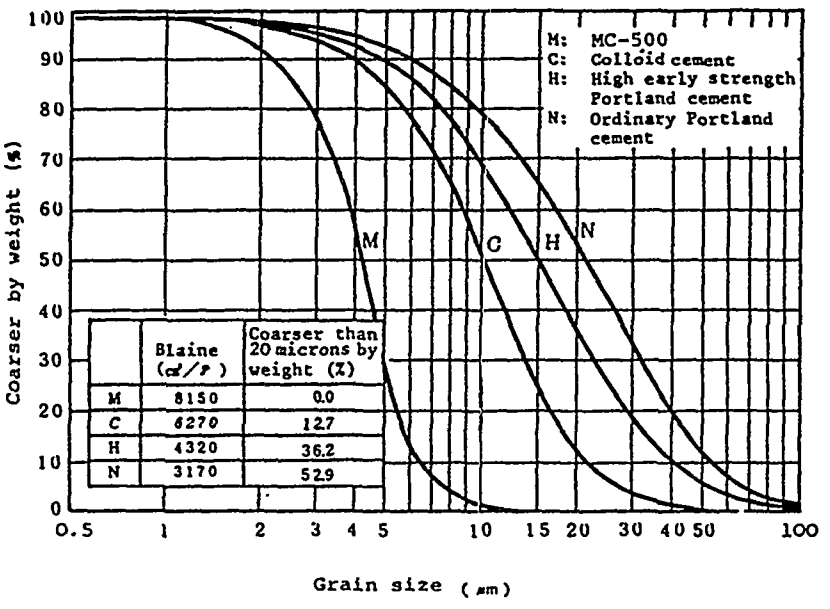


Figure B.1-4
Particle Size Gradation for Typical Microfine Grout Compared to Some Conventional Grouts

Table B.1-2
Comparison of Ultrafine and Reground Type III Cement

Characteristic	Ultrafine ALOFIX MC Cement	Reground Type III Portland Cement
Unit weight (bulk)	62.5 lb/ft^3	88 lb/ft^3
Blaine fineness	8,880 cm^2/gm	5,733 cm^2/gm
50% grain size	4 microns	9 microns
Weight per sack	2 kg = 44 lbs	88 lbs
Viscosity of 1:1 mix (by volume) at 15 min. after mixing (Marsh cone)	1.5 Cp	6.8 Cp
7-day strength of 1:1 mix (by volume)	2,000 psi	2,650 psi
Total cement used	110 tons	410 tons
Approximate volume of grout injected	194 yd^3	516 yd^3
Admixtures	Dispersant, 1% (by weight)	None
Cement suppliers	Onoda Cement Company Tokyo, Japan	Kaiser Cement Corporation Permanente, CA, USA

B.1.4 Seal/Backfill Placement Operations

B.1.4.1 Sequence of Operations

Prior to initial filling of the penstocks, the shear zone was grouted with a neat cement mix (portland Type II). Upon initial pressurization in the fall of 1982, seepage was noted. The seepage increased to 716 gpm; additionally, the pore pressure in the rock near the plug increased to 786 psi—about 96% of the pressure in the tunnel and greater than the minimum principal stress. After only 1 day of pressure treating, a failure occurred in another section of the tunnel, causing an 8-month delay. This time was used to implement a grouting program.

A detailed drilling and mapping program was performed to accurately delineate the shear zone. Two adjacent zones were discovered: a 15-ft-wide highly sheared zone and a

35-ft-wide moderately to slightly sheared zone. Both were near-vertical. In addition to crossing the penstock and tailrace access tunnel, they passed within 22 ft of the powerhouse bypass tunnel.

A grouted envelope was created around the high-pressure tunnel using microfine cement. The grout zone would be limited to the intersection between the shear zone and the pressure tunnels. Rings of eight, 2-in.-diameter holes were drilled on 10-ft centers, in stages to a depth of about 40 ft; Figure B.1-5 shows the drilling pattern.

The holes were grouted in stages, each stage involving a deeper hole and a higher pressure. This enabled use of high grout pressures without damaging the concrete lining or hydrofracturing the wall rock. The stages were: (1) contact grouting behind the lining at 100 psi, (2) 5 to 15 ft at 200 psi, (3) 15 to 25 ft at 350 psi, and (4) 25 to 40 ft at 700 psi. Active grout holes were drilled and grouted 180 ft apart. Contact grouting proceeded sequentially down the tunnel, while stage grouting used split spacings (40 ft, 20 ft, and then finally 10 ft center to center). This enabled monitoring of grout take.

It took 60 men a total of 30,000 man-hours, working three shifts per day, to place 110 tons of ultrafine grout and 410 tons of Type III grout.

B.1.4.2 Conditions of Emplacement

The only access to the length of tunnel to be grouted was a 21-ft-long crawl hole through a concrete pressure seal (tunnel plug) in the penstock access tunnel. This made access difficult and limited the size of the equipment used.

One hundred and ten tons of ultrafine cement and 410 tons of Type III portland cement were injected, for a total in-place volume of 710 yd^3.

B.1.4.3 Problems Encountered

No major problems were reported in the published literature.

B.1.4.4 Quality Control

Daily records were kept of pressure, mix, and grout take for each hole. Figure B.1-6 is a summary and explanation of the grout take (in sacks) for each hole and stage.

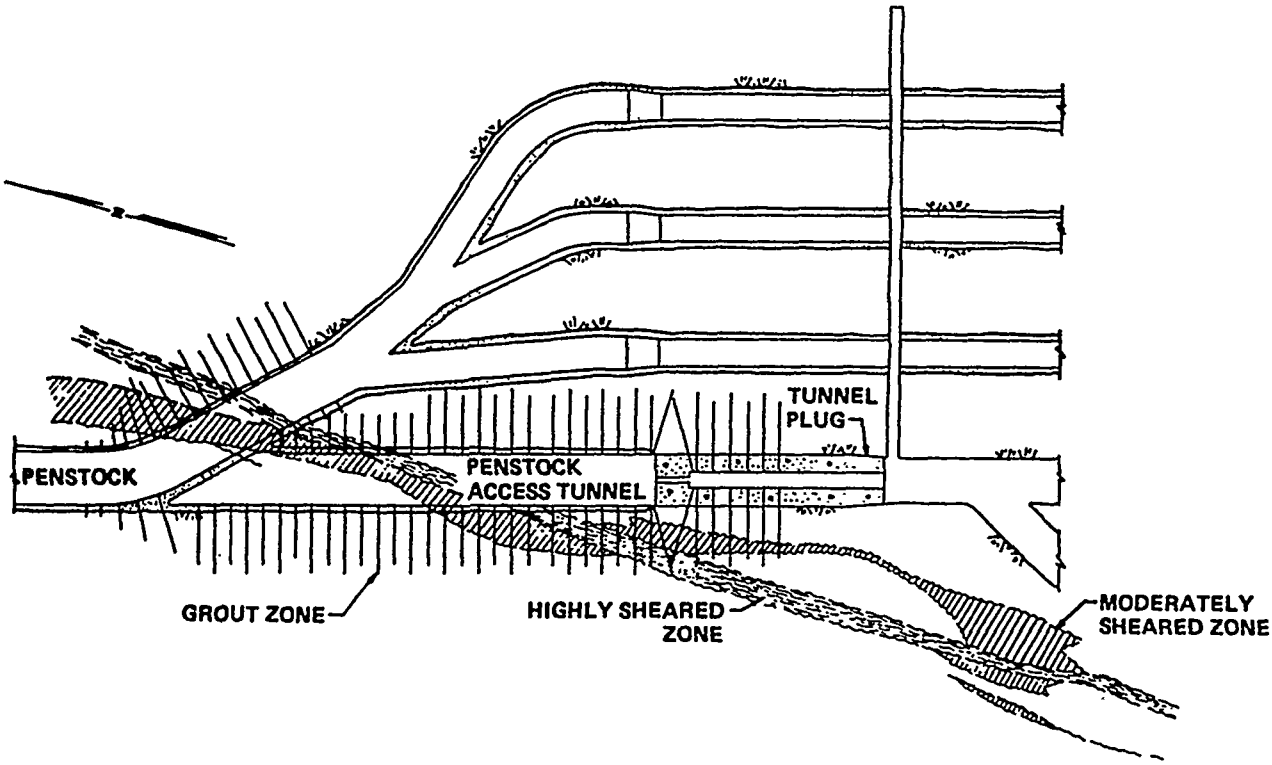


Figure B.1-5
Plan View of Grout Site Showing Pattern of Drilling

Additionally, the effectiveness of the grout barrier was tested by pressure testing additional holes with packer equipment in critical areas where grout takes were high. Several cores were taken at critical locations, and the cores were visually examined for grout penetration.

B.1.5 Discussion

This case history demonstrates the effective sealing of a shear zone using microfine cement. The effectiveness of the program can be summarized by these published observations, made when the tunnel was repressurized in August 1983.

It is possible that similar shear zones will be encountered at Yucca Mountain, although any water inflows would be considerably less. Some of them may require sealing. The case history discussed here involved placement of a large amount of grout; the effort did not completely seal the shear zone. However, sealing objectives were achieved.

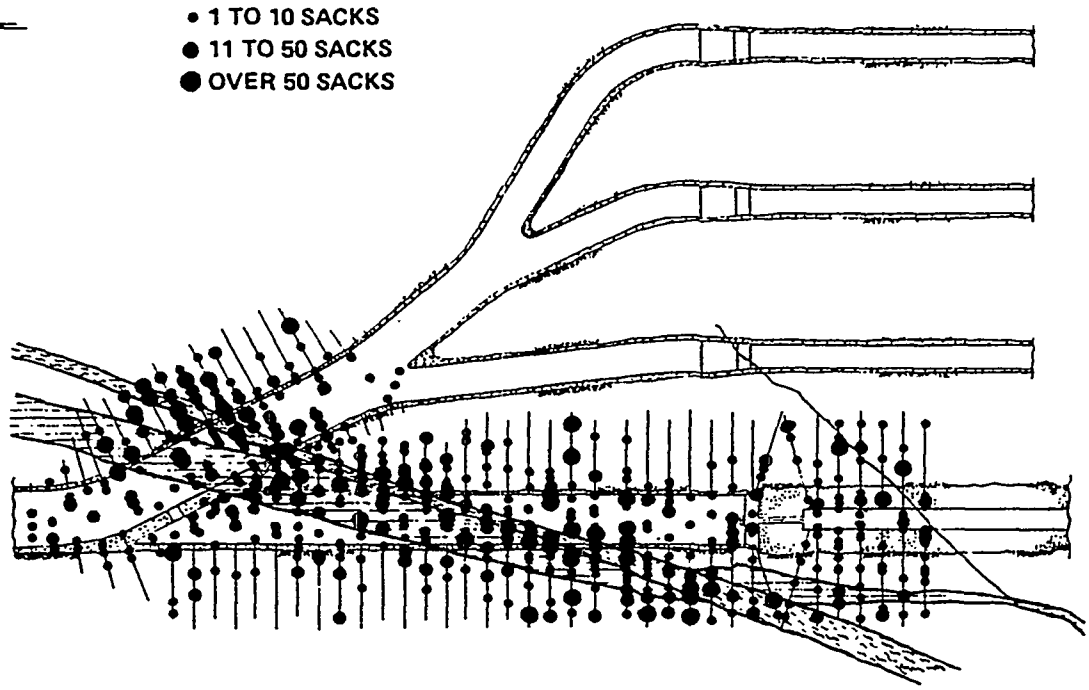


Figure B.1-6
Grout Take (in sacks)

The Helms project case history did show that ultrafine cement

- penetrates better than conventional portland cement grouts,
- bonds well to granitic rock and achieves strengths comparable to conventional grouts,
- works best in a 1:1 water/cement ratio, and
- was useful in meeting the sealing objectives of this project.

Table B.1-3
Summary of Grouting Program Results

Parameter	Before Grouting	After Grouting	Percentage Reduction
Lag-time of pore pressure (after tunnel pressure)	Several hours	Several days	---
Pore maximum pressure (in shear zone)	N/A	N/A	20%
Pore maximum pressure (at depth in rock mass)	786 psi	338 psi	57%
Pore maximum pressure (wall rock)	N/A	88 psi	N/A
Maximum seepage	716 gpm	430	40%

B.1.6 References

Moller, D. W., H. L. Minch, and J. P. Welsh, 1983. "Ultrafine Cement Pressure Grouting to Control Ground Water in Fractured Granite Rock," presented at the American Concrete Institute, Kansas City, September 29th. (MOL.19940719.0019)

APPENDIX B.2

New Waddell Dam

Site Visit: August 24, 1993

Purpose: To demonstrate the very effective use of cementitious grouts to seal a large dam foundation in fractured volcanic rock, representing the state-of-the-art in large-scale cement grouting in fractured rock.

Table of Contents

B.2.1 Introduction B.2-1

B.2.1.1 Description B.2-1
B.2.1.2 Objective of Sealing/Backfilling Operation B.2-1
B.2.1.3 Purpose B.2-1

B.2.2 Materials B.2-3

B.2.2.1 Description B.2-3
B.2.2.2 Properties B.2-3

B.2.3 Equipment B.2-3

B.2.3.1 Drilling B.2-3
B.2.3.2 Grouting B.2-3
B.2.3.3 Monitoring B.2-4

B.2.4 Sealing Operations B.2-4

B.2.4.1 Sequence of Operations B.2-4
B.2.4.2 Conditions of Emplacement B.2-7
B.2.4.3 Problems Encountered B.2-7
B.2.4.4 Quality Control B.2-8

B.2.5 Discussion B.2-8

B.2.6 References B.2-9

APPENDIX B.2
New Waddell Dam

B.2.1 Introduction

B.2.1.1 Description

The New Waddell Dam represents one of the final components of the Central Arizona Project (CAP), originally conceived by the U.S. Bureau of Reclamation (USBR) in the 1940s. The 337-mile-long CAP aqueduct brings water from the Colorado River across Arizona via canals, siphons, tunnels, pipelines, and pump stations. Located near Phoenix (Figure B.2-1), the New Waddell Dam is a zoned earthfill dam 4,700 ft long and 440 ft high, containing 17-million cubic yards (cy) of fill. It will be operated as a seasonal pumped storage facility, being filled from the Waddell Canal during the off season and delivering electricity and irrigation water during periods of peak demand. The new dam replaces the original Waddell Dam, a concrete structure that will be seasonally submerged in the 900,000-acre/ft impoundment, Lake Pleasant.

B.2.1.2 Objective of Sealing/Backfilling Operation

The New Waddell dam site is a difficult site that was primarily selected for practical and political reasons rather than geologic suitability. A fractured and faulted volcanic sequence underlies the impoundment, requiring extensive treatment in the form of excavation, dental concrete, cutoff walls, and grouting to make it suitable as a dam foundation. A very substantial cement grouting program was conducted to protect the embankment against uplift, piping, and erosion as well as to conserve valuable water. The latter consideration provided a large part of the justification for the additional grouting costs on this arid-region project, with CAP water running some 8 to 12 times the cost of local pumped groundwater.

B.2.1.3 Purpose

The New Waddell project is a case study in grouting a large dam foundation in fractured volcanic rock using cementitious grout. At present it represents state-of-the-art in large-scale cement grouting of fractured rock. At a review of the Yucca Mountain Repository Sealing Program (11/12/91, Seattle) by the presidentially appointed Nuclear Waste Technical Review Board (NWTRB), Chairman Don Deere recommended the New Waddell project as

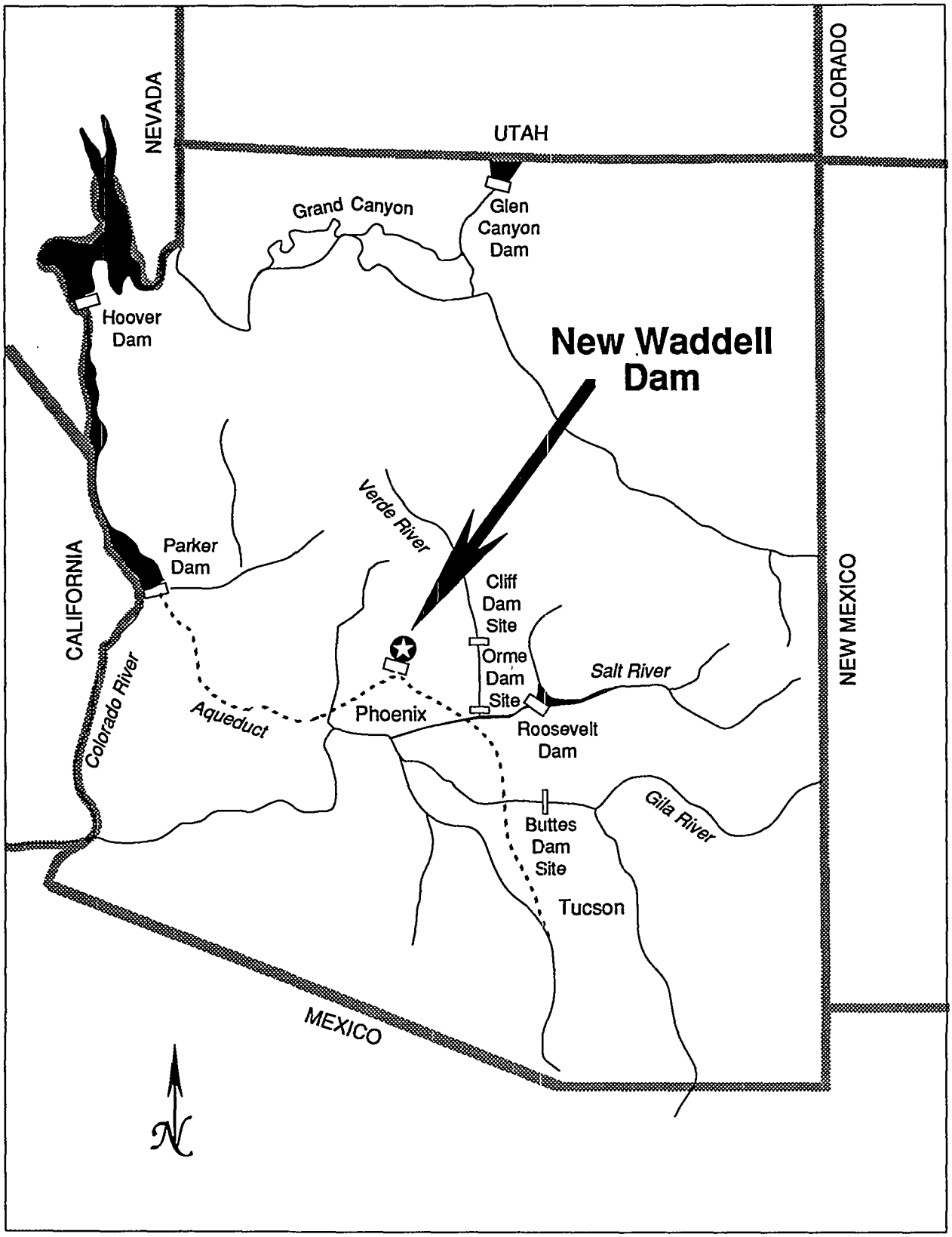


Figure B.2-1
Location of New Waddell Dam

an excellent case study of grouting in fractured volcanic rocks with possible application to Yucca Mountain.

B.2.2 Materials

B.2.2.1 Description

All grouting was performed using a Type II (moderate sulfate resisting) portland cement. Mixes ranged from 5:1 to 0.8 :1 water to cement by volume. The thinner grouts were applied first to seal the fine fractures, followed by progressively thicker grouts. Superplasticizers were added to the mix when water takes were low (typically less than 5 cubic ft per 5 min per stage) or when other conditions warranted. For example, superplasticizer was added to some of the holes in the Stage I (see Section 4.1) grouting in an attempt to obtain better penetration of any caved material in the grout holes. This was considered successful, and the use of superplasticizer was generally specified for subsequent grouting. No microfine cements or chemical grouts were applied on this project.

B.2.2.2 Properties

There is no information on the grout properties.

B.2.3 Equipment

B.2.3.1 Drilling

Following USBR practice, percussion drilling was permitted for the blanket holes, while rotary drilling was specified for curtain holes. Most drilling was performed with water to ensure good flushing of holes and to minimize clogging of fractures with drill cuttings; however, some dry percussion drilling was used for blanket holes where caving problems were encountered. The grout take in the dry holes was generally found to be about the same as in the wet holes. No drilling muds were used. Above the maximum water surface, casing was driven with a downhole hammer, and no grouting was performed. All drilling equipment used is available off the shelf.

B.2.3.2 Grouting

An automated batch plant was employed by the grouting contractor to provide convenient preparation of preselected mixes. High-speed centrifugal colloidal mixers were used. Grout pumps were "moyno" type (helical screw). Packers were about 3 ft long and inflatable. All grouting equipment used is available off the shelf.

B.2.3.3 Monitoring

A computerized monitoring system was employed for nearly all grouting at the New Waddell Dam (Aberle et al., 1990). This system has been in development by the USBR since 1982, and predecessors of the current system have been used on several previous dam projects, including Ridgeway Dam in Colorado, Upper Stillwater Dam in Utah, and Brantley Dam in New Mexico. Figure B.2-2 shows details of the system. Data are acquired in the field from flowmeters, density meters, and pressure transducers located between the grout plant and the header assembly at the grout hole. Data are transmitted from the sensors to the computer monitoring trailer using radiotelemetry.

Computers at the monitoring trailer enable real-time monitoring of grouting progress. Figure B.2-3 is a sample graph similar to that which would appear on the computer screen. Such information enables the grouting contractor to quickly correct any adverse conditions, and provides a complete record of grouting operations. Computer monitoring equipment is available off the shelf.

B.2.4 Sealing Operations

B.2.4.1 Sequence of Operations

Grouting was performed in two stages following USBR practice: Stage I involved blanket and initial curtain grouting. The grout blanket extends beneath zone one of the dam (the core) and consists of holes on 10-ft centers with alternate rows staggered. The curtain grouting was partly used for exploration purposes and was not generally taken to closure, although primary, secondary, and tertiary holes in a single row would generally be grouted. The information thus obtained was used to design Stage II grouting, which involved curtain closure holes on split spacing and additional rows of holes or multiple-row curtains as required. The USBR feels that specifying a two-stage grouting program leads to a more rigorously designed grouting program and reduced contractor claims.

Blanket holes are a minimum of 2 in. in diameter and are 30 ft deep. The holes are grouted in two 15-ft stages. The 15- to 30-ft stage is grouted at 20 pounds per square with (psi), while the upper stage is grouted at 10 psi. No grout caps were employed; however, all dental concrete was grouted.

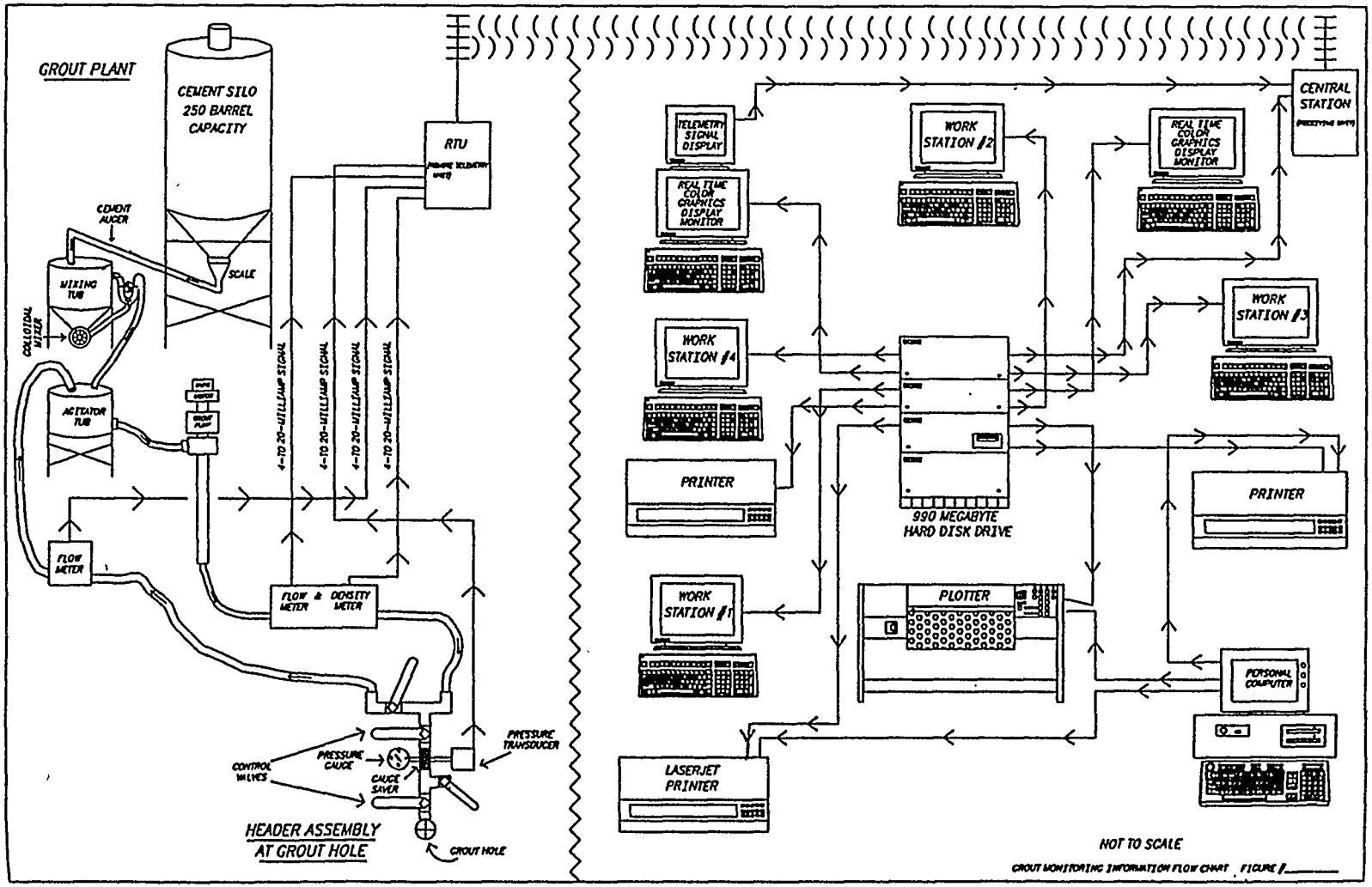


Figure B.2-2

New Grout Monitoring Hardware Scheme with Field Setup and Telemetry

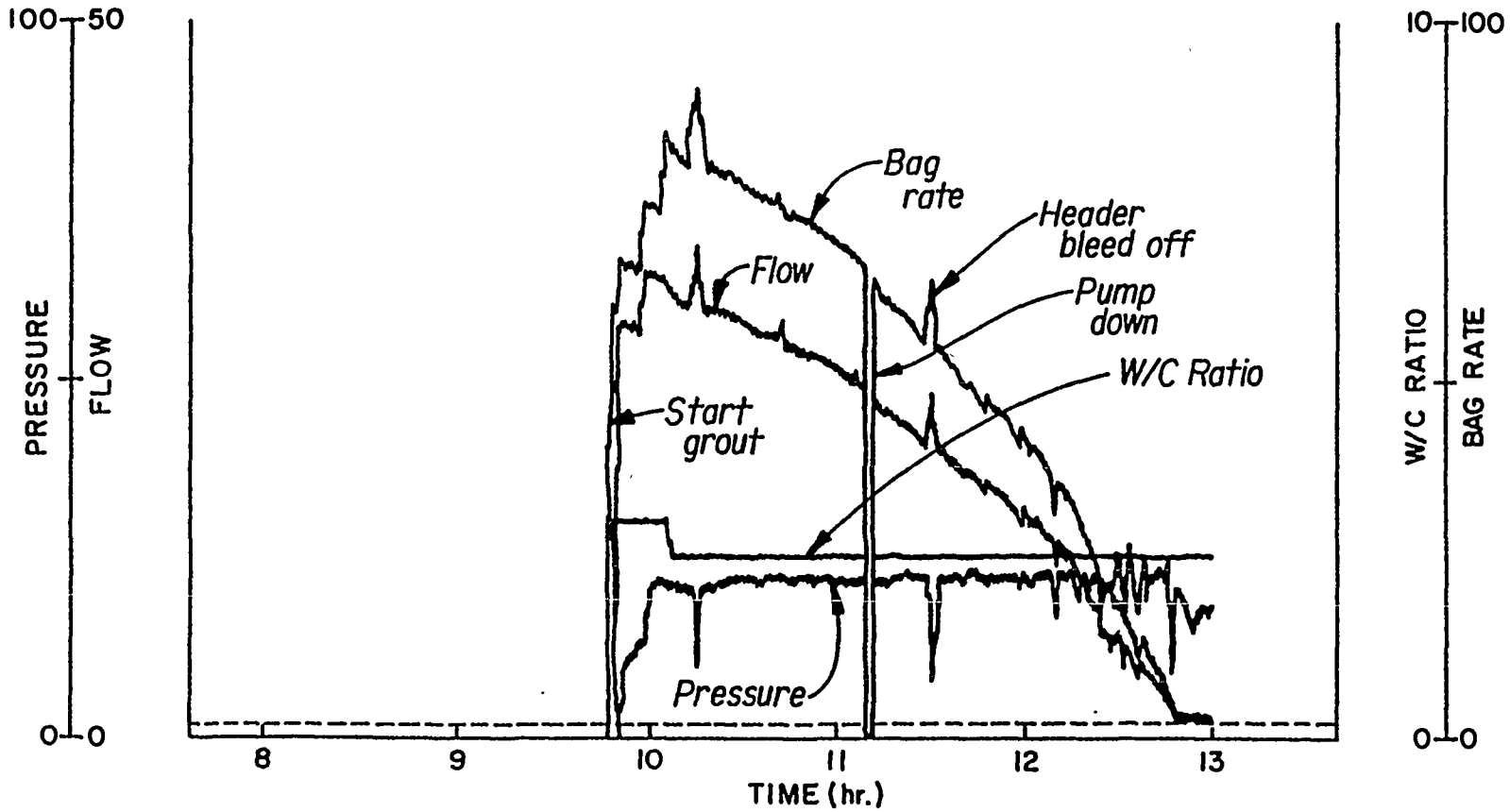


Figure B.2-3

Graphical Screen Display at Completion of Typical Grout Hole (Aberle et al., 1990, p. 9)

Stage I curtain grouting involves a single row of holes located 5 ft downstream from the dam centerline. Holes on 10- or 20-ft centers are grouted in 20- to 30-ft stages using inflatable packers. Stage-up techniques are used when possible. Grouting pressures were nominally 1- to 1.5-psi/ft depth. The curtains were taken to closure in Stage II.

Two major areas required grouting: (1) the main dam foundation and (2) the right abutment ridge, a narrow 7,000-ft-long spur that forms the southern rim of the reservoir. Stage I grouting for the main dam foundation was completed in October 1989 and involved a total of 275,000 ft of drilling into which 130,000 bags of cement were injected. Complete information for the other stages is not currently available. Stage II for the right abutment ridge is in process at the time of this writing.

B.2.4.2 Conditions of Emplacement

The New Waddell Dam is sited in a geologically complex volcanic series containing fractured andesites and interbedded tuffs and conglomerates with some eroded areas containing deep alluvium. Aside from the alluvium, the andesites and the andesite-tuff contact were generally the most conductive units requiring the most grouting. The conglomerate units were relatively tight. The tuff units required only moderate grouting, but were extremely susceptible to weathering, frequently requiring excavation. If tuff was exposed that was not to be excavated, it was immediately covered to prevent weathering.

Dental concrete was required in some locations. The conglomerate surface required dental concrete to fill erosional irregularities. The andesite occasionally formed an irregular surface when blasted. Some tuff surfaces were covered with concrete to prevent weathering.

Alluvium was removed where possible. Beneath the main dam foundation, the river bed had deep beds of alluvium that was impractical to excavate. A concrete cutoff wall, 400 ft long and 165 ft deep, was constructed across this channel.

Some faults that required grouting were encountered in the right abutment.

B.2.4.3 Problems Encountered

Several problems were encountered and solved in the course of the grouting project. First, some of the grout holes experienced caving during drilling, water testing, and prior to grouting. Drill water was found to cause slaking and deterioration of tuff units, and dry

percussion methods were used for some of the blanket holes to improve hole stability; superplasticizer was used to obtain better penetration of caved material in the holes. Because caving causes difficulties with packers, stage down methods were sometimes required.

Difficulties were experienced with early use of plasticizers. If a hole was initially grouted without plasticizer, a cement buildup occurred in the grout lines, especially in the prevailing hot summer temperatures. When plasticizers were then added to the mix, the buildup was loosened, clogging the hole. Procedural changes eliminated this problem.

B.2.4.4 Quality Control

Grouting was performed by contractors, while quality control was implemented by the USBR. Many aspects of the quality control program have already been discussed, including the two-stage grouting sequence. The computer monitoring system described earlier was a key component of the quality control system and also provided complete records of grouting for each hole. This is a significant improvement over other projects where quality assurance/quality control are performed by manual readings and hand-written notes in the field.

A very stringent refusal criteria of one-half bag per foot of grout stage, half the normal criteria, was used on this project. This reflects the high cost of water leakage.

The high degree of quality control exercised on this project is evidenced by the low rate of seepage observed through the grouted areas and the favorable measurements from instrumentation placed in the foundation and on the abutments.

B.2.5 Discussion

This case history demonstrates the highly effective sealing of a faulted, fractured, and eroded dam foundation using state-of-the-art conventional cement grouting. It is likely that some cementitious grouts will be used to seal a potential repository, and similar geologic conditions are likely in some areas at Yucca Mountain. Although grouting technology under ordinary conditions is mature, questions remain on whether suitable technology for grouting high-temperature rock is available.

B.2.6 References

Rosenbaum, D. B. (1992), "Huge Dam Caps Statewide Project," *Engineering News Record*, November 2, pp. 26-29. (NNA.940607.0140)

Aberle, P., R. Reinhardt, and A.D. Mendenhall (1990), "Electronic Monitoring of Foundation Grouting on New Waddell Dam," 1990 Annual Civil Engineering Convention, San Francisco, California, November 5-8.

APPENDIX B.3

Vat Tunnel

Case History from Literature Review

Purpose: To demonstrate the technologies of sealing high water inflow, seal emplacement, and handling and transporting large volumes of concrete.

Table of Contents

B.3.1 Introduction B.3-1

B.3.1.1 Description B.3-1
B.3.1.2 Objective of the Sealing Operation B.3-1
B.3.1.3 Purpose B.3-1

B.3.2 Materials B.3-3

B.3.2.1 Description B.3-3
B.3.2.2 Properties B.3-3

B.3.3 Equipment B.3-3

B.3.4 Seal/Backfill Placement Operations B.3-5

B.3.4.1 Sequence of Operations B.3-5
B.3.4.2 Conditions of Emplacement B.3-7
B.3.4.3 Problems Encountered B.3-9
B.3.4.4 Quality Control B.3-9

B.3.5 Discussion B.3-9

B.3.6 References B.3-9

APPENDIX B.3
Vat Tunnel

B.3.1 Introduction

The information presented below is condensed from a U.S. Bureau of Reclamation (USBR) report on the Vat Tunnel (USBR, 1989).

B.3.1.1 Description

The Vat Tunnel is part of the Strawberry Aqueduct System located in central Utah (Figure B.3-1). The tunnel is 38,760 ft in length, having a concrete liner over its entire length with a finished diameter of 8 ft 3 in. Construction of the tunnel began in August 1976 at the tunnel outlet using a drill-and-blast method for the first 52 ft. Tunnel excavation with tunnel boring machine (TBM) began in October 1976. On December 21, 1978, the TBM was stopped at Station 1057+97 due to high water inflows. Prior to proceeding with excavation, high-pressure grouting was required. Two years were required to penetrate through approximately 4,140 ft in a zone of high water inflow.

B.3.1.2 Objective of the Sealing Operation

The objective of the high-pressure grouting was to stabilize the rock so that the TBM drive could proceed. The difficulty with the operation was the back pressure and quantity of water encountered, 250 pounds per square inch (psi) and 1,300 gallons per minute (gpm), respectively.

B.3.1.3 Purpose

There are three areas of potential relevance of this case history to the Yucca Mountain sealing program.

- Demonstrating that a high water inflow can be effectively sealed off.
- Demonstrating that a seal can be emplaced.
- Demonstrating handling and transport of large volumes of concrete.

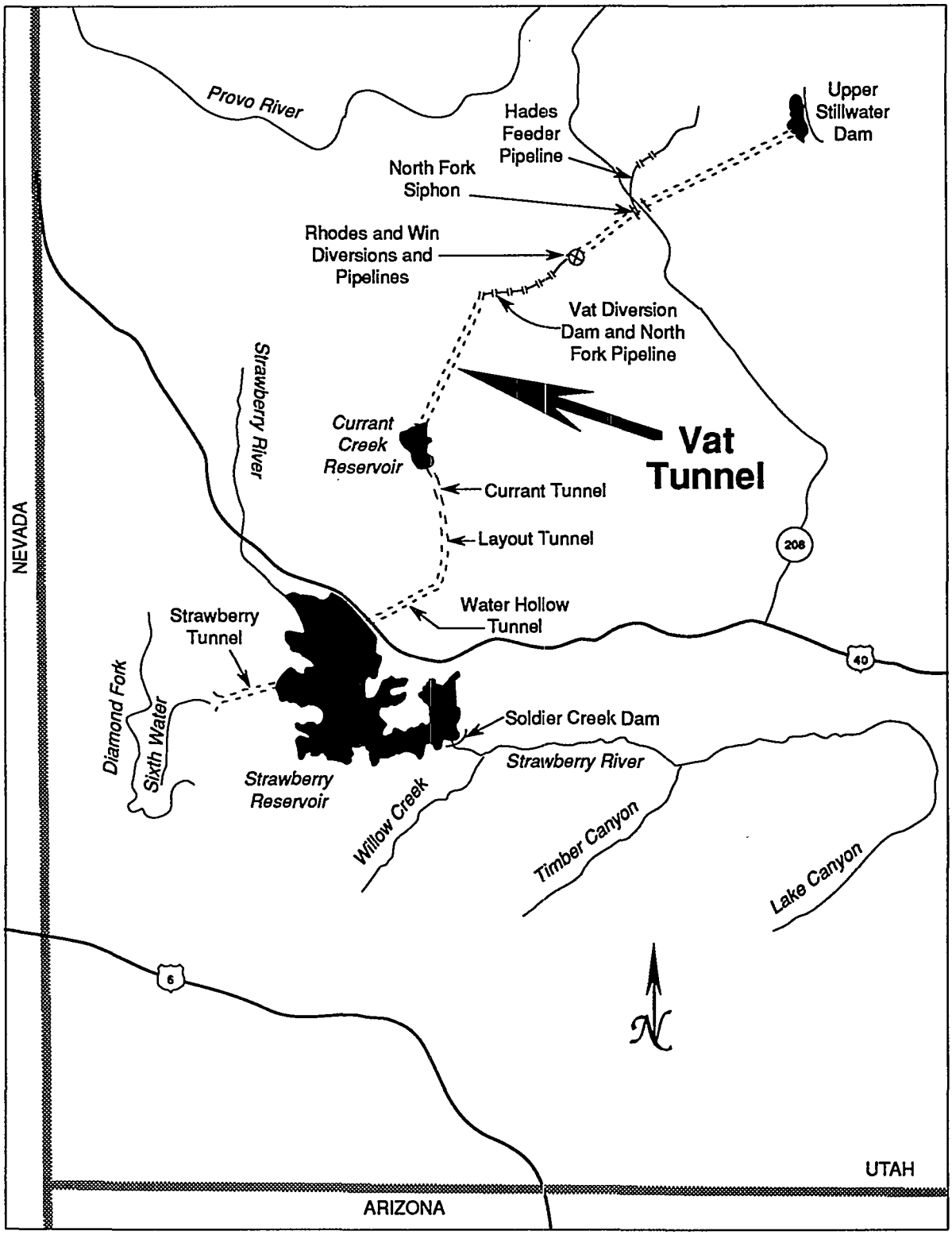


Figure B.3-1
Location of Vat Tunnel

B.3.2 Materials

B.3.2.1 Description

Type II cement was used in the grout. A total of 55,082 sacks of cement were used in 26,473 linear feet of drill hole, for an average of 2.08 sacks of cement per foot.

To line the 38,760 linear feet of tunnel, 48,943 yd^3 of concrete (or 1.26 yd^3 of concrete per linear foot of tunnel) were placed. Type V cement was used in the concrete.

B.3.2.2 Properties

The grouts were initially tried using thick grout mixtures of 1:1, 2:1, and 3:1. Because very little grout travel was noted in a 118-ft core hole, a thinner grout mixture was used when first injecting grout into a hole.

B.3.3 Equipment

Grouting was performed by Gardner-Denver, 10 by 3 by 10, piston-type pump together with 20 ft^3 side-by-side mixing tubs. Generally, grouting was performed using only one pump, with a second available as a backup.

Concrete batching and mixing was needed primarily for the concrete liner. The concrete aggregate was carried by a movable (stacking) conveyor belt to a triple deck screening and rewash unit. Sand was transported by front-end loader to a loading hopper. The concrete was mixed in a 3-yd^3 Model 56, Turbo-Flo mixer. The batch size was 3.5 yd^3.

The concrete was delivered to the placement site by four Moran cars containing a total of 28 yd^3 of concrete. The concrete was placed with a modified Thomson concrete pump with twin 9-in.-diameter, 36-in.-stroke pistons. The concrete pump and related equipment were mounted on a special rail car (Figure B.3-2). The concrete was pumped through a 6-in.-inside diameter slickline and transported up to a distance of 280 ft. Prior to concrete placement, 2 yd^3 of sand/cement grout were pumped through the slickline for lubrication. Finally, the concrete was consolidated by a Model TV7 air-driven form vibrator.

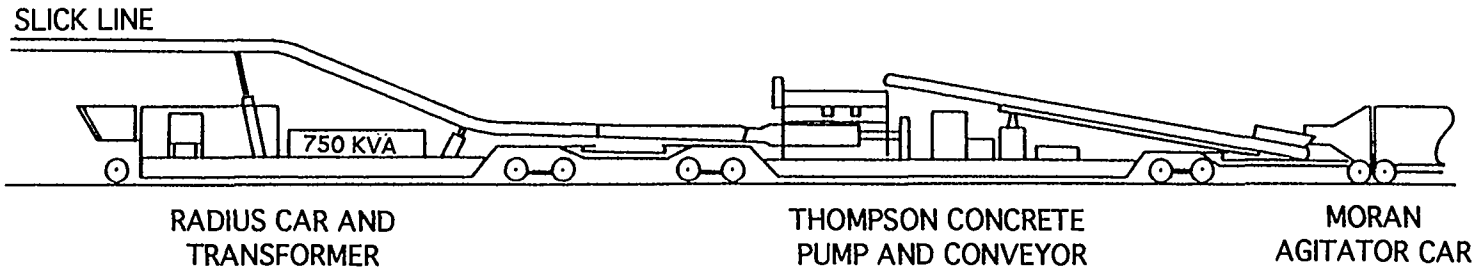


Figure B.3-2

Placing Equipment Layout for Concrete Lining Vat Tunnel

B.3.4 Seal/Backfill Placement Operations

B.3.4.1 Sequence of Operations

Between stations 1058+97 and 1012+97 (a distance of 4,600 ft), high-pressure grouting was required to reduce the water inflow. Over this distance, a total of 15 reaches were needed to control water. Grouting this distance took over 2 years. Following this grouting, normal TBM operations resumed.

For the first seven reaches of grouting, audits were excavated to allow initiation of the drilling and grouting program. These audits were 28 to 32 ft in length. Following initial application of groutcrete, a concrete plug was placed. Then grouting was performed around the plug. The grout holes were 1.5 to 2.0 in. in diameter. Prior to grouting, back pressures were taken. The back pressures ranged up to 260 psi, and water inflows were as high as 600 gpm. The rock pressures ranged up to 260 psi, and water inflows were as high as 600 gpm. The rock grouting over the 15 reaches is summarized in Table B.3-1. Injection pressures as high as 1,000 psi were used. In all cases, the grout mixture started out at a ratio of 10:1 and thickened to as high as 1:1. The pumping pressure for backfill grouting (i.e., grouting behind the concrete liner) was 30 psi plus the backfill pressure before grouting. Where the pressures were high (260 psi), backfill grouting could not be performed.

Starting on the second grout reach, some stage grouting was implemented. When holes were interconnected, they were grouted simultaneously. Two or more interconnected holes were grouted together approximately 30% of the time. After the grout reach was completed, the TBM drive continued until excessive water flows saturated the muck. The majority of the grouting took place in the Twin Creek Formation. The following procedure presented in the USBR report (1989) was used in controlling water flow.

1. The trailing gear was unhooked and pulled back about 8 ft from the mole.

2. A 4-ft by 7-ft-wide passageway, along whichever side of the mole has the most competent rock, was spaded and supported by steel ribs and lagging that would stand independently of the mole.

3. Steps 1 and 2 were repeated until 30 ft of passageway had been constructed.

Table B.3-1
Summary of Grouting Reaches

Reach	Station to Station		Cement (Sacks)	Number of Holes	Depth (Feet)	Injection Pressures	Total Feet of Holes	Grout Mix
1	1057+97	1059+09	8146	26	15 - 163	350 - 1000	1697	
2	1057+09	1055+37	9453	14	59 - 245	350 - 1000	2269	10:1 - 7:1
3	1055+37	1054+01	3717	9	42 - 278	100 - 900	1737	10:1 - 6:1
4	1054+01	1052+33	4263	10	40 - 293	100 - 850	2249	10:1 - 4:1
5	1052+33	1051+41	2344	8	190 - 255	200 - 950	1618	10:1 - 8:1
6	1051+41	1049+37	3561	9	195 - 240	150 - 1000	1865	10:1 - 2:1
7	1049+37	1046+65	1780	9	155 - 238	200 - 850	1775	10:1 - 2:1
8	1046+65	1044+21	1357	9	173 - 310	150 - 750	1883	10:1 - 2:1
9	1044+21	1042+33	1751	9	68 - 205	200 - 950	1571	10:1 - 2:1
10	1042+33	1037+93	734	9	50 - 210	150 - 950	1740	10:1 - 2:1
11	1037+93	1029+81	886	11	75 - 250	150 - 850	1233	10:1 - 2:1
12	1029+81	1026+77	3206	8	165 - 300	175 - 850	1690	10:1 - 2:1
13	1026+77	1021+21	4881	9	53 - 300	150 - 850	1840	10:1 - 3:1
14	1021+21	1016+57	2964	7	48 - 320	100 - 700	1826	10:1 - 1:1
15	1016+57		6039*	5	110 - 315	50 - 350	1480	10:1 - 1:1

*Risers installed in voids, Stations 1016+61 and 1016+64. Grout take was 3,270 sacks of cement at a 1:1 grout mix. Intermittent grouting on 1-hour intervals.

4. Existing steel ribs and subinverts were removed from behind the mole.

5. The mole was pulled back and the sides of the passageway shotcreted.

6. To provide a safe working area for grouting, temporary ribs or shotcrete supports were installed in front of the mole, from the end of the newly constructed passageway to within ± 12 ft of the heading.

7. An 11-ft-long (37-yd^3) concrete or shotcrete plug was placed with embedded plastic grout pipes and a drain on the bottom. The seepage water on the face of the tunnel walls was panned off.

8. To help decide the grout plan, core drilling was accomplished ahead at least 100 ft to examine the rock formation.

9. Grouting proceeded through the embedded pipes.

10. If the plug moved, a collar was installed. If there was no movement, drilling and grouting ahead continued, using information from the core hole as a guide.

The grouting operations reduced the water into the tunnel by 80% from varying flows of 900 to 1,300 gpm to 200 gpm.

B.3.4.2 Conditions of Emplacement

The Vat Tunnel was constructed through 11 formations. These formations were comprised of shales, mudstones, claystones, sandstone, siltstone, and limestone. In the majority of formations, the water inflows ranged from no water to 30 gpm. It was within the upstream portion of the Entrada Sandstone Formation (Je) that large water inflows were encountered near station. Three holes, up to 50 ft, were drilled to bleed off the flow. Back pressures of 250 psi were encountered. However, rather than bleeding off the flow, these holes enhanced the flow by 400 gpm up to a total of 1,300 gpm within 70 ft of the heading. High water inflows also continued in the following formation, the Twin Creek Limestone Formation (Jtc), necessitating periodic high-pressure grouting in most of this formation. A portion of the geology in this area is illustrated in Figure B.3-3.

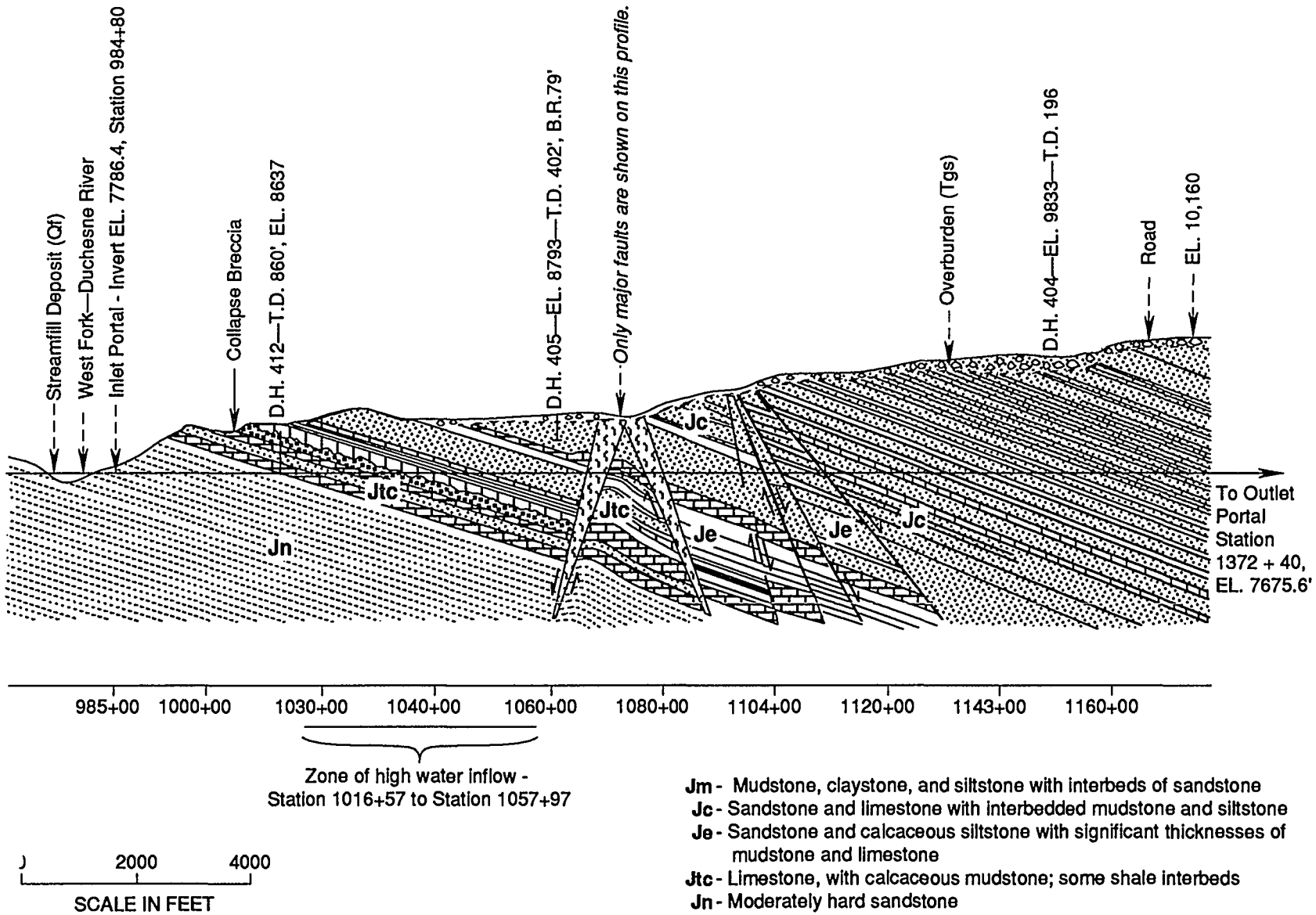


Figure B.3-3

Partial As-Built Geologic Profile of the Strawberry Aqueduct Vat Tunnel (Modified from U.S. Bureau of Reclamation, Drawing 66-436-1830)

B.3.4.3 Problems Encountered

Some difficulties encountered in the concrete placement included:

- Concrete did not have consistent slump at the concrete pump due, in part, to segregation and varying moisture content of the sand.
- Rock or lagging falls out and into the concrete.
- Slickline is removed too fast, resulting in poor consolidation and large voids in the concrete.

B.3.4.4 Quality Control

Concrete samples were taken at the batch plant by scooping the amount from the agitator car loading door after loading. Periodic slump and entrained air tests were taken at the tunnel placement sites to assess the effects of transportation from the batch plant to the placement site on these properties.

B.3.5 Discussion

Review of this case history illustrates the ability to emplace grout under adverse conditions of high formation water pressure and high volumes of water, although these operations proceeded slowly. The case history also illustrates common methods for concrete preparation, transportation, and placement. Some of the difficulties presented suggest the need for control measures and procedures during placement of the materials. While these conditions are not anticipated at Yucca Mountain, this case history infers the comparative ease in grouting in a more benign site like Yucca Mountain, where water pressures and volumes would be much lower.

B.3.6 References

U.S. Bureau of Reclamation (1989).

APPENDIX C.1

Nevada Test Site (NTS)

Case History from Literature Review

Purpose: To review the technologies used at the NTS for stemming tunnel complexes and large-diameter boreholes and to assess their relevance to the YMP Repository Sealing Program.

ACKNOWLEDGMENTS

The summary in this appendix is meant to provide a cursory review of the U.S. underground testing program at the Nevada Test Site (NTS). The technologies used at the NTS represent decades of practical experience in stemming nuclear explosions underground. The Defense Nuclear Agency (DNA) is responsible for conducting these tests with assistance from the military services, their contractors, and the U.S. Department of Energy laboratories and their contractors.

The material on horizontal line-of-sight testing contained in this appendix was extracted by William H. Barrett, Sandia National Laboratories (SNL) (Organization 9311), from the Containment Review documents that the DNA prepares for the Containment Evaluation Panel (CEP). The CEP is a group that reports to the Manager of Nevada Operations, Department of Energy. The purpose of the panel is to evaluate, as an independent organization, the containment design of each proposed nuclear test and advise the Manager of Nevada Operations of the technical adequacy of such design from the viewpoint of containment, thus providing the manager a basis on which to request detonation authority.

Bob Bass, SNL (retired), was of immense help in providing guidance for the containment section.

Dave Thomson, SNL (Organization 9331), summarized information on the vertical line-of-sight testing program.

John Boa (Waterways Experiment Station) provided important information on grout designs and concrete and grouting placement techniques to the authors of the report.

J. W. "Spike" LaComb, Jr. (Raytheon Services-Nevada) provided information on stemming considerations and materials and selected procedures for placement of the grout.

Table of Contents

C.1.1 Introduction C.1-1

C.1.1.1 Overview of the Nevada Test Site C.1-1
C.1.1.2 Underground Nuclear Tests C.1-1
C.1.1.3 Effects Tests C.1-2

C.1.2 Definitions C.1-4

C.1.3 Type of Testing C.1-6

C.1.3.1 Containment C.1-6
C.1.3.2 Vertical Tests C.1-7
C.1.3.3 Horizontal Tests C.1-13
C.1.3.4 Cavity Tests C.1-35

C.1.4 Materials C.1-39

C.1.5 Placement C.1-42

C.1.5.1 Mixing and Transport C.1-48
C.1.5.2 Stemming and Grouting Operations C.1-49

C.1.6 Discussion C.1-54

C.1.7 References C.1-54

APPENDIX C.1
Nevada Test Site (NTS)

C.1.1 Introduction

C.1.1.1 Overview of the Nevada Test Site

The Nevada Test Site (NTS), located 65 miles northwest of Las Vegas, is the nation's only facility for testing nuclear weapons. The 1,350-square-mile site, bordered on three sides by the Air Force's Nellis Bombing and Gunnery Range, is a secure facility operated under the purview of the U.S. Department of Energy's (DOE) Nevada Field Office (NV), headquartered in Las Vegas. DOE/NV maintains an on-site presence in the Nevada Test Site Operations Office, which provides direct oversight and guidance in carrying out the site's primary mission of nuclear weapons testing.

Of the 849 announced tests that have been conducted through September 1992, 5 were shot underwater, 212 were atmospheric, and 632 were underground. The underground tests (UGT) were of three types: tunnel (67), vertical shaft (535), and crater (9). Of the 849 tests, 721 were carried out within the confines of the NTS. One hundred were atmospheric (an additional five were done on the bombing range just outside the NTS), and 621 were performed underground. Of the 621, 9 were placed in unstemmed holes, another 9 produced (expected) craters, 1 event (Sulky) was expected to produce a crater but did not, 535 were stemmed vertical shafts, and 67 were horizontal tunnel events. All of the tunnel shots and all of the stemmed vertical shafts were designed to contain radioactive gases and particulates.

C.1.1.2 Underground Nuclear Tests

There are two major types of nuclear tests conducted underground at the NTS. The most common type of test, and the one most people think of when UGTs are mentioned, is the nuclear weapons development test. These are conducted in vertical holes in the valley known as Yucca Flats, about 30 miles north of Mercury on the NTS. Los Alamos National Laboratory (LANL) and Lawrence Livermore National Laboratory (LLNL) conduct these tests for the development of new nuclear weapons and to study weapons physics. The other type of test is the weapons effects test. These tests are conducted by the Defense Nuclear Agency (DNA) to study the effects of nuclear weapon radiation on our military weapons systems. The military services, their contractors, and DOE laboratories all participate in the DNA tests.

C.1.1.3 Effects Tests

Effects tests are of three types: vertical line-of-sight (VLOS), horizontal line-of-sight (HLOS), and cavity. Each test is given an unclassified code name for more convenient reference.

1. The VLOS tests are conducted in vertical holes drilled in Yucca Flats. They differ from the weapons effects tests conducted in the same area in two major ways. First, they have an open line-of-sight vacuum pipe from the nuclear explosive device to a large vacuum chamber on the surface. Second, large satellite, or other electronic hardware in the vacuum chamber, is exposed to the radiation device. The last VLOS test was code-named HURON KING and was conducted in 1980.

2. The HLOS effects tests are conducted in horizontal tunnels mined into the side of the adjoining Rainier and Aqueduct Mesas in Area 12, about 45 miles north of Mercury on the NTS. Figure C.1-1 shows the six major tunnel complexes that have been used for effects tests over the last 30 years. Each tunnel complex has been used for many tests. The last HLOS test was HUNTERS TROPHY and was conducted in N-tunnel in September 1992. The purpose of HLOS tests is to expose structures, materials, and electronics from U.S. missiles, warheads, satellites, and battlefield weapons and communications systems to the radiation from a nuclear weapon. The tests are part of the broad effort to make the U.S. military system more resistant to a nuclear explosion. In military language, they are "hardened" to make them more "survivable." HLOS events are conducted in large, tapered vacuum pipes that have varied in length from about 800 to nearly 2,000 ft, depending on the objectives of the experiments that were exposed to the nuclear radiation. The test chambers where the experiments are mounted have varied in size from about 8 to 38 ft in diameter, with 10 to 12 ft more typical of recent shots.

3. Cavity tests are also performed in the horizontal tunnels, but have a different purpose than the HLOS tests. They are conducted in hemispherical, air-filled chambers 30 to 60 ft in diameter. The purpose is to study air blast and the interaction of the nuclear explosion with the ground, including cavity formation, ground shock, and the response of buried, scaled structures (such as missile silos and bunkers) to ground shock.

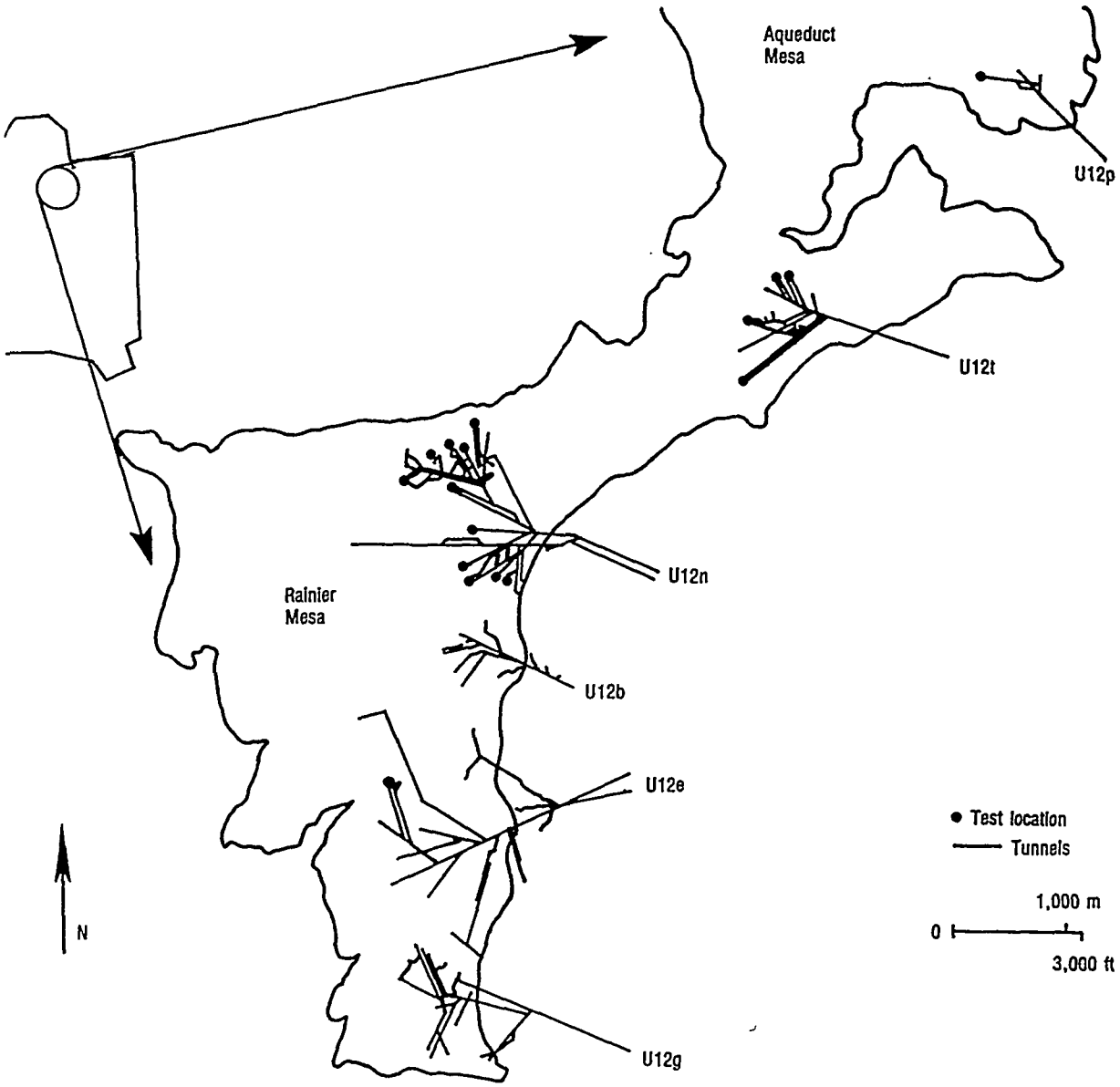


Figure C.1-1
Locations of Tunnel Complexes in Rainier and Aqueduct Mesas on the NTS

C.1.2 Definitions

Because of the nature of underground testing at the NTS, unique acronyms and definitions have been developed. A subset of these are given below to enhance the readability of this appendix.

1. **Box A or A-Box.** The steel box containing the nuclear explosive device.

2. **Construction Station (CS).** The distance measured from the beginning of a tunnel or drift. Since the working point is at the end of the main (or pipe) drift, the CS is a large number, while the range station is 0.

3. **Device.** The nuclear explosive. In nuclear testing parlance, it is not a weapon since it has no delivery hardware or fusing components.

4. **Drifts.** Tunnels mined off the main access tunnel. These may be used for the LOS pipe, work drifts, access drifts, or recording alcoves.

5. **Groutcrete (GC).** A mixture of portland cement, concrete sand, aggregate, and water, proportioned to render a mortar-like consistency. The mixture is pump-placed and usually used to form the top portions of various underground massive containment structures.

6. **Grouts.** Mixtures of suspended solids, emulsions, chemicals, and other solutions usually pressure-injected into the subsurface to improve geotechnical properties, form structures, improve foundation support, and form water and gas barriers.

7. **HSG.** High-strength grout.

8. **HSGC.** High-strength groutcrete.

9. **HSRM.** High-strength rock-matching grout.

10. **Invert.** The floor of a mined excavation.

11. **Low temperature rock-matching grout (LTRM).** A rock-matching grout that cures at a lower temperature than normal rock-matching grout, typically 125°F.

12. **Portal.** The entrance to the tunnel.

13. **Rib.** The side walls of a mined excavation.

14. **Rock-matching grout (RMG).** A grout that has been formulated to match the geotechnical properties of a particular kind of rock.

15. **Recording Oscilloscope Sealed Environmental System (ROSES).** A metal box about 8 x 8 x 12 ft that is used to house recording instrumentation in underground recording alcoves. It provides an environmental and electromagnetic interference (EMI) seal from the tunnel environment.

16. **Range Station (RS).** The distance measured from the working point to a specified location.

17. **Stemming.** Material such as sand, grout, and concrete that is placed in open drifts or around structures. Also, the action of emplacing these materials and the action during ground shock passage that closes the line-of-sight pipe.

18. **Superlean grout (SLG or HSSL).** A pumpable grout mixture, proportioned to contain a very low cement content, a very high water content, and large volumes of graded sand or desert fines and bentonite that will render a very low strength, density, and sonic pulse velocity.

19. **Vistanex.** A thick, very viscous liquid used in cable gas blocks to entrap any cavity gases traveling inside cable jackets.

20. **Working point (WP).** The center of the nuclear device.

21. **Zero room.** The excavated volume immediately surrounding the Box A.

C.1.3 Type of Testing

The types of testing discussed below include VLOS, HLOS, and cavity tests.

C.1.3.1 Containment

Containment is the general term used to describe the analysis, materials, structures, and design efforts that go into making sure that the nuclear explosion is controlled. That is, only the desired portion of the weapon output is allowed to interact with the effects test hardware, and all of the weapon output is kept underground. For HLOS and VLOS tests, only the radiation is allowed to fall on the test objects. For cavity tests, the entire weapon output, radiation, debris, and hot, high-pressure, radioactive gases are kept inside the premined or nuclear cavity by massive plugs and stemming.

Containment practices have evolved steadily since the first test detonation in Rainier Mesa on September 19, 1957. In addition to the effects tests conducted in these areas, there have been a significant number of high-explosive (HE) simulation experiments directed toward a better understanding of containment grouts out to approximately the same distance from the WP as in the main drift. This operation is called "button-up" stemming. Vessel I is usually terminated beyond the mechanical closure locations where earth shock has decreased to considerably below the kilobar level. Vessel I is designed to protect the experiment by preventing damage to the equipment and allowing it to be recovered.

Vessel II is intended to protect the remainder of the tunnel system in the event of loss of containment in Vessel I. Vessel III is intended solely to prevent leakage of radioactive materials to the atmosphere (U.S. Congress, Office of Technology Assessment, 1989). Each of the containment vessel regions is separated from the others by massive concrete plugs faced on the explosive side with steel bulkheads. The plugs between Vessels II and III are designed to withstand temperatures of 1000°F and pressures of 1,000 pounds per square inch (psi). The plugs between Vessel III and the rest of world are designed to withstand 500°F and 500 psi. The tuff regions surrounding these plugs are drilled and pressure-grouted to prevent leakage and are pressure-tested to 2 psi before test execution. Manways that are designed for the same temperature and pressure levels are provided through these plugs for late time access.

Necessary tunnel facility lines, such as tunnel dewatering lines passing through these plugs, are redundantly valved and repeatedly tested prior to test execution. Hundreds of power and experiment cables emplaced through the plugs either use factory gas blocked cable, or the individual conductors or fibers are fanned out and run through Vistanex baths in cable gas blocks. Prior to the adoption of the cable gas blocking efforts, radiation leakage through cables was a major problem.

Vessel I, shown in Figure C.1-2 for an HLOS event, is designed to contain the nuclear explosion. The tapered vacuum pipe of an HLOS event is only a few inches in diameter inside the Box A, which contains the nuclear device, and typically expands to 10 to 12 ft at the end of the 1,000-ft pipe. Since there is no vacuum pipe in a cavity test, very massive stemming columns and plugs provide the primary containment.

An HLOS test is a fully tamped event; that is, the nuclear device is completely surrounded by rock or stemming material. In this configuration, the vast majority of the energy of the device is coupled into the surrounding media. Sufficient plastic work is done on the surrounding formations during the cavity growth that a postshot residual stress field is formed. These compressive stresses are large enough compared to the cavity pressure that containment is assured.

C.1.3.2 Vertical Tests

Of all tests conducted at the NTS, 90% are of the vertical configuration. Most of these are of a lesser yield (<20KT) and are conducted at Yucca Flats; larger-yield vertical tests are conducted on Pahute Mesa. Since March 1976, the maximum yield has been less than 150KT, by agreement with the former Soviet Union.

Since 1962, all tests at the NTS have been designed to be contained UGTs. Stemming the borehole or shaft is an essential part in containing underground vertical tests. Stemming, in general, is backfilling the borehole shaft/drift with appropriate material to prevent the scattering of activated debris.

C.1.3.2.1 Stemming Design

The primary concern in any stemming design is to confine the debris carried by the pressure wave and contain the gas generated from the device to the extent that the stemmed shaft/drift does not provide a preferential path for this material/gas to exit the cavity. Pressure waves from the device detonation may generate fracture paths for the gas to escape, but a successful

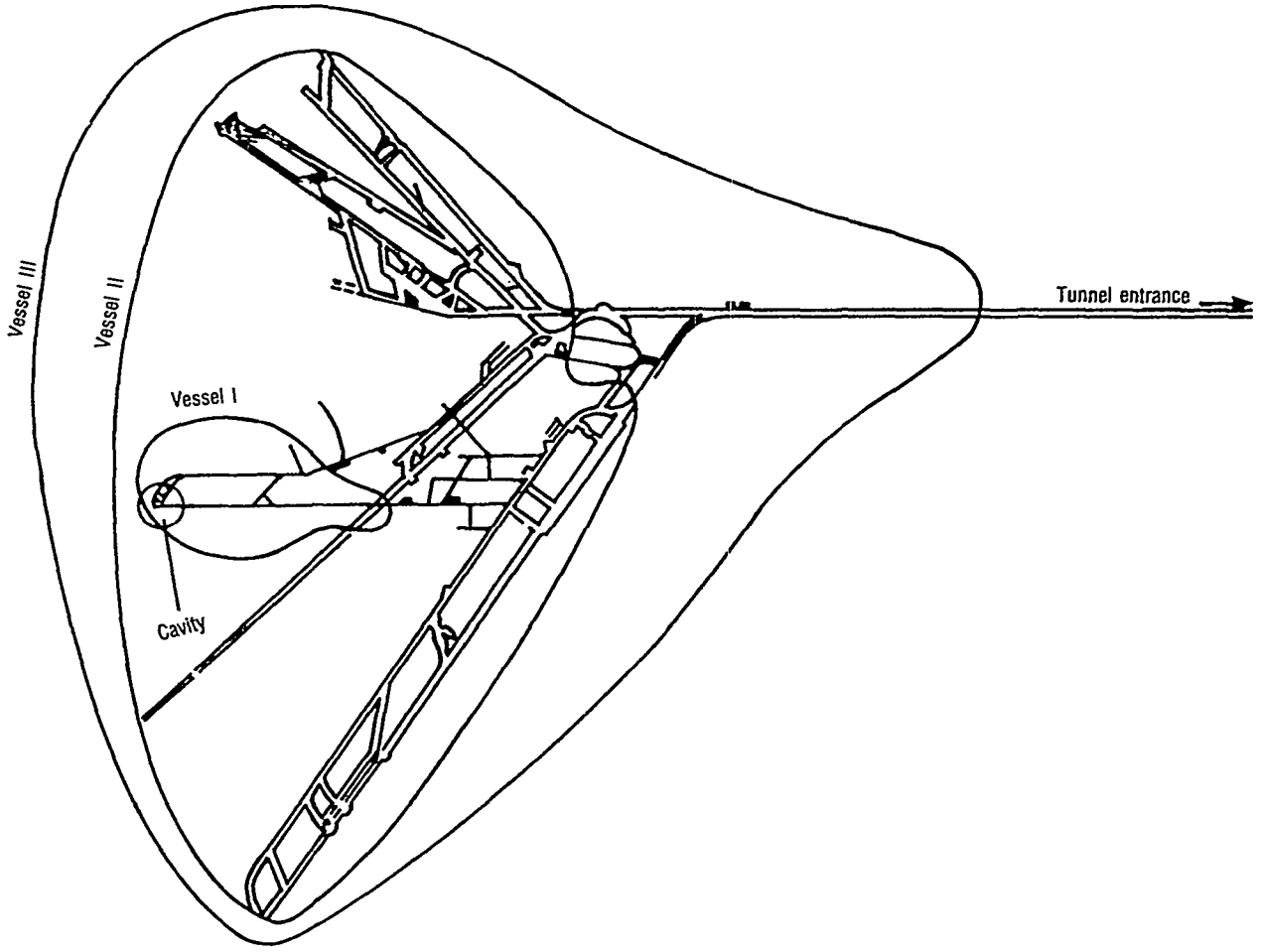


Figure C.1-2

Three Redundant Containment Vessels (Plan View) for MIGHTY OAK in the T-Tunnel Complex (U.S. Congress, Office of Technology Assessment, 1989)

containment design will have fracture paths generated in the surrounding medium rather than the stemmed shaft. The primary factor in providing the necessary restriction is the mechanical strength of the material used.

At the NTS, conventional drill bits are used for boring holes with diameters from 9 in. to about 30 in. Boreholes larger than 30 in. are considered "large-bore" holes, and larger bits have been developed at the NTS to bore holes of larger (up to 142 in.) diameters. Typically, canisters (or racks in LANL terminology), the framework on which the bomb and recording equipment are mounted, are about 86 in. in diameter. This implies a drill hole of about 96 in. in diameter below the surface casing and a surface casing with an inside diameter of about 98 in.

The casing is typically stainless steel and runs to a depth of between 25 to 75 ft. Its function is to provide a stable footing at the hole's edge to allow heavy equipment to approach the hole without causing the sides to give way. The depth to which the casing extends down the hole will depend on the type of soil into which the hole is being bored. The depth to which the hole will be bored will depend on the expected yield of the device to be employed and the soil characteristics of the area in which it is being emplaced.

A typical example of the stemming configuration used by each weapon development laboratory is shown in Figure C.1-3; U19az is LANL's configuration, and U20az is LLNL's. The items of common interest in these drawings are the shaft diameter and depth, the depth of the casing at the surface, the shielding about the working point (WP = bomb location in terms of depth of burial [DOB]), and the manner in which the acquired data are brought to the surface—specifically, in the penetration of the cables through the gas seal plugs. Also, both stemming plans have included Remote Air Monitoring System (RAMS) detectors within the stemming to monitor migration of the radioactive gases back up the stemmed shaft.

The figure also shows that both laboratories use an alternation of coarse and fine stemming material. To simplify procurement of these materials, the two laboratories have standardized on common source requirements. The coarse material is aggregate of 5/8-in. diameter or less. Its function is to use the interstitial space between the gravel as an expansion chamber to attenuate the gas pressure, in stages. The fines material is desert sand, and its function is to impede the propagation of the gas by creating a back pressure that will force the gas into the surrounding rock medium. LANL mixes bentonite (a very fine clay) with their sand (3 parts sand to 1 part bentonite) to create more back pressure by filling the interstitial spaces in

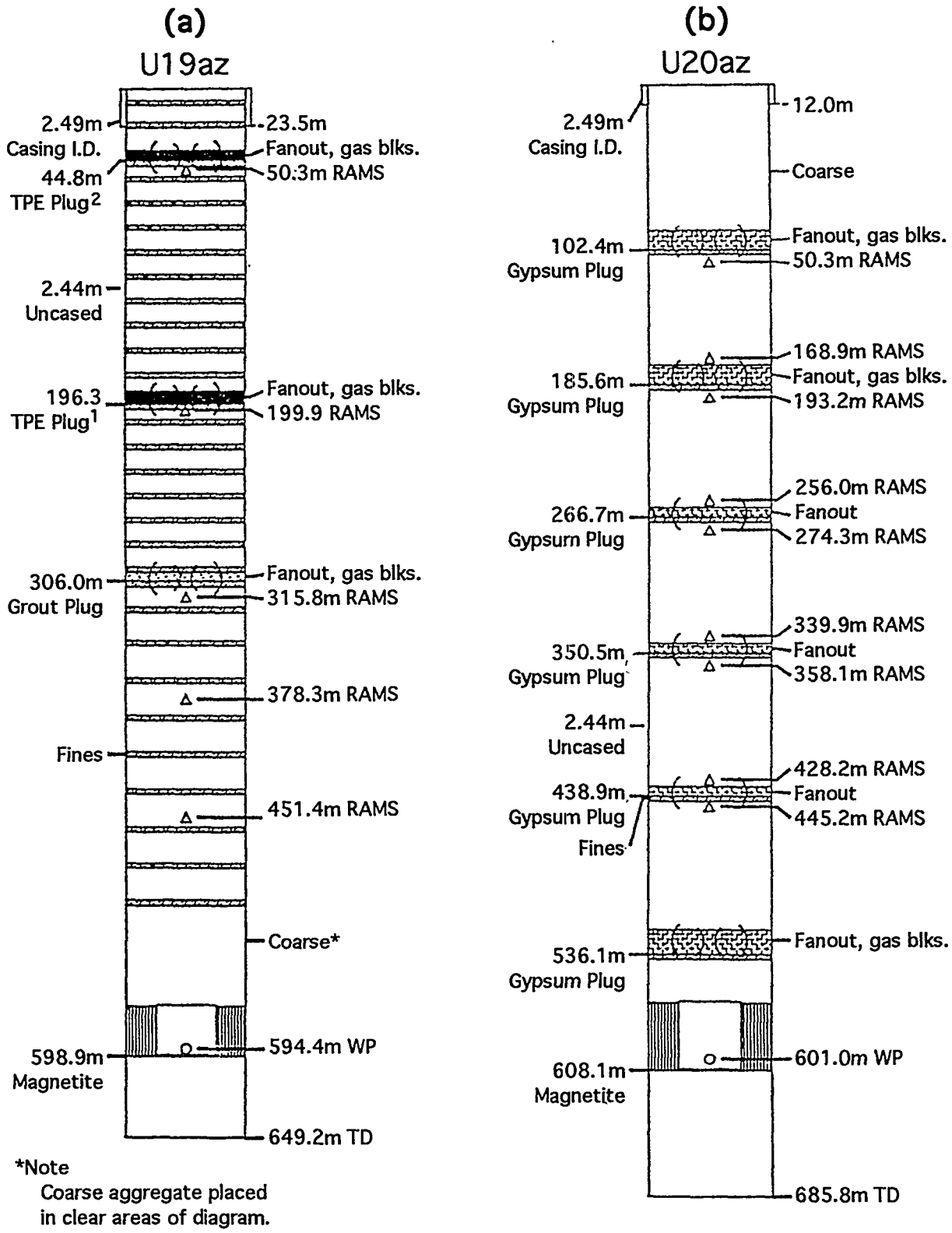


Figure C.1-3

Typical Stemming Design for Vertical Shaft:
(a) LANL's Configuration and (b) LLNL's Configuration

the sand; LLNL simply uses sand. Also, LANL uses a more rapid alternation of stemming materials to drive more of the gas into the surrounding rock at a greater depth.

LLNL uses gypsum plugs as a gas confinement barrier, while LANL prefers a pair of Two Part Epoxy (TPE) plugs, supplemented by a grout plug, for gas containment. In both cases, the data cables connecting the downhole detectors to the uphole recording equipment must penetrate the plugs. Because the cable jackets provide a potential preferential path for the gas to escape, the plug penetration is used to preclude this possibility. To accomplish this, the outer cable jacket is stripped from the penetrating portion, and the individual (but still insulated) wires are separated to truncate the path the gas could use (within the cable jacket) to escape the downhole environment. Furthermore, the cables are cut and connected to a terminal block, thus physically severing the cable path a gas could use. The terminal block is then placed within a polyvinyl chloride (PVC) pipe and potted with a polyurethane material to ensure that a propagation path does not exist. This technique is known as discrete gas blocking and has proven to be effective over the years. Recently, cables have been designed where gas blocking material is designed into the cable. This is accomplished by wrapping the conductors with tape that, when heated, melts and flows into the interstitial spaces about the conductors. This is known as continuous gas blocking; it has not as yet garnered the confidence of the field test community that discrete gas blocking enjoys.

The stemming configuration used by LLNL shows large sections (about 250 ft thick) of coarse material. Gravel is used because of its ease of delivery and it doesn't require compaction. About 250 tons/hr is delivered downhole by a conveyor system with an attached weightometer, which weighs the amount of material dumped downhole over a period of time. At various locations, as specified in the stemming plan, 15-ft plugs of fine (i.e., sand) material is laid down as a bed upon which the gypsum plugs can be poured. The liquid gypsum is highly non-viscous and would readily filter through the coarse material before setting up. Gypsum ($CaSO_4.2H_2O$) has been selected by LLNL as their plug material for two reasons. First, it is not highly exothermic while curing, and hence, the generated heat will not adversely affect the data cable impedance characteristics. Secondly, gypsum has a resilience in the presence of the nuclear blast shock wave and tends to flow with the ground motion while maintaining its integrity. Of the six gypsum plugs shown, only two are designated for gas blocking - the three 20-ft thick "thin" plugs (at 266.7 m, 350.5 m, and 438.9 m) are designed for attenuation of the gas-generated overburden pressure and to hold the chimney in place should the material below it collapse into the cavity. The deepest, 50-ft thick plug (at 566.1 m), identified as the stemming platform, is placed for gas containment

only if the device does not reach criticality; if criticality is reached, the deepest plug will probably be within the cavity radius, depending on yield. The top two plugs (each 40 ft thick) are emplaced for gas containment, the top being a redundant back-up to the working plug below it.

LANL uses two plugs that are TPE and one plug that is grout. All three are approximately 10 ft thick, and all are designed to be containment plugs. LANL had originally used Coal Tar Epoxy (CTE), but switched to TPE because the TPE was less expensive, easier to work with, and because of the potentially toxic effect of the fumes from the CTE. LANL views each of their fines layers as, at least, a partial gas block because of the addition of the bentonite, a thixotropic type of material that, under the pressure of the shock wave, flows with the earth motion while maintaining its integrity.

Finally, magnetite (Fe_3O_4) is placed about the test canister to extract the energy from the fast fission neutrons through inelastic collisions. The secondary gamma rays generated from this interaction have limited ranges of travel in the iron-rich environment and hence do not interfere with the prompt gamma rays resulting from the implosion. About the WP, this shielding effect is further enhanced by using boron-enriched stemming material.

In vertical shafts, keyways are seldom used because they are unnecessary; the structural integrity of the plugs is usually sufficient to hold the weight of stemming material above it. However, this does not preclude diligent efforts to ensure the quality of materials and procedures used in the backfilling process, especially the gypsum grout. During stemming, samples of the materials are taken at the drill hole surface every 10 minutes. Gravel, which has already been checked at the source, is examined for moisture content; the slurry is examined for density, sand content, and, after being allowed to set up, for breaking strength.

C.1.3.2.2 Equipment for Backfilling and Sealing Vertical Holes

No exotic equipment is used for backfilling the holes. The LLNL canister is hung in position on a drill pipe with the prefanned and gas-blocked cables also secured to the pipe; LANL uses two to four wire ropes of 2- to 3-in. in diameter, each with the data cables secured to the wire rope. A 250-ton/hr conveyor belt system moves the magnetite and coarse and fine stemming material from the loading hopper and drops it downhole while integrating the weight of the material moved to determine when sufficient material of each type has been emplaced. When a gypsum plug is to be poured, the gypsum is mixed uphole with retarders to extend the setup time from ~20 min to ~45 min and then pumped

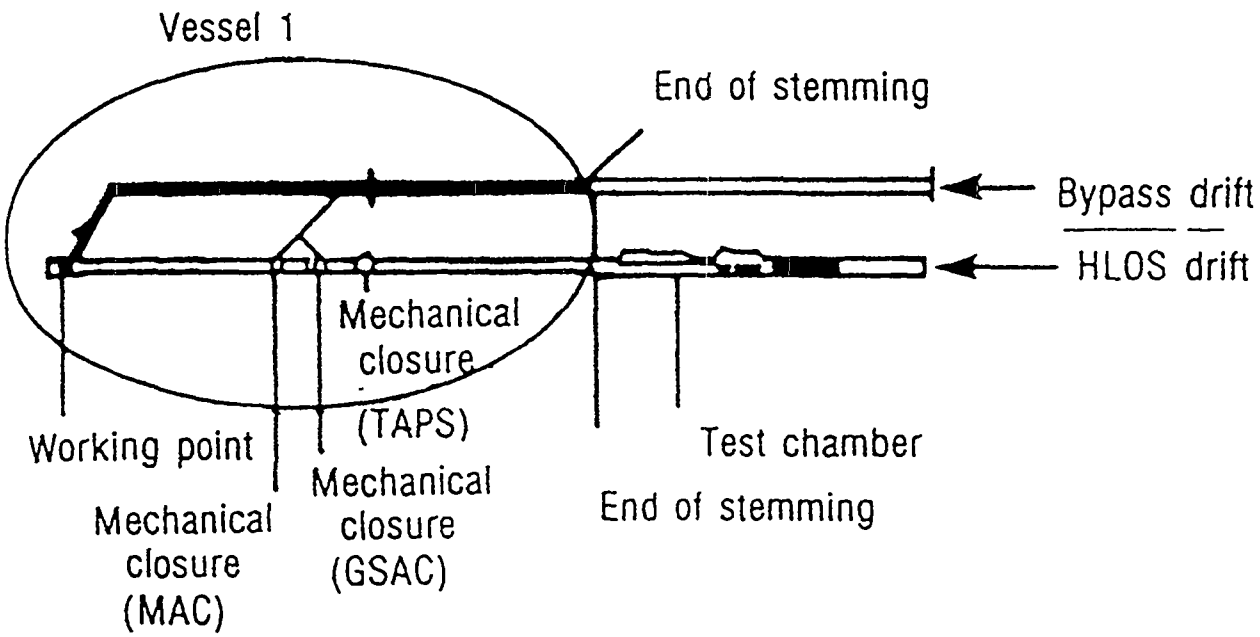
downhole via a pipe. Backfill material is placed downhole in the sequence and amounts as called for in the stemming plan. This operation requires about one week for a typical 1,500-ft shaft. LANL stemming is performed at a slower pace than LLNL. They also use a hopper and conveyor belt, but instead of using a weightometer on the conveyor belt to integrate the weight of material used, LANL has a load cell in the hopper that gives the weight of material in the hopper, which can then be translated to volumetric data.

C.1.3.3 Horizontal Tests

As mentioned earlier, the HLOS tests are performed in tunnel complexes at the NTS. The typical HLOS test consists of the main drift, a bypass drift, and the recording and signal-conditioning alcoves. The long HLOS pipes extend from 800 to 2,000 ft and taper from a few inches at the WP to about 10 ft at the test chamber. Together with the HLOS pipe in the main drift are mechanical closures, which are designed to shut off the pipe after the radiation created by the explosion has traveled down to the test chamber, but before the material from the blast can fly down the pipe and destroy the equipment. The typical closures are, as positioned from the WP, modified auxiliary closure (MAC), the gas seal auxiliary closure (GSAC), and a tunnel and pipe seal (TAPS). In some of the tests, a "muffler" was placed between the WP and the MAC to reduce flow down the pipe by allowing expansion and creating turbulence and stagnation. The MAC and GSAC are designed to withstand pressures up to 10,000 psi, while the TAPS is designed to withstand pressure up to 1,000 psi and temperature up to 1,000°F. In lower yield tests, the fast-acting closure (FAC) replaces the MAC and can withstand pressures of 30,000 psi. Figure C.1-4 represents a schematic of what is discussed above.

After emplacing the HLOS pipe, the area around the pipe is filled with grout, so that the only clear pathway between the explosion and the test chamber is internal to the pipe. Typically, there are three types of grout in the main drift: a rock-matching grout near the WP is placed to develop a uniform stress field, hence uniform cavity development, around the WP; a weak (superlean) grout is then placed before the MAC or FAC for extrusion and sealing of the HLOS; and a strong grout or concrete is placed from the end of the superlean grout to the end of stemming to stop the superlean grout flow. Following the sealing of the HLOS, the bypass drift is sealed in a comparable manner with a rock-matching grout, a superlean grout, and a high-strength grout.

Outside of Vessel I, plugs are constructed to separate each vessel. Typically, these plugs are constructed of high-strength grout and concrete 10 to 30 ft thick. The side of the Vessel II



Key: GSAC = gas seal auxiliary closure; MAC = modified auxiliary closure; TAPS = Tunnel and pipe seal

Figure C.1-4
Schematic of Vessel I

plugs facing the WP are made of steel. Vessel III plugs are made of massive concrete. In addition to the containment vessels, there is a gas seal door at the entrance of the tunnel. In the following section, four HLOS tests are discussed and specifics on the tests given to illustrate the preceding discussions.

MIDAS MYTH (MILAGRO), 2/15/84

MIDAS MYTH is an example of a high-yield test bed. It was a typical example of an HLOS in Rainier Mesa through and including MIGHTY OAK in April 1986. The 984-ft HLOS was located in T-tunnel in the U12t.04 drift 1,184 ft below the surface of the mesa. There were only two features that made it different than a typical high-yield test: The first was MILAGRO, a large LANL experiment that was added to the MIDAS MYTH event. The second was the unusually large signal conditioning and ROSES drift near the end of the pipe, which came to be called the "ballroom" (see Figure C.1-5).

The HLOS pipe string included a muffler, a MAC, a GSAC, and TAPS (see Figure C.1-5). The stemming diagram is illustrated in Figure C.1-6. The differences between the low- and high-yield test beds are not in the specific types of rock-matching or high-strength grout, but in the length of the stemming columns and the location of the mechanical closures. The same concept is used in both; that is, rock-matching grout near the WP for symmetrical cavity formation, a weak grout for extrusion and sealing the HLOS, and a stemming barrier of strong grout or concrete to stop the grout flow.

Overlapping pressure grout rings were drilled and pressure-grouted in the pillar zone between the TAPS cable gas block and the bypass cable gas block alcove at RS 135.6 m (445 ft). The grout rings overlapped in the pillar zone for a distance of nearly 10 ft. These grout rings are designed to fill any fracture area in the tuff surrounding the drifts and seal any possible gas leak path outside the stemming area.

Vessel II was the standard region surrounding all tunnel volumes inside the three drift protection plugs (DPP) (see Figures C.1-5 and C.1-6). The plugs were pre- and post-structure pressure-grouted during the HUSKY PUP event in U12t.03 in 1975. All penetrations through the plugs were installed to meet overburden plug design criteria, 1,000 psi and 1,000°F. This suggests that some plugs experience multiple ground shock events.

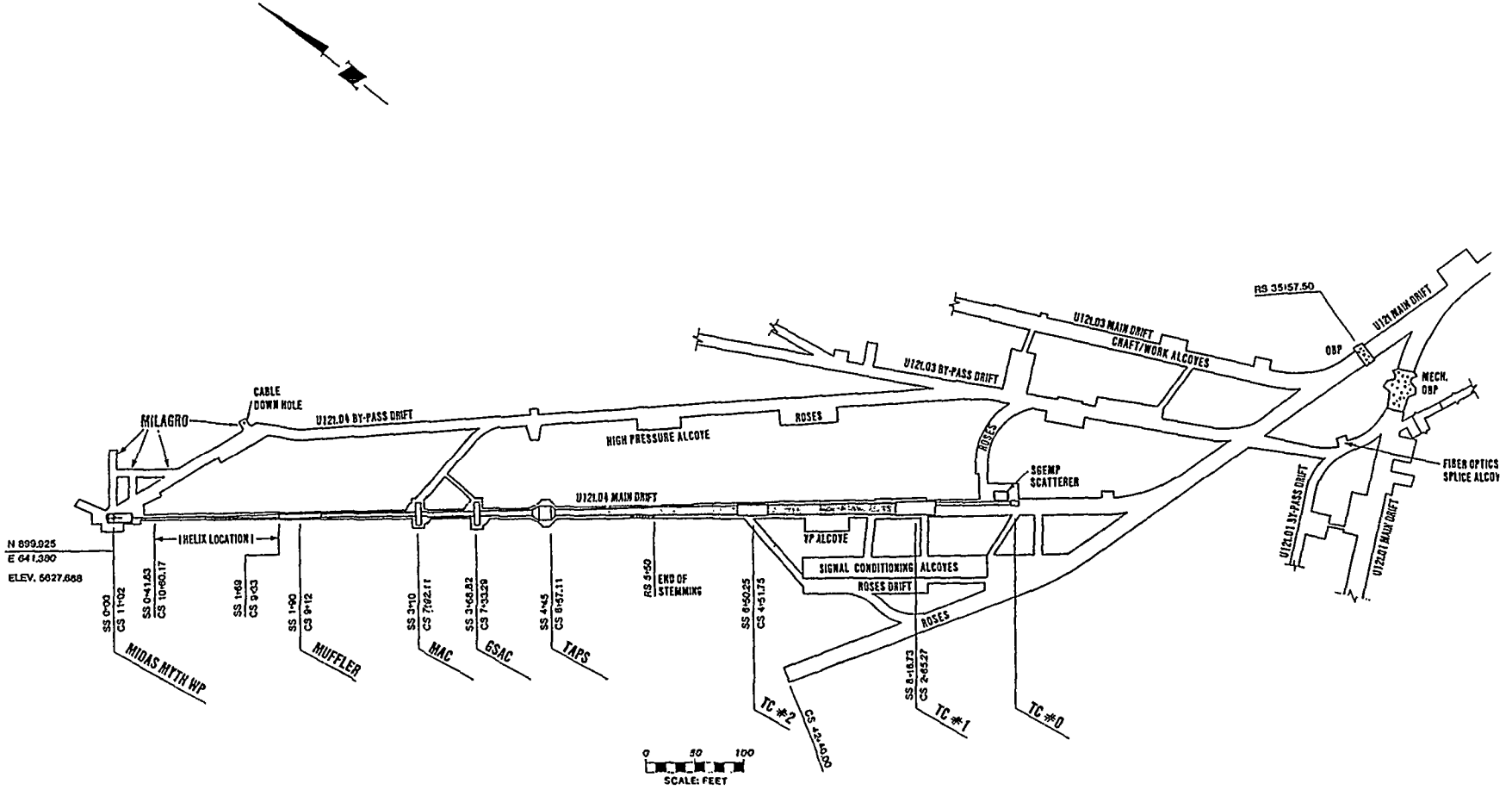


Figure C.1-5

Tunnel and Pipe Layout (Note location of Overburden Plug, also called Drift Protection Plugs)

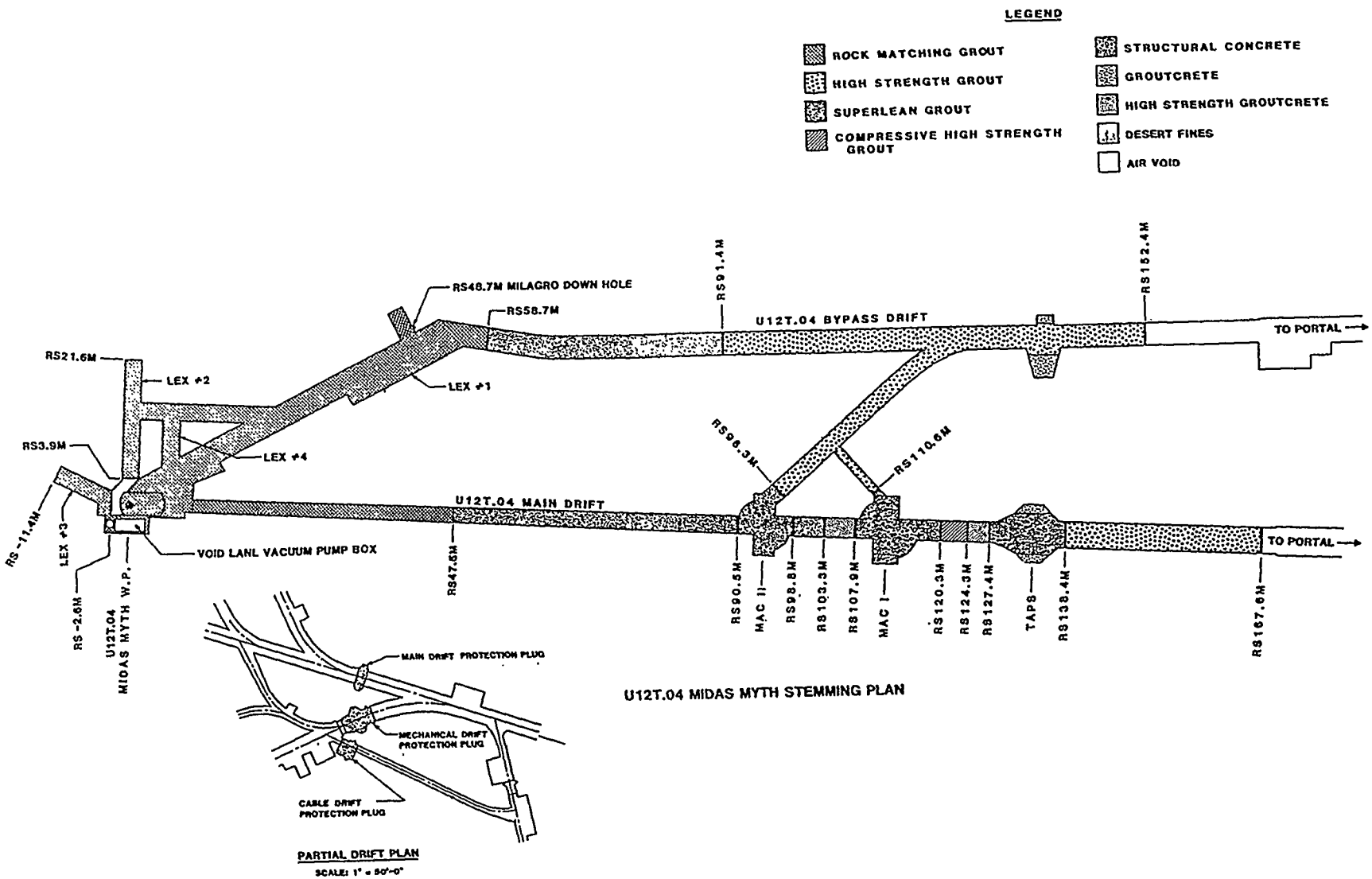


Figure C.1-6
MIDAS MYTH Stemming Plan

There were no special instrumentation protection provisions in this event. What you do not see in the MIDAS MYTH design is any FDPPs or crosscut plugs. After the serious leak out of Vessel I in MIGHTY OAK, this protection was provided, but MIDAS MYTH was conducted more than 2 years prior to MIGHTY OAK.

Vessel III was the standard region of all tunnel volume inside the gas seal plug (GSP). T-tunnel has only one main access tunnel, so there is only one GSP. The design pressure and temperature for the GSP are 500 psi and 500°F, respectively, as in N-tunnel. The GSP is 2,620 ft from the MIDAS MYTH WP and 3,010 ft from the tunnel portal. The plug was pre- and post-structure pressure-grouted for the HUSKY PUP event. The penetrations through these plugs are handled in much the same way as the DPPs. There is also a gas seal door in T-tunnel. As in N-tunnel, it is not considered part of the current containment design, but is closed as an added safety feature.

Following the event, it was found that the invert had heaved and smashed the ROSES against the back of the alcove. This is mentioned to illustrate the high ground shock experienced by the stemming design and plugs.

Another unique feature was the MILAGRO cable hole extending from 48.7 m (160 ft) from the WP at the tunnel location to the surface of the mesa above. The total length of stemming was 351.1 m.

Except for the top 72 ft, the hole is completely filled with grout. Cable gas blocks, in nonfactory-gas-blocked cables, were located at about 300 and 745 ft below the surface. A coal tar plug at the surface was the last line of defense against low-pressure boundary leaks in the cable bundle.

MIGHTY OAK, 4/10/86

MIGHTY OAK was the fifth event in T-tunnel and was conducted in the U12t.08 drift. Figure C.1-7 illustrates the MIGHTY OAK tunnel and pipe layout. MIGHTY OAK was the last of the high-yield test bed containment designs and was very similar to the two preceding shots. There were three differences between MIDAS MYTH and MIGHTY OAK. MIGHTY OAK had no large "ballroom" instrumentation alcove as did MIDAS MYTH. There was a large volume of instrumentation alcove space, but it was divided into two alcoves with large rock pillars between them and the main drift. The second difference was

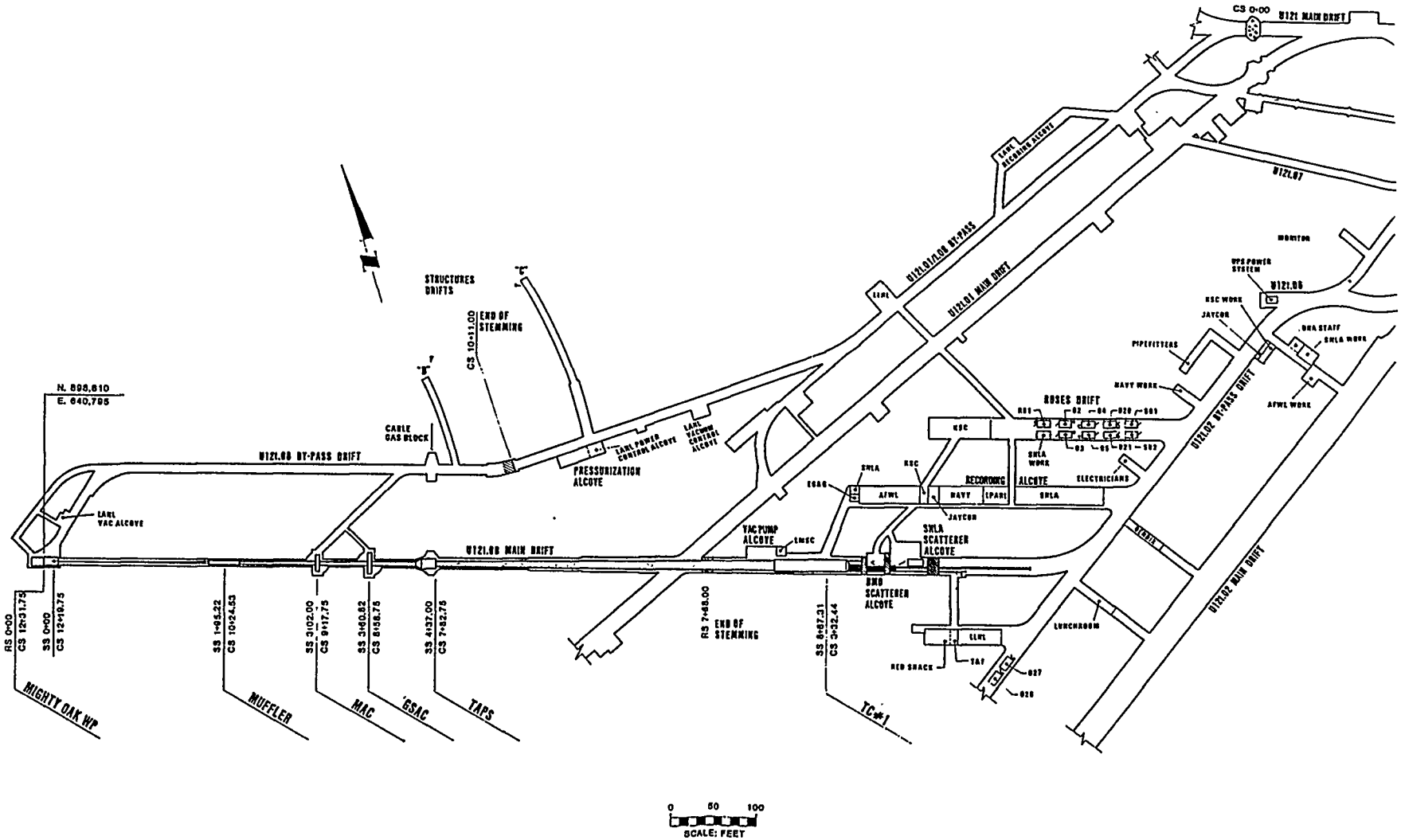


Figure C.1-7
Tunnel and Pipe Layout for MIGHTY OAK

the presence of the structures drifts. Several scale models of deep-buried structures were tested for ground shock response. The third was that an old drift, U12t.01, intersected the main pipe drift of MIGHTY OAK. A longer stemming column had to be used to seal off this drift. This may be the source of the radioactivity that reached the north side of the ROSES alcove at about the same time that radioactivity reached the south side of the recording alcove, presumably from the end of the HLOS pipe.

The DNA standard three-vessel containment design was used for MIGHTY OAK. Included within Vessel I was the standard region of the working point, cavity zone, and stemmed drifts. Structures drift B was inside the stemmed area, so it was included in Vessel I. The mechanical closures were the same as for MIDAS MYTH, the MAC, GSAC, and the TAPS, and they were located at the same distance from the WP. The stemming was very similar to past events except for the longer stemming column to seal off the U12t.01 drift (see Figures C.1-7 and C.1-8). Overlapping pressure grout rings were drilled and pressure-grouted in the pillar zone between the TAPS cable gas block and the bypass drift cable gas block at RS 135.6 m (445 ft). The DNA standard pressure grouting technique was used. The grout rings overlapped in the pillar zone for a distance of nearly 10 ft. The size of the main test drift varies from about 8.5 ft wide by 8.5 ft high near the WP to 18 ft wide by 16 ft high at the intersection of the U12t.01 drift. The bypass drift is about 12 ft wide by 11 ft high. The distance from the centerline to centerline of the main and bypass drifts is 109 ft.

Vessel II for MIGHTY OAK was a little different than for MIDAS MYTH. The MIGHTY OAK main drift intercepted the U12t.01 and U12t.02 drifts; therefore, additional DPPs were constructed in these drifts. There were a total of six DPPs on MIGHTY OAK (see Figure C.1-9). All six plugs were constructed to meet the standard overburden plug (OBP) design criteria of 1,000 psi and 1,000°F.

Figure C.1-10 is a photograph (Photo #118-3) of the placement of a low-temperature rock matching (LTRM) grout. Figure C.1-11 illustrates the location of the pour related to the WP. The camera is at station CS15 + 70/4 ft from the right rib looking at GZ. The right rib intersects the reaction history bulkhead at about CS15 + 95 ft. The LTRM grout represents the seventh lift at "submarine," a large LANL experiment. This lift was 8,328 ft^3 of grout. The total of all the lifts was 27,334.7 ft^3. The rebar in the left portion of the photo is a rebar brace for prestemming. The pipes in the back are fill and bleed lines, and the pours were placed against a metal bulkhead.

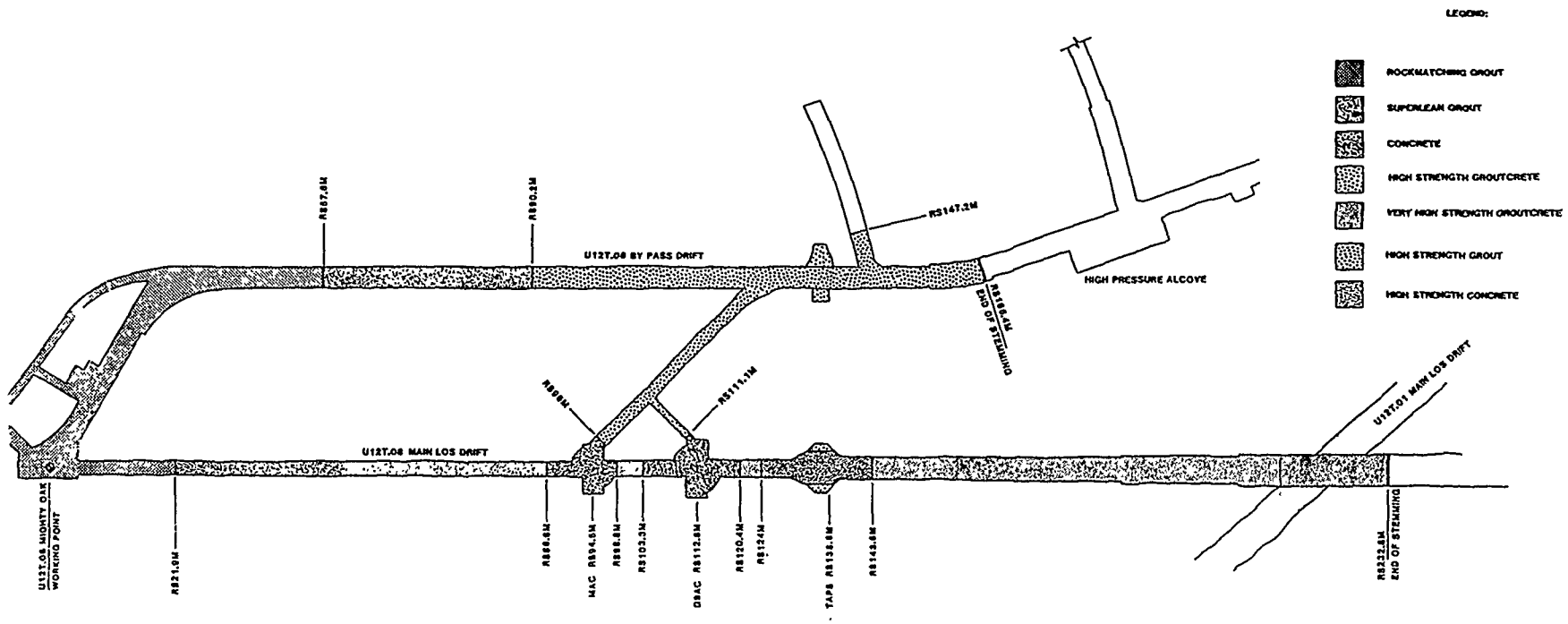


Figure C.1-8
MIGHTY OAK Stemming Plan

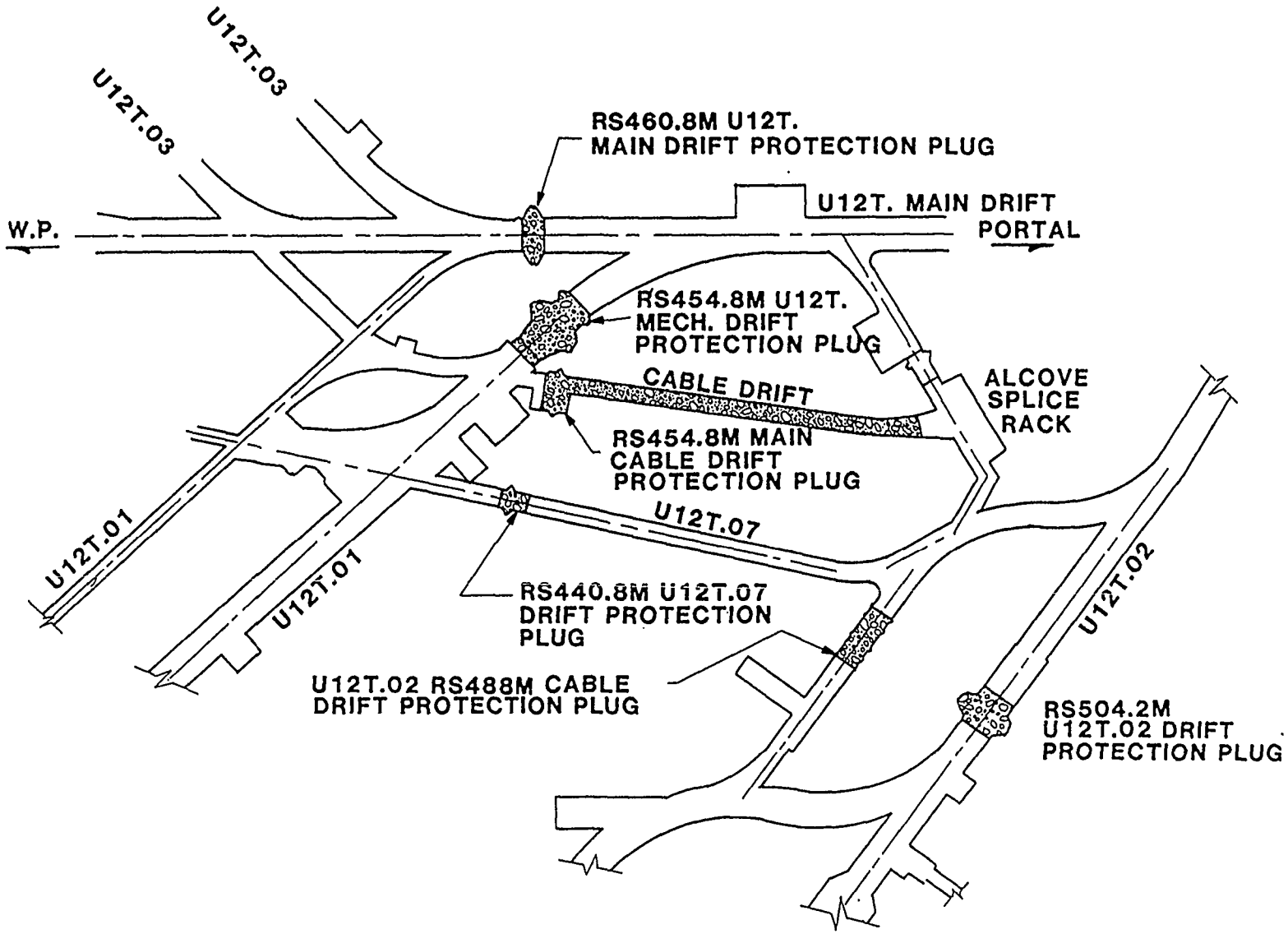


Figure C.1-9
Location of Drift Protection Plugs

Figure C.1-10
Photograph of Placement of a Rock-Matching Grout Near MIGHTY OAK Working Point
(Photo courtesy of DNA/Johnson Controls)

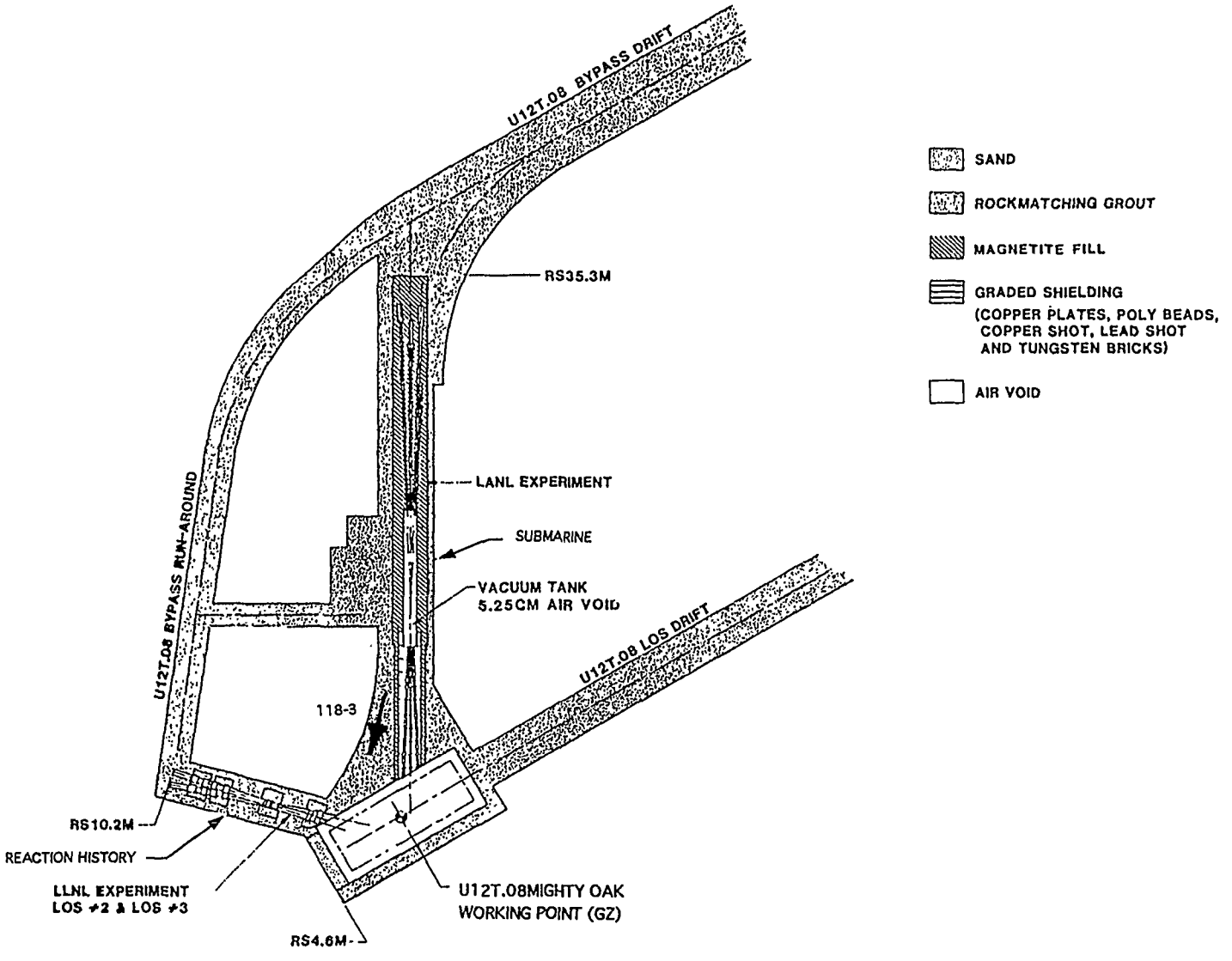


Figure C.1-11

MIGHTY OAK Working Point Area Stemming - Location of Figure C.1.10 Photograph (#118-3, Looking to GZ)

MIDDLE NOTE, 3/18/87

MIDDLE NOTE was a low-yield test bed containment design. It was located in the U12n.21 drift in the N-tunnel complex. The MIDDLE NOTE complex consisted of the main drift containing the LOS pipe, the bypass drift, and the recording and signal-conditioning alcoves in the U12n.11 and U12n.14 drifts. The LOS pipe string consisted of an 800-ft vacuum line-of-sight pipe with three experimental test chambers and four mechanical closures. The closures included a Fast Acting Closure (FAC), GSAC, TAPS, and a Debris Barrier System (DBS). The DBS was installed in the MIDDLE NOTE LOS pipe to divert any gas flowing in the pipe to an isolated 500,000 ft^3 of the tunnel complex that is void of all instrumentation and isolated from the remainder of the tunnel complex by FDPP, shown in Figure C.1-12.

The stemming design is similar to that in the other shots. In the vicinity of the WP, there is a rock-matching grout. Next is a superlean grout to the FAC, followed by either a very high-strength groutcrete or concrete.

Within Vessel II, the instrumentation in the tunnel complex is protected primarily by the LOS pipe closures and the stemming in the main and bypass drifts. Should these be bypassed, FDPP were placed in the bypass drift beyond the end of stemming and on the portal side of the experiment station in the main drift to provide further isolation of the main tunnel complex. Steel-faced grout plugs were placed in the crosscut drifts, outside of stemming, between the main and bypass drifts to isolate the rest of the tunnel from the main drift. At the edge of Vessel II are the DPP. These plugs have various penetrations from 9.5- by 12.1-ft trainways to cable, water, and gas sampling lines. After the trainway is filled with concrete, a manway with turn-tube closures provides late preshot and reentry personnel access.

The following photographs help to illustrate the construction techniques as well as postshot performance of the grouts and concrete. Figure C.1-13 (Photo 82-15) represents the fourth lift of a low-temperature rock-matching grout designated LTRM-1. The size of this pour is 157.4 yds^3, and the total pour was 619.2 yd^3. The operation is early in the button-up stemming operation, looking toward the WP from about 60 ft away. The size of the drift is about 13 by 12 ft.

Figure C.1-14 (Photo 131.4) illustrates the deformable nature of the superlean grout. The superlean (left side of photo) is squeezed into fractures of the native rock (left-center of

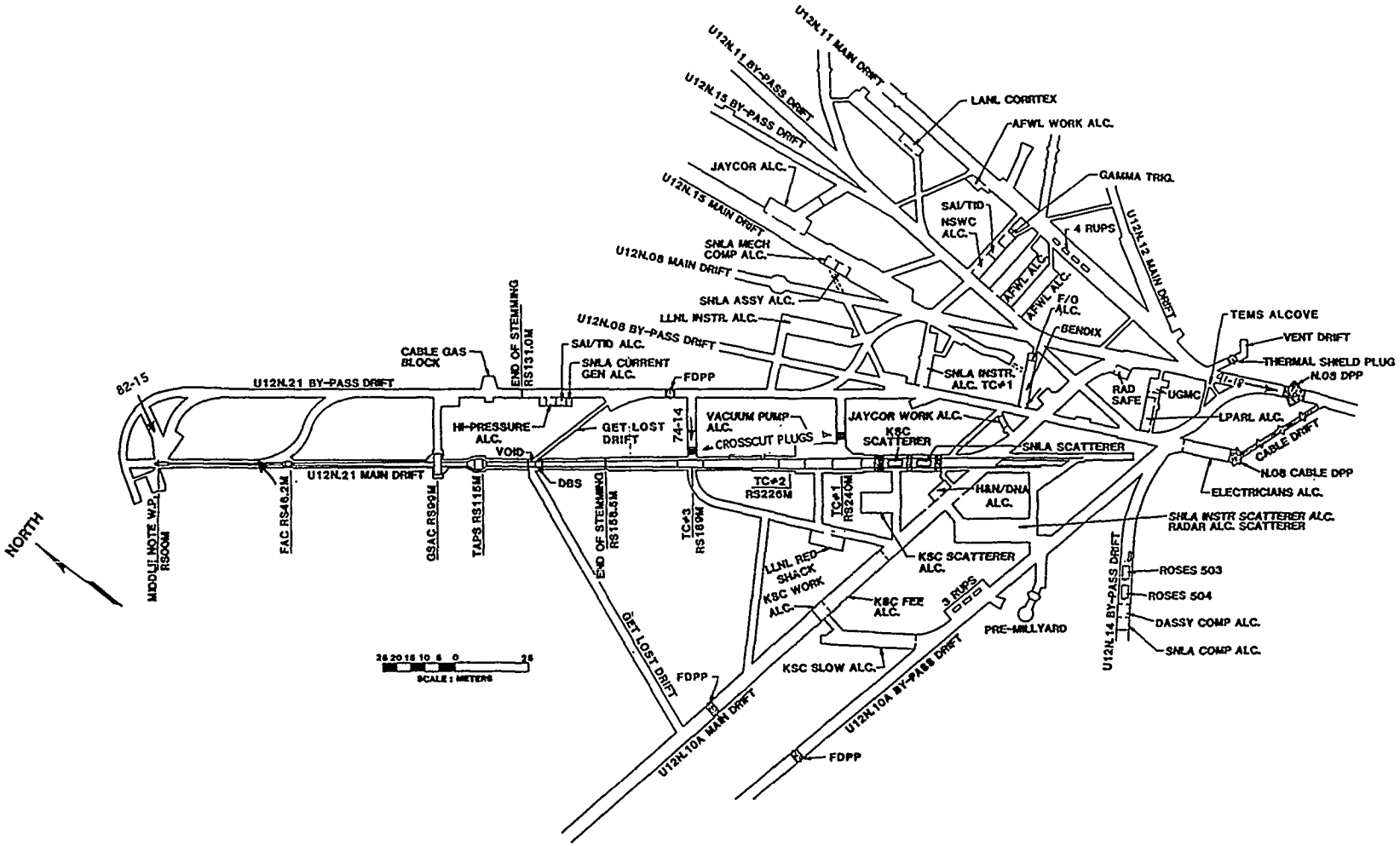


Figure C.1-12
MIDDLE NOTE Horizontal Line-of-Sight Pipe Closures

Figure C.1-13

Fourth Lift of a Low-Temperature Rock-Matching Grout Pour for MIDDLE NOTE

(Photo courtesy of DNA/Johnson Controls)

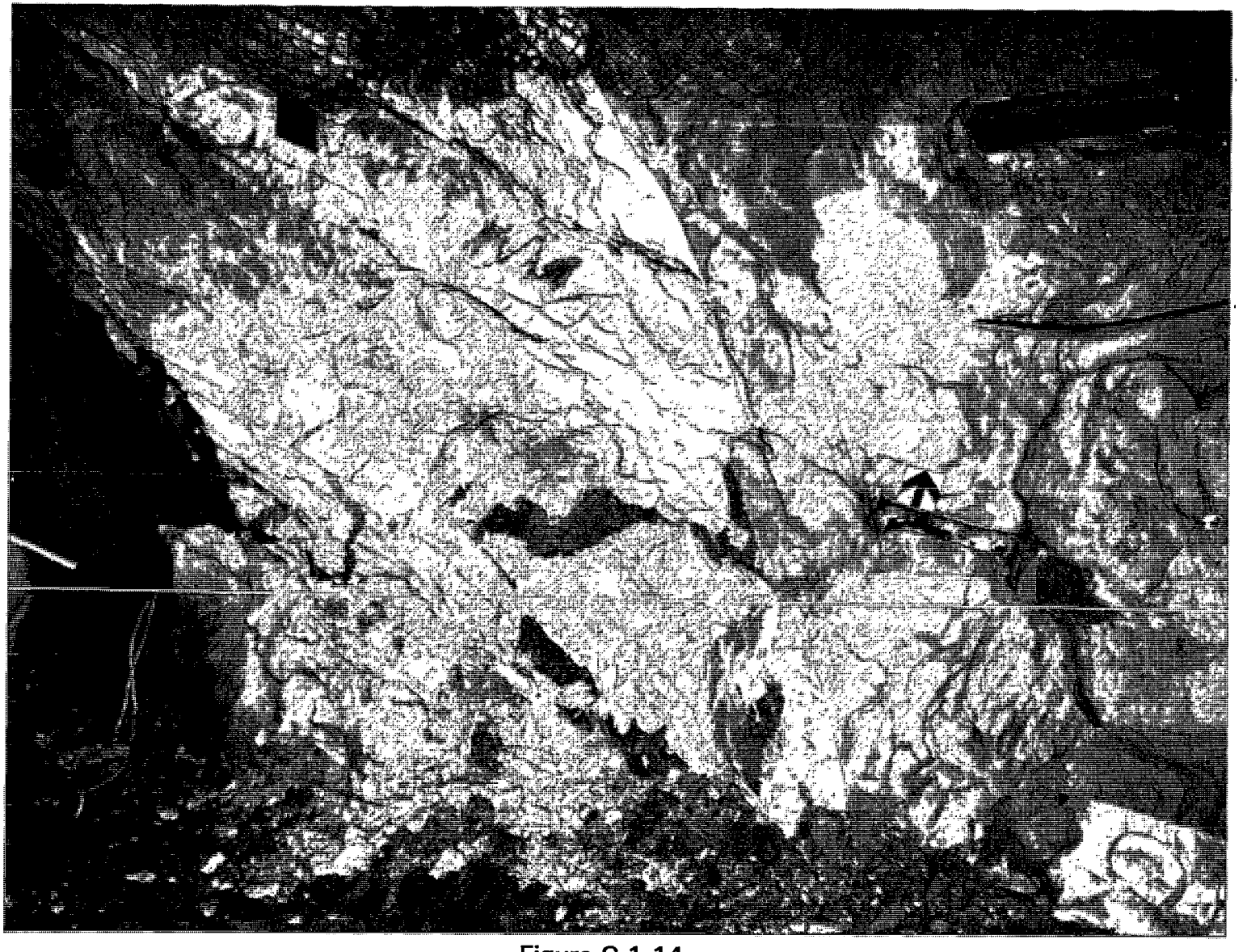


Figure C.1-14

Penetration of Superlean Grout Data Fracture on Working Point Side of Fast-Acting Closure

(Photo courtesy of DNA/Johnson Controls)

photo). At both the extreme left and right of the photo are higher-strength rock-matching grouts.

Figure C.1-15 (Photo 74-14) shows the face of a crosscut bulkhead outside of Vessel I. As with many bulkheads, penetrations are needed for a variety of reasons. The Vistanex lines are the two 1-in. lines at the bottom right of the door. (Vistanex is used as gas seal around cables.) The 1-in. lines with valves are pressure grout lines. Some of the larger lines are the fill and bleed lines. A 42-in.-diameter steel pipe provides a crawlway.

Figure C.1-16 (Photo 82-3) was taken with the "N" extension gas seal plug near the portal of N-tunnel. The larger lines are fill and bleed lines. The small lines on the perimeter are pressure grout lines. The large pipe is a crawlway to gain early posttest access to tunnel complex. The lower two openings are for personnel access, while the top window is for viewing the progress of the pour. This photo is inside plug (before pour), looking toward the portal.

MINERAL QUARRY, 7/25/90

The MINERAL QUARRY test was conducted in the U12n.22 drift of N-tunnel, the next HLOS test in that tunnel after MIDDLE NOTE. MINERAL QUARRY had a higher yield than MIDDLE NOTE, resulting in stemming design differences. Mechanical closures included a FAC, GSAC, and TAPS. There was no DBS on MINERAL QUARRY. The mechanical closures were moved further from the zero room to accommodate the higher yield.

The stemming diagram is given in Figure C.1-17. A corresponding table (Table C.1-1) is also given to illustrate the complexity of a stemming operation. Not shown is the low-temperature rock-matching grout (LTRM-1), 1,000 psi, 120°F maximum curing temperature. Overlapping pressure grout rings were drilled and pressure-grouted in the pillar zone between the TAPS cable gas block at RS 182.2 m (598 ft) and the bypass drift cable gas block alcove at RS 186 m (610 ft), see Figure C.1-18. The grout rings overlapped by more than 13 ft. These grout rings are designed to fill any fracture area in the tuff surrounding the drifts and seal any possible gas leak path outside the stemming area.

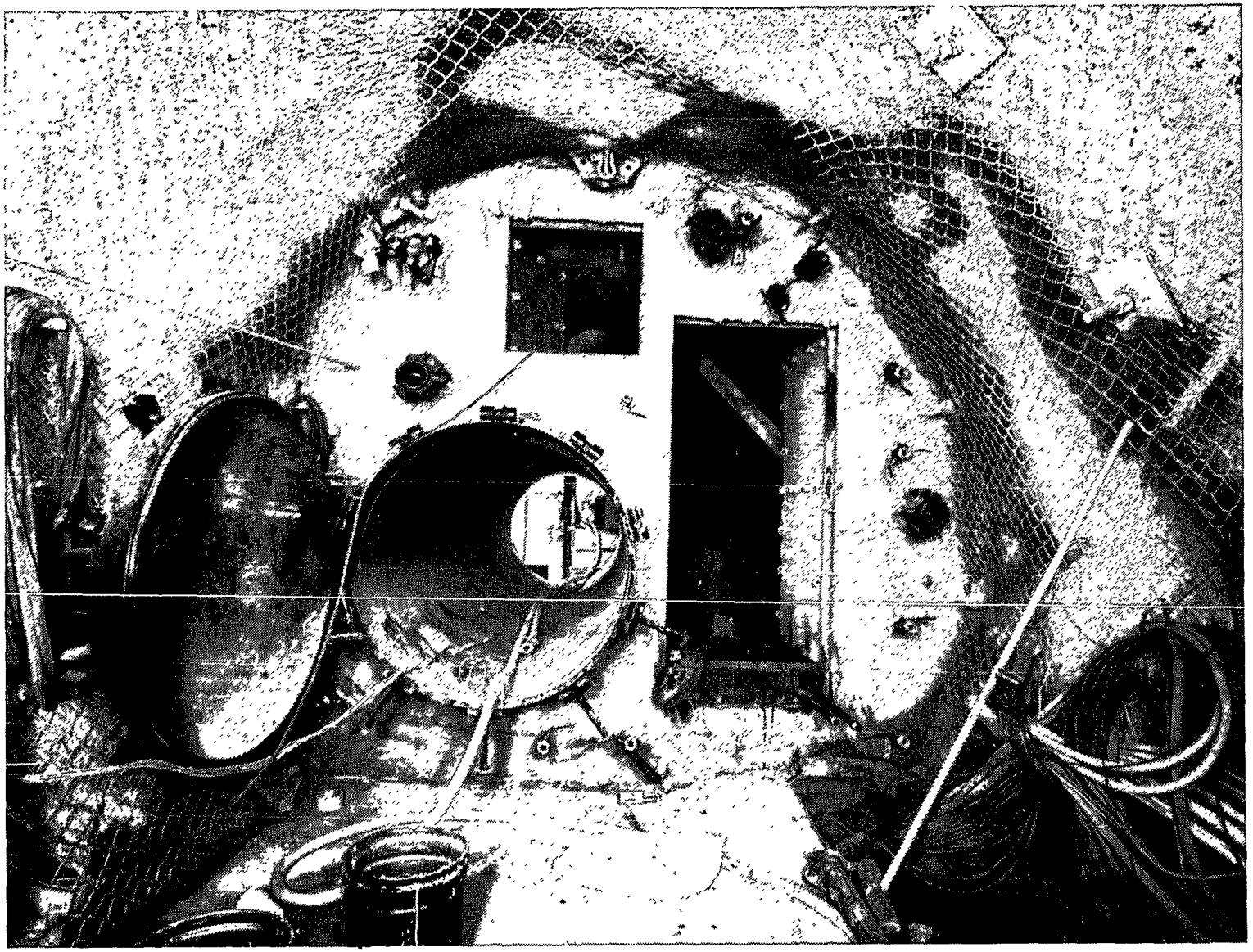


Figure C.1-15
Crosscut Bulkhead #3
(Photo courtesy of DNA/Johnson Controls)

Figure C.1-16
Inside N-Tunnel Gas Seal Plug Prior to Pour
(Photo courtesy of DNA/Johnson Controls)

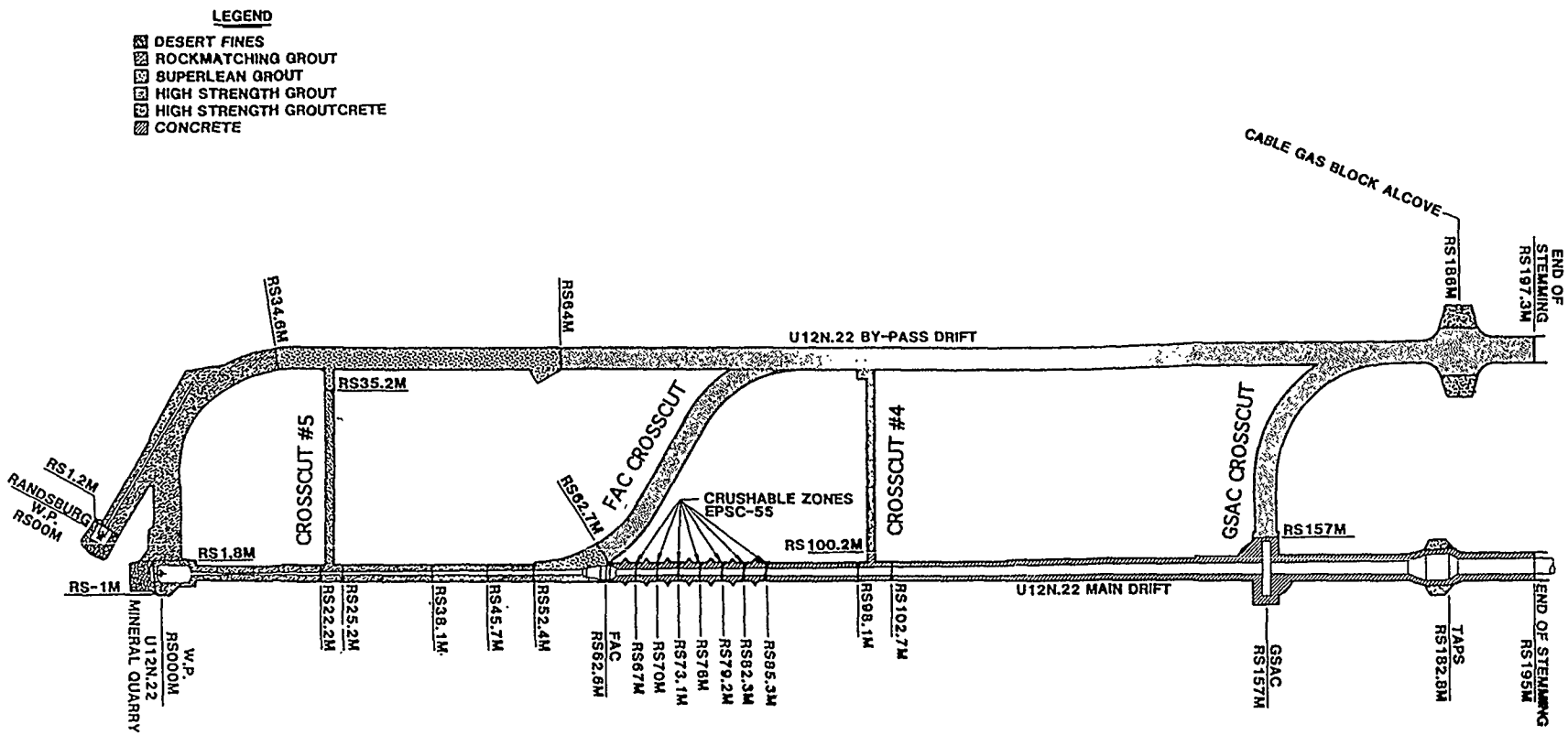


Figure C.1-17
Stemming Diagram for MINERAL QUARRY

Table C.1-1
Simplified Stemming Description*

Designation	Location	Description
	MAIN DRIFT	
ME8-05	WP to RS 22.2 m	Strong Rock-Matching Grout [3,000 psi @ 7d; ChemComp Slump 11" +]
HLRM (CC)	RS 22.2 m to RS 25.2 m	Rock-Matching Grout [1000 psi @ 28d; ChemComp; Slump 11.5"]
DSRM-2	RS 25.2 m to RS 38.1 m	Rock-Matching Grout [1,000 psi @ 28d; Class A; Slump 11.5"]
HSSL 50/50 (RED)	RS 38.1 m to RS 45.7	Superlean Grout [200-300 psi @ 7d; Class A; Slump 11" +]
DSRM-2	RS 45.7 m to RS 52.4 m	Rock-Matching Grout [1,000 psi @ 28d; Class A; Slump 11.5"]
HSSL 50/50 R-4	RS 52.4 m to RS 62.6 m	Superlean Grout [200-300 psi @ 7d; Class A, Slump 11"+]
MZC (CC) Basalt	RS 62.6 m to RS 195 m (end of stemming)	Concrete [14,000 psi @ 8d; ChemComp; Slump 11.5"]
	CROSSCUTS	
HLRM (CC)	Crosscut #5	Rock-Matching Grout [1,000 psi @ 28d; ChemComp; Slump 11.5"]
MZHSG (CC)	FAC Crosscut, Crosscut #4, GSAC Crosscut	High-Strength Grout [3,000 psi @ 3d; ChemComp]
	BYPASS DRIFT	
NA	WP to RS 1.8 m	Strong Rock-Matching Grout
HLRM (CC)	RS 1.8 m to RS 34.6 m	Rock-Matching Grout [1,000 psi @ 28d; ChemComp, Slump 11.5"]
HSSL 50/50	RS 34.6 m to RS 64.0 m	Superlean Grout [200-300 psi @ 7d; Class A; Slump 11" +]
MZHSG (CC)	RS 64.0 m to RS 197.3 m (end of stemming)	High-Strength Grout [3,000 psi @ 3d; ChemComp]
	KEYWAY	
HLHSGC (CC)	Bypass Cable Gas Block Alcove, TAPS	High-Strength Groutcrete

*Table does not include foundation material or pressure-grouting mixtures

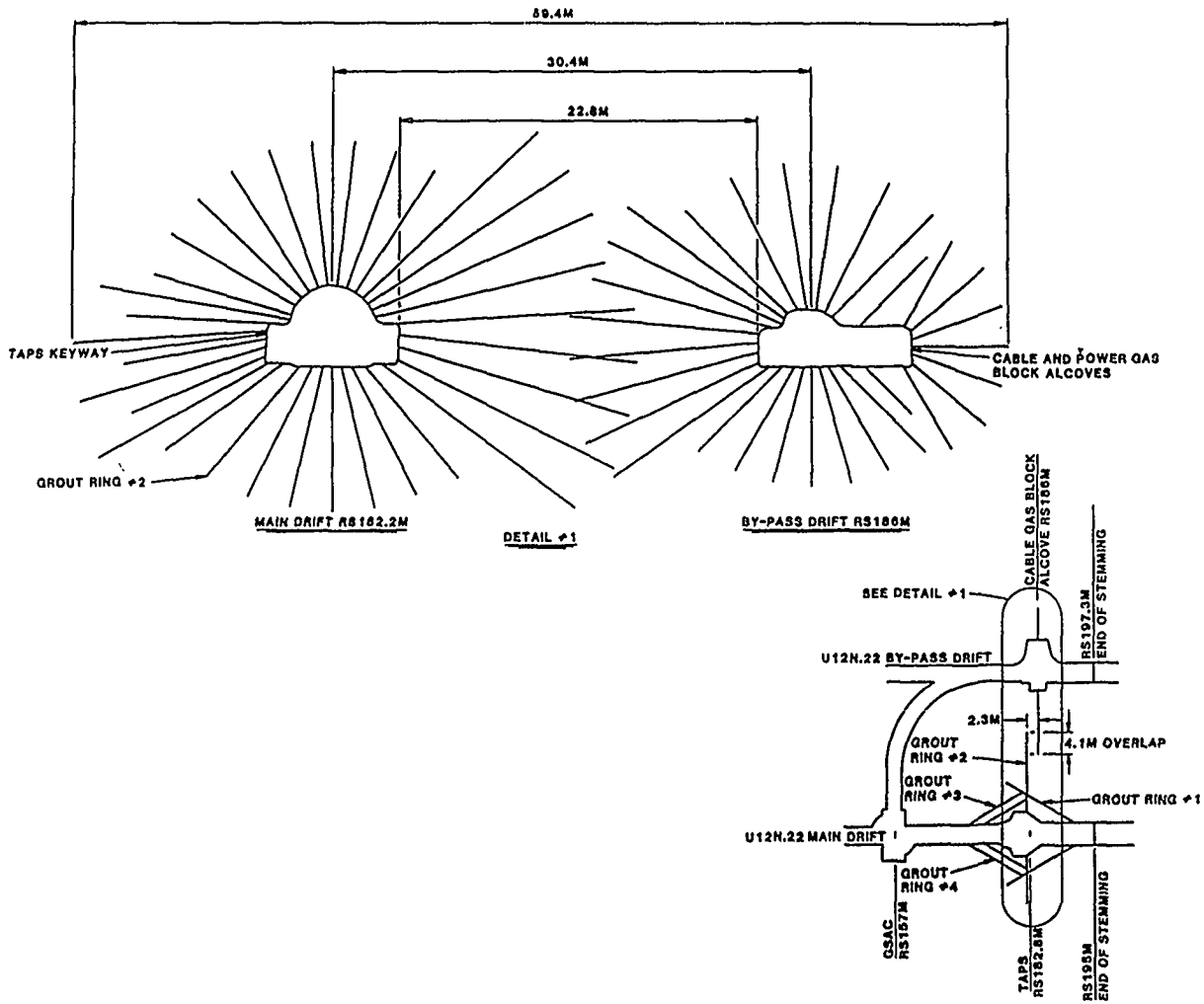


Figure C.1-18

MINERAL QUARRY Detail of Overlapping Pressure Grout Rings

C.1.3.4 Cavity Tests

Cavity events have no line-of-sight pipe, except for small diagnostic pipes that are fully stemmed, and no mechanical closures. They are designed and evaluated for containment purposes as a series of three nested containment vessels in exactly the same manner as HLOS events. Vessel I is the WP region, mined cavity, nuclear cavity, and the stemmed access drift. Vessel II is all the tunnel volume inside the DPPs, and Vessel III is all the tunnel volume inside the GSP.

MILL YARD, 10/9/85

MILL YARD was conducted in the U12n.20 drift on N-tunnel. Containment of the MILL YARD event was achieved through the use of drift stemming columns. The excavations consisted of an 11 m (36-ft) hemispherical cavity. A cylindrical invert was mined 16.42 ft below the hemisphere. This cylindrical region was completely filled with alluvium from the Yuma, Arizona, area. This was done to facilitate direct comparisons between the results of HE tests done in the Yuma area and the results of the nuclear MILL YARD event. Four 12-ft-wide by 10-ft-high drifts entered the cavity at various elevations. The cable drift entered the cavity at the bottom of the cylindrical excavation. The lower drift entered the cavity at the elevation of the top of the alluvium. The middle drift entered the cavity 12 ft above the top of the alluvium. The upper drift entered the cavity 24 ft above the top of the alluvium (see Figure C.1-19).

The DNA standard three-vessel containment design was used for MIGHTY OAK.

Figure C.1-19 also shows the stemming plan. The main access drift out to RS 24.3 m (80 ft) was filled with high-strength groutcrete, to RS 27.1 m (89 ft) with high-strength grout, and to RS 29.4 m (96.4 ft) with a concrete anchor plug. The plugs for the Vessel I stemming consisted of five concrete plugs located in the cable drift, lower drift, middle drift, the upper drift, and the main access drift. All of the plugs contained a Vistanex barrier that covered the full tunnel cross section (rock to rock). The three plugs that directly faced the cavity were "self-sealing plugs." The "self-sealing plug" was tested on an HE cavity shot prior to MILL YARD and proved to be very effective.

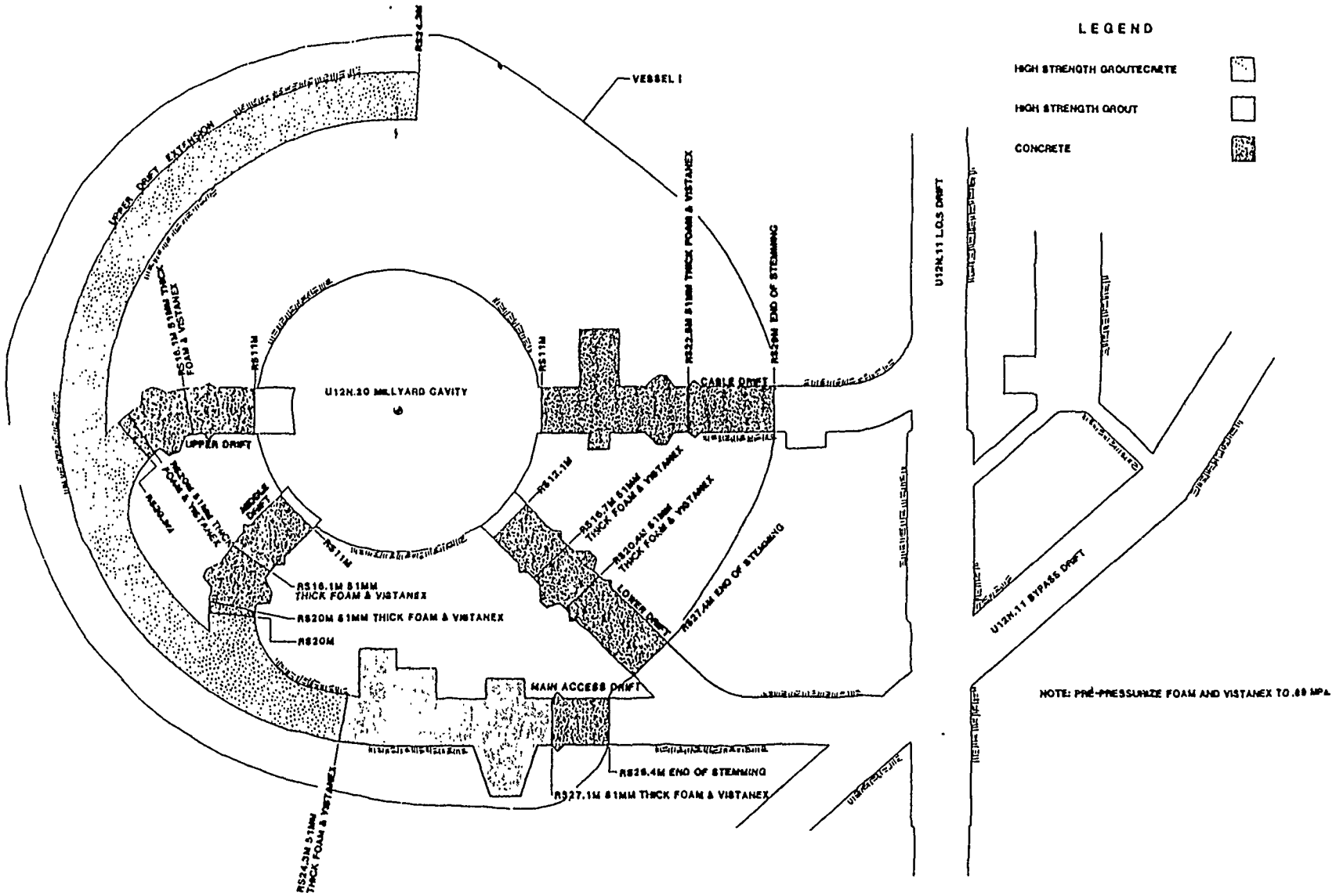


Figure C.1-19
U12n.20 MILL YARD Vessel I Outline

MISTY ECHO, 12/10/88

The MISTY ECHO event was a fully coupled nuclear explosion in a hemispherical cavity of 11-m (36-ft) radius. It was constructed in N-tunnel in the U12n.23 drift, located south of the N-tunnel extension. The yield of MISTY ECHO was about the same as the typical high-yield HLOS. The vertical depth of burial was 400.2 m (1313 ft). It had a main access drift that was similar to a bypass drift used in a typical HLOS event. At about 60 m (197 ft) from the WP, this drift splits into two access drifts into the cavity. The drift at tunnel level entered the base of the cavity, and the other drift ramps up to enter the cavity halfway between the top and the floor. The main drift was completely stemmed following the construction practices used successfully on previous bypass drifts.

The main access drift stemming was 244 m (800 ft) long, or over six postshot cavity radii, in length (see Figure C.1-20). It is longer, when scaled for yield, than a typical HLOS event. Prestemming in the MISTY ECHO section below the mined hemispherical cavity was completed after a majority of the cables and experiments were installed. Button-up stemming in the U12.n23 main and upper access drifts was completed after device insertion. The cylindrical section below the WP was stemmed with high-strength rock-matching grout to RS 22.8 m (75 ft) in the main access drift and RS 21.3 m (70 ft) in the upper access drift. This is followed by rock-matching grout in the main and upper access drifts. There were sections of superlean in both drifts and another section of rock-matching grout. Superlean grout was again used in both drifts past their junction and beyond to RS 61 m (200 ft) in the main access drift. From this point, rock-matching, superlean, and high-strength grout alternated out of RS 159.4 m (523 ft). A high-strength grout anchor plug completes the access drift stemming from RS 159.4 m (523 ft) to the end of stemming at RS 244 m (800 ft). The cavity remained an air-filled volume of 2,788 m^3 (98,445 ft^3).

Penetrations through the MISTY ECHO stemming were similar to those of the bypass drift of an HLOS event and were gas-blocked in essentially the same way. Near the cavity, all 1,175 instrumentation and power cables were interrupted in epoxy-filled J-Boxes. This was not mentioned before for HLOS events, but it was frequently done either in the bypass drift or crosscuts inside stemming. The 921 cables that exited the stemming region were routed down the main access drift. The 521 nonfactory gas-blocked cables went through steel Vistanex gas block boxes at the main gas block alcove.

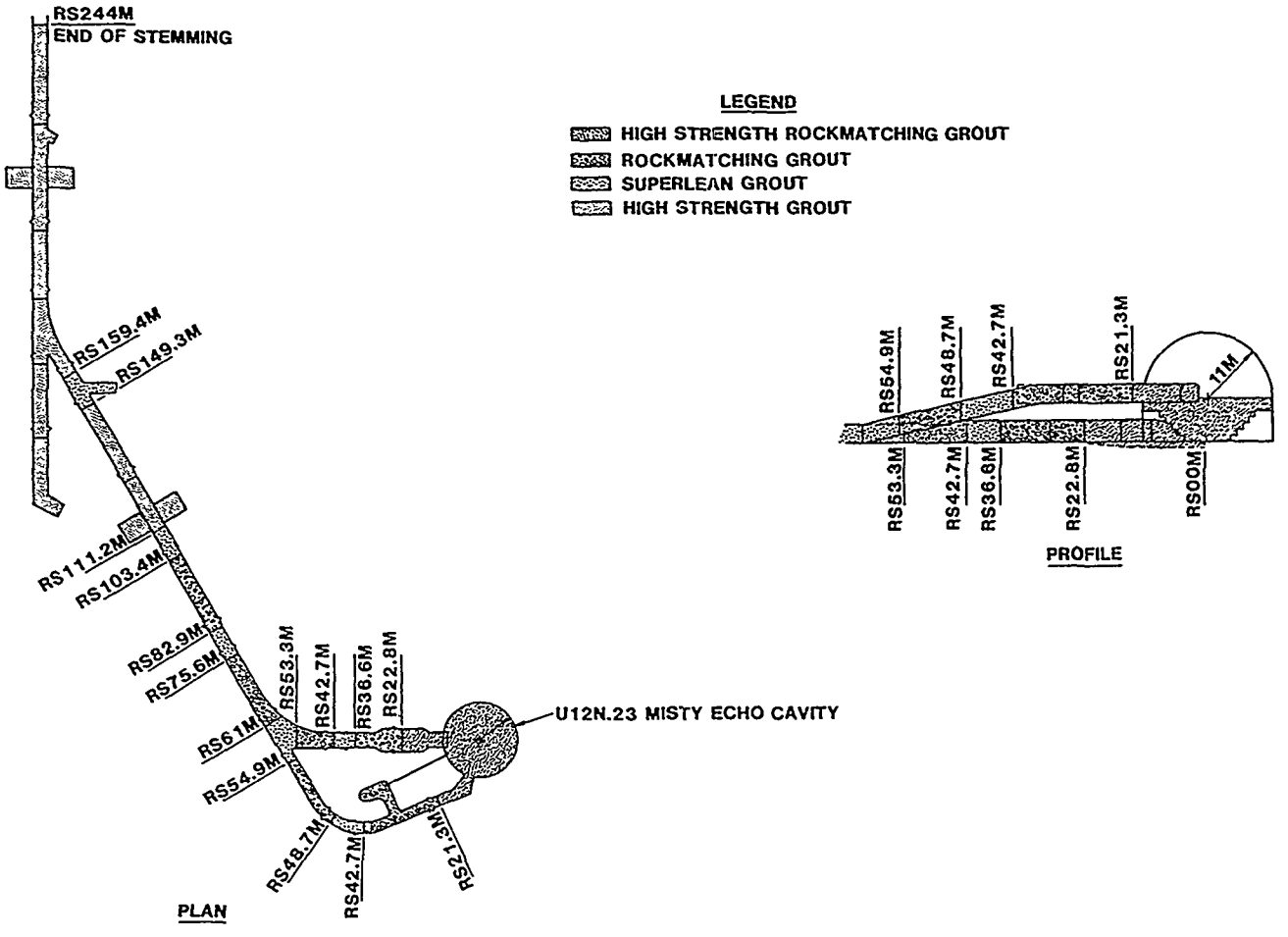


Figure C.1-20
MISTY ECHO Stemming Plan

The sealing associated with the event was similar to the HLOS tests; i.e., DPP, FDPP, alcove drift protection plugs, and GSP were part of the containment assign. Figure C.1-21 represents a bulkhead at the end of stemming. It is presented to illustrate basic design practice; a large number of data acquisition cables; bleed, fill, and top-off lines in face of bulkhead; and dewatering lines at invert.

C.1.4 Materials

The contents of Sections C.1.4 and C.1.5 represent a synopsis of three decades of experience on HLOS tests at the NTS. This experience was obtained by DNA through the Waterway Experiment Station and other contractors. DNA has limited experience placing noncementitious materials underground, having placed "sand" plugs on a couple of past events, filling a drift with "Monterey" sand on HYBLA FAIR, and partially filling a cavity with alluvial sand from Yuma, Arizona (MILL YARD). DNA and its contractors, however, do have a comprehensive understanding of cementitous materials. This understanding has evolved through a systematic program involving laboratory design and testing, field quality control and placement, and subsequent laboratory verification of field cast specimens.

As a result of the nuclear testing program, hundreds of different cementitious materials have been formulated. Mixtures are typically designed to produce certain hardened physical properties. Typical properties include uncontinued compressive strength, density, static modulus of elasticity, and sonic pulse velocity. Occasionally, other properties of interest include permeability, Poisson's ratio, dynamic modulus of elasticity, triaxial data at various confining pressures, and temperature history while hardening.

The materials used in providing containment of cavity gases and debris generated at the WP are termed stemming materials. These stemming materials can be categorized in four groupings: grout, groutcrete, concrete, and superlean grout. Grout mixes are normally used when physical rock properties must be matched. Groutcrete are typically used when higher strengths or matching of physical rock properties are required. Concretes are used when high strength is required. Superlean is a special material used for filling any potential gas paths that occur as a result of dynamic loading of the HLOS and drift. Table C.1-2 summarizes some relevant characteristics of these materials.

In many instances expansive cement is used in the majority of the mixtures based at the NTS. This type of cement enables the mixtures to expand even after hardening to produce

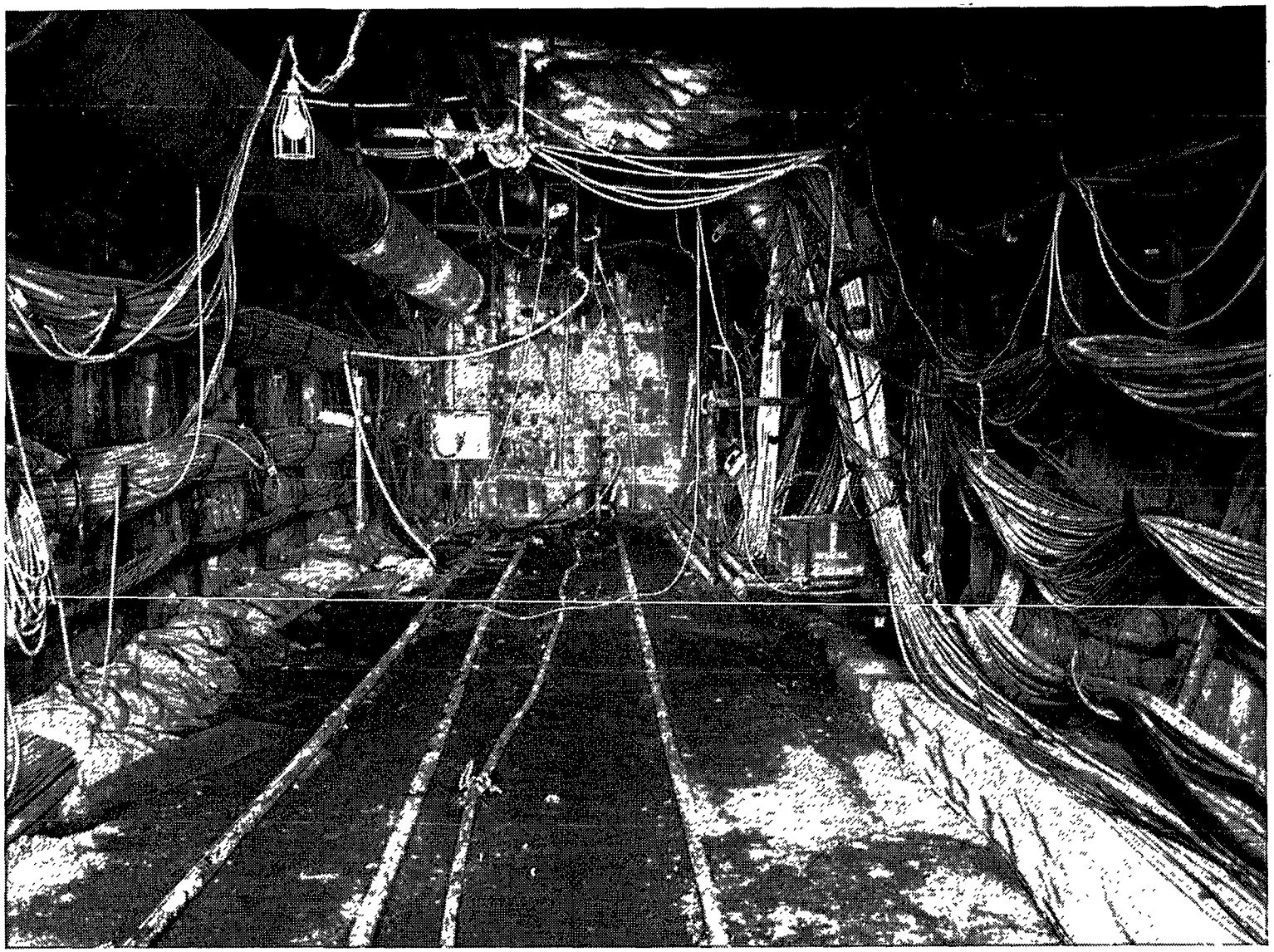


Figure C.1-21
Bulkhead at End of Stemming, MISTY ECHO
(Photo courtesy of DNA/Johnson Controls)

Table C.1-2
Typical Classes of Materials Used at NTS

Material Class	Compressive Strength, psi	Application
Grout	1,500 to 3,500	Rock-matching characteristics required: • Between WP and FAC where differences in materials is critical to cavity growth • Drill holes with stress gages, accelerometers, or specific experiments • Button-up stemming to meet early high-strength requirements
Groutcrete	2,500 to >5,000	High-strength or rock-matching characteristics required: • Top-off mixes for concrete placement
Concretes	>5,000	High-strength required: • Hardened pipe section • Anchor plug regions of the LOS and bypass drifts (contains coarse aggregate)
Superlean grout	200 to 300	Specialized stemming material: • Filling fractures and voids • Acts as a fluid under high pressures filling voids (large percentage of sand)

seals that will prevent leakage after denotation. Cements often used to obtain expansion of the stemming materials against the drift wall are ChemComp and ChemStress. ChemComp contains amounts of anhydrous calcium sulfoaluminate that forms a high-sulfate ettringite, which produces an expansion resulting in a shrinkage compensating mixture. ChemStress contains an increased amount of expansive producing compounds than are found in ChemComp cement. Table C.1-1 illustrates the number of mixtures using ChemComp cement on MIDDLE NOTE.

In addition to the use of these cements, a broad range of other cements, admixtures, and aggregates have been used. From the large volume of testing performed by the WES Structures Laboratory, relationships between these materials and the mixture properties have been established. Tables C.1-3 through C.1-6 illustrate this relationship in a qualitative manner.

C.1.5 Placement

Stemming and grout material placements are performed using numerous and detailed procedures. Therefore, the discussion below represents only a synopsis of the procedures used at the NTS. The discussion for stemming and grouting operations considers mixing and transport of these materials to the placement location, procedural considerations, and placement activities.

In general, the success of the stemming operations at the NTS is due to the following:

- use of experienced personnel,
- flexibility in scheduling,
- an awareness to document any unusual observations,
- meticulous recording and photo documentation of all operations,
- an adherence to detail,
- detailed checklists for all activities and the need to have the checklists completed during each sequence of the operation,

Table C.1-3

Effects of Various Cements on Physical Properties of Grouts, Groutcretes, and Concretes Used at NTS*

		Chem Comp	Class A PC	Class H PC	Gypsum Cement	Type I PC	Type II PC	Micro Fine
Density	Decrease	X			X			X
	Normal		X			X	X	
	Increase			X				
Water Required	Less			⊗				
	Normal		X		X	X	X	
	More	X						⊗
Viscosity	Decrease			⊗	X			
	Normal		X			X	X	
	Increase	X						⊗
Early Strength	Decrease			X				
	Normal		X			X	X	
	Increase	X			⊗			⊗
Final Strength	Decrease				X			
	Normal		X			X	X	
	Increase	X		X				X
Water Loss	Decrease	X						X
	Normal		X		X	X	X	
	Increase			X				
Pulse Velocity	Increase	X		X				X
	Normal		X			X	X	
	Decrease				X			
Modulus	Increase	X		X				X
	Normal		X			X	X	
	Decrease				X			
Tempera-ture	Increase	X						X
	Normal		X		X	X	X	
	Decrease			X				

X Denotes Minor Effect

⊗ Denotes Major Effect and/or Principal Purpose for which used

*Table from WES Structures Laboratory, Concrete Technology Division - personal communication with grouting staff

Table C.1-4

Effects of Aggregates on Cementitious Mixtures of Grouts, Groutcretes, and Concretes Used at NTS*

		Cole-manite Sand-Sized	20-40 Silica Sand	Overton Silica Sand (Fine)	NTS Concrete Sand (Derived from Tuff)	Natur-alite Sand	Desert Fines Sand	Lime-stone Sand	Basalt Sand
Density	Decrease	X				⊗	X		
	Normal		X	X				X	
	Increase				X				X
Water Required	Less								
	Normal	X	X					X	X
	More			X	⊗	⊗	⊗		
Viscosity	Decrease					X			
	Normal	X	X		X			X	X
	Increase			X			⊗		
Strength	Decrease	X		X		X	⊗		
	Normal		X		X				
	Increase							⊗	⊗
Durability	Decrease	X				X	X		
	Normal		X	X	X			⊗	⊗
	Increase								
Water Loss	Decrease								
	Normal	X	X	X	X			X	X
	Increase					⊗	⊗		
Pulse Velocity	Increase							X	X
	Normal		X	X	X				
	Decrease	X				X	X		
Modulus	Increase				X			X	X
	Normal		X	X					
	Decrease	X				X	X		
Tempera-ture	Increase								
	Normal	X	X	X	X	X	X	X	X
	Decrease								

X Denotes Minor Effect

⊗ Denotes Major Effect and/or Principal Purpose for which used

*Table from WES Structures Laboratory, Concrete Technology Division - personal communication with grouting staff

Table C.1-4

Effects of Aggregates on Cementitious Mixtures of Grouts, Groutcretes, and Concretes Used at NTS* (Continued)

		Iron Oxide (Powder)	Magnetite Sand	NTS 3/4-Inch Aggregate	3/4-Inch Limestone Aggregate	3/4-Inch Basalt Aggregate	Steel Fiber	Poly–Porylene Fiber
Density	Decrease							⊗
	Normal			X				
	Increase	⊗	⊗		X	X	⊗	
Water Required	Less				X	X	X	X
	Normal							
	More	⊗	X	X				
Viscosity	Decrease			X	X	X		
	Increase	⊗	X				X	X
Strength	Decrease							X
	Normal	X	X					
	Increase			⊗	⊗	⊗	X	
Durability	Decrease	X	X					
	Normal			X				
	Increase				⊗	⊗	X	X
Water Loss	Decrease				X	X	X	X
	Normal							
	Increase	X	X	X				
Pulse Velocity	Increase	X	X		X	X	X	X
	Normal			X				
	Decrease							
Modulus	Increase	X	X		⊗	⊗	X	
	Normal			X				
	Decrease							X
Tempera-ture	Increase							
	Normal			X	X	X		X
	Decrease	X	X				X	

X Denotes Minor Effect

⊗ Denotes Major Effect and/or Principal Purpose for which used

*Table from WES Structures Laboratory, Concrete Technology Division - personal communication with grouting staff

Table C.1-5

Effects of Mineral Fillers and Pozzolans on Grout, Groutcrete, and Concrete Mixtures Used at NTS*

		Silica Fume (Pozzolan)	Fly Ash (Pozzolan)	Barite (Powder)	Silica Flour (Powder)
Density	Decrease	⊗	⊗		
	Increase			⊗	⊗
Water Required	Less				
	More	⊗	X	X	X
Viscosity	Decrease				
	Increase	⊗	X	X	⊗
Early Strength	Decrease		X	X	X
	Increase	⊗			
Final Strength	Decrease		X	X	X
	Increase	⊗			
Durability	Decrease				
	No effect			X	
	Increase	⊗	⊗		X

X Denotes Minor Effect

⊗ Denotes Major Effect and/or Principal Purpose for which used

*Table from WES Structures Laboratory, Concrete Technology Division - personal communication with grouting staff

Table C.1-6

Effects of Some Additives on the Physical Properties of Mixtures*

		Bentonite	Perlite	Diatomaceous Earth	Pozzolan	Sand	Barite	Ilmenite	Calcium Chloride	Sodium Chloride**	Lignosulfonates	CMHEC†	Diesel Oil	Low-Water-Loss Material	Lost Circulation Material
Density	Decrease	⊗	⊗	⊗	⊗										
	Increase					⊗	⊗	⊗		X					
Water Required	Less	⊗	X	⊗	X	X	X								X
	More														
Viscosity	Decrease								X		⊗				
	Increase	X	X	X	X	X	X	X							X
Thickening Time	Accelerated	X							⊗	⊗					
	Retarded			X						X	⊗	⊗	X	X	
Early Strength	Decreased	X	X	X	X		X				⊗	⊗		X	X
	Increased								⊗	⊗					
Final Strength	Decreased	X	X	⊗	X		X	X				⊗		X	X
	Increased										X				
Durability	Decreased	X	X	X									X		X
	Increased				⊗										
Water Loss	Decreased	⊗									X	⊗	X	⊗	X
	Increased		X	X											

X Denotes Minor Effect.

⊗ Denotes Major Effect and/or Principal Purpose for which used. Ref. API Cementing Practices in the United States.

† Carboxymethyl hydroxyethyl cellulose.

* Table from Engineers' Handbook - Dowell.

** Small percentages of sodium chloride accelerate thickening. Large percentage may retard API Class A Cement.

- independent checks on multiple aspects of the operations, and

- posttest evaluations to resolve problems that may have occurred during the test.

C.1.5.1 Mixing and Transport

Stemming mixes are batched and mixed at different locations on the NTS. Batch plants are in centralized locations to provide batching, blending, and weighing of materials for the stemming mixtures. The primary batch plant has weighing and dry-blending equipment for blending dry fine materials, including small quantities of dry powdered mixture modifying additives. The secondary batch weighing plant includes equipment for handling cement, water, ice, and fine and coarse aggregate. At this plant a commercial ice shaver/slinger unit is used to charge the drums of the transmix trucks containing the groutcrete or the concrete mixtures. It is also used in conjunction with a water chilling system to depress groutcrete and concrete mixture temperature to between 40° and 50°F. Water is chilled using a liquid nitrogen system capable of lowering the temperatures to about 35°F.

Typical volumes of mixtures batched are between 50 to 500 cubic feet. Field control is performed by supervising the dry batching to the nearest 10 lb. The wet blend is then checked for density to the nearest 0.1 lb per gallon. When the wet mixture weighs right and "looks and feels right," it is released for placement.

Once batched, the mixture can then be transported in a dry or wet form to the portal. If it is in a dry form, it is mixed slowly with water at the portal. Wet mixtures are transported in conventional transmix trucks to the tunnel portal. At the portal, the mixture can be pumped or transported to the placement area. Moran cars are used to transport superlean, groutcrete, and concrete mixes underground that cannot be pumped up line. At the placement area, the mixture is dumped from the Moran car into a receiving hopper of a concrete pump at the placement area. Grouts that contain smaller-sized aggregate can be pumped via a 3-in. line routed from the pump truck to the placement location. The pump truck used to pump this mix from the portal is used for the placement of highly fluid grouts. Placement of grouts pumped from the portal can be transported from the tunnel portal horizontal distances up to approximately 8,800 ft or pumping "downhole" to depths of several thousand feet. In all cases, the grout is designed to maintain its pumpable characteristics until placement is complete. This sometimes requires that the mixture be highly pumpable for up to 3 hours.

Therefore, in summary, stemming and containment operations may require as many as three different transport hauls: (1) from the primary batching system to the secondary batching system, (2) from the secondary batching system to the portal, and (3) from the portal to the underground placement locations.

C.1.5.2 Stemming and Grouting Operations

Normal placement procedures in a tunnel would involve first placing concrete up to a certain level in a continuous pour; then placing the more fluid groutcrete (before the concrete hardened) to the tunnel roof; followed by pressure-grouting with a strong, expansive neat (i.e., cement and water) cement grout mixture at pressures up to 250 psi through pipes emplaced prior to initial concrete/groutcrete placement. For vertical holes, a continuous grout placement or a stage-grouting sequence is employed.

C.1.5.2.1 Preparatory Activities—Stemming

Prior to the placement of stemming materials, preparatory activities are required for the successful placement of the stemmed material. These include, but are not limited to, the following:

- Provide backup equipment, such as pressure gages and pumps.
- Remove construction materials, such as wood, ties, and tracks.
- Remove all temporary cables.
- Clear pour area of all loose materials down to the undisturbed rock.
- Wash the pour area of all dust and debris, and then remove the water.
- Coat existing concrete with concrete glue (weldcrete), and sand to roughen surface, thereby providing a good bond between the stemming mixture and the penetration.
- Install three gas block rings per line to prevent boundary leaks for all steel penetrations.
- Rough-coat all steel penetrations and gas rings with weldcrete and sand.

- Spray the inside of the bulkhead with sodium silicate.

- Photo-document the stemmed area prior to commencing stemming operations.

C.1.5.2.2 Preparatory Activities—Grouting

Prior to the placement of grouts, preparatory activities are required for the successful placement of these materials. These include, but are not limited to, the following:

- Place all fill, bleed, and top-off lines and any postgrout lines through the bulkhead.

- Place celite grout around the grout pipes at the collar of the grout holes. The celite prevents concrete material from flowing along the grout pipe and into the grout hole while the concrete is being poured for the plug. This mixture will break down when the cement grout, under pressure, is pumped through the grout pipes after the plug concrete is set up. This allows the grout to flow into any cracks or small openings along the interface between the plug and the tunnel wall. The celite grout is worked into the annulus between the hole and the grout pipe to a depth of 3 in. below the hole collar.

- Visually inspect all lines.

- Assure that the supply of grout is available by assuming the calculated volume of the holes is required.

- Prewet holes and then blow dry.

- Fill all fill, bleed, and top-off lines with water approximately 24 hours prior to placements. This procedure is designed to slick the lines and allow authorized personnel the opportunity to check the integrity of the lines and rubber seals used as line couplings.

- Begin pumping only after all the items on the checklists have been signed off.

C.1.5.2.3 Placement

The placement of the pour can occur either as a monolith or as a series of pours. In monolithic pours, if problems arise, keyways should be installed perpendicular to the length of the plug. The placement of multiple pours is described below.

Figure C.1-22 illustrates initial placement of a cementitous material between two stemming bulkheads. Depicted here are the fill lines (F-1, F-2, F-3), the bleed lines (B-1, B-2, and B-3), and the top-off lines (TO-1, TO-2, and TO-3).

The first lift in a pour is intended to bury the cable to provide (1) temperature protection for the latter mass placement, (2) physical protection for the later construction, (3) early determination of any voids, and (4) a uniform working surface. Because several months may occur between the invert placement and the following lifts, keyways may be required across the drifts.

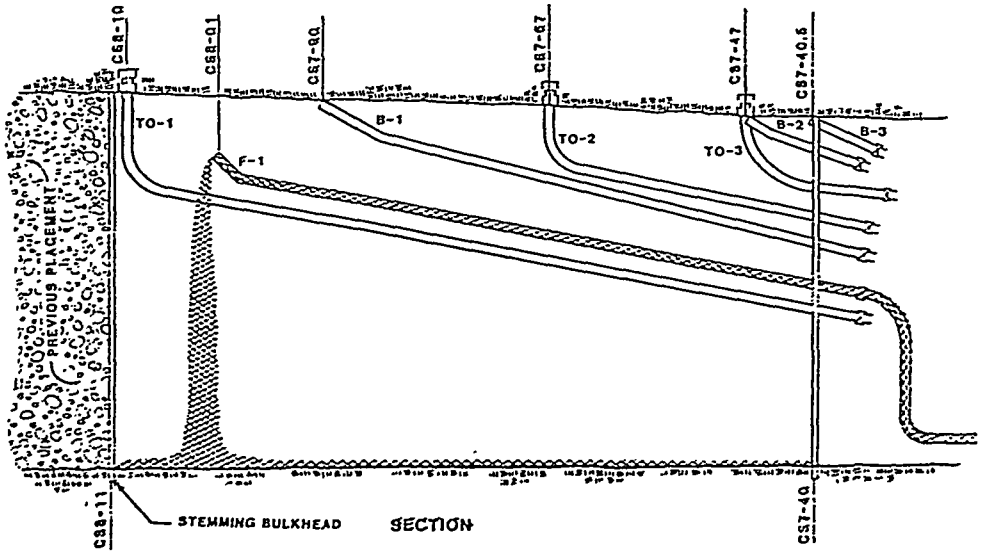


Figure C.1-22
Initial Placement of Stemming Material

When emplacement progresses, slurry is pumped through the fill lines to slick the pump and lines and then to provide a slurry for the stemming material-to-tuff interface. This slurry is floated on top of the heavier stemming mixes.

Fill lines are the primary lines in which stemming material is pumped through the bulkhead. Bleed lines are used primarily as an outlet for air from between the bulkheads as the area between the bulkheads is filled with grout. Top-off lines are used to complete the fill of the stemming void. These lines are typically 2 in. in diameter, 3 ft long, and placed in a 4-in.-diameter hole that is 4 ft deep into the tunnel back. They are located in the highest point of the tunnel and in other areas as determined while in the field. These lines exhaust the air during final pumping operations. When the returns occur in the top-off lines, the lines will be immediately pressure-grouted with the main pour stemming materials or the pressure grout material. They are drilled into fractured regions or slabby tuff on the edge of the placement.

Figure C.1-22 illustrates the previous placement at the left but does not show the placement of an invert. However, in most instances this would be done. Figure C.1-23 shows partial

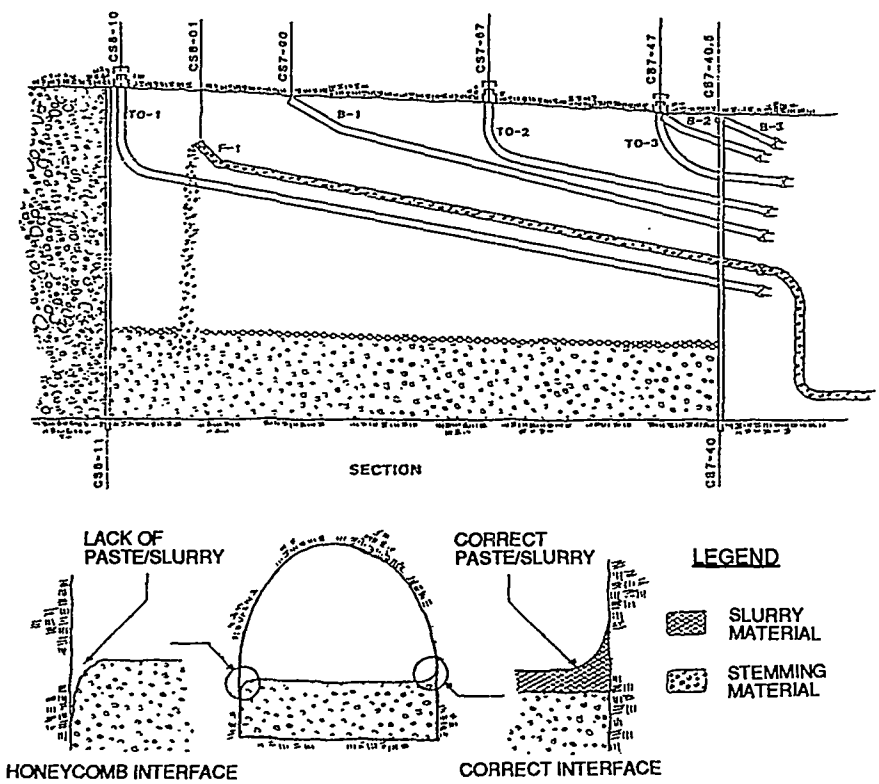


Figure C.1-23

First Pour of Stemming Operation with Honeycomb Effect Illustrated

placement. A slurry (bottom right) is placed on top of the heavier mixture to avoid a honeycomb interface. The water in the slurry will be absorbed in the rock matrix and preclude the downturn in the surface of the stemming material, as shown on the left. Finally, Figure C.1-24 illustrates the placement complete, because all bleed lines and top-off lines have returned full grout, the top-off pressure has been attained and stabilized, and the valve closed. After setup of the grout, the ends of the lines are covered with a steel plate. During the entire grouting process, pressures and volumes of grout take and the hole sizes are recorded.

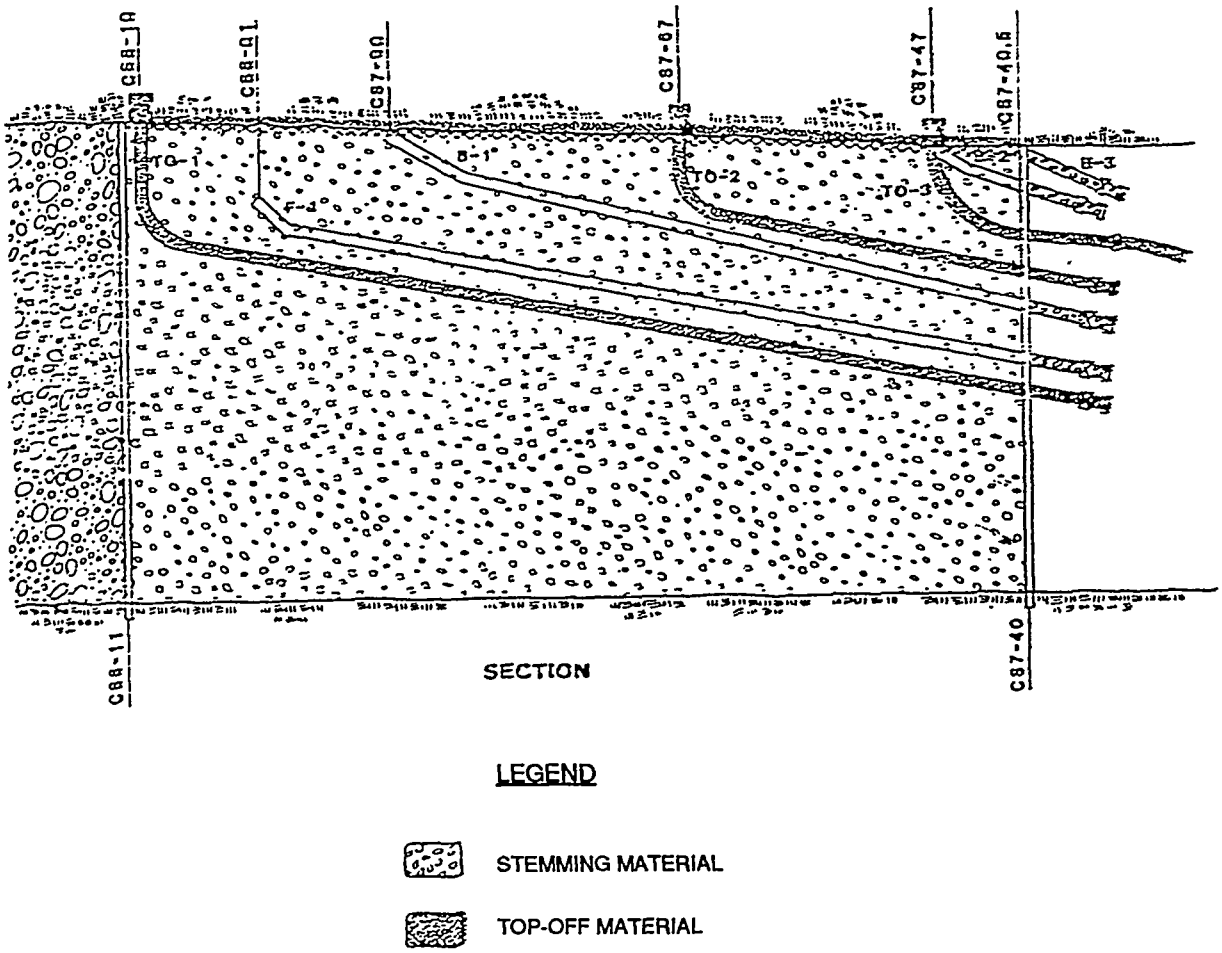


Figure C.1-24
Completion of Top-Off Returns

Additional activities performed include:

- Locating plugs in fractured areas so that the pre- and post-pressure grouting fills the fractures and voids around the plug, preventing the gas passage.

- Taking quality control samples from the form underground for testing of the compressive strength, pulse velocity, dynamic modulus of elasticity, and density. Slump tests are performed and temperatures recorded.

C.1.6 Discussion

An impressive amount of field-testing experience has been obtained from the U.S. nuclear testing program. The experience most relevant to the Yucca Mountain Site Characterization Program is the placement of stemming materials to contain the nuclear detonations within tunnel complexes. Cementitious materials are the primary type of stemming materials.

The information of most value can be placed into two categories: the placement of materials and the formulations of the materials. Problems, such as shrinkage of stemming materials along the interface with the rock, have been addressed by the use of expansive materials. "Honeycomb" development along the ribs of the tunnel have been addressed by placing a slurry on top of the heavier stemming materials during placement. Control of set time has been addressed by prechilling the water as low as 35°F so that the mixture is placed at temperatures between 40° and 50°F.

Because of the vast amount of stemming and grout testing, correlations between material constituents and mixture properties have been developed by the WES-SL and other participants in the NTS testing program. These correlations are presented in Section C.1.3.

Certain operational conduct from the nuclear testing program is also relevant; these are listed in the introduction to this section (C.1.5).

C.1.7 References

U.S. Congress, Office of Technology Assessment, *The Containment of Underground Nuclear Explosions*, OTA-ISC-414, U.S. Government Printing Office, Washington, D.C., (October 1989).

APPENDIX C.2

Waste Isolation Pilot Plant

Site Visit: September 1993

Purpose: The purpose of the case history is to demonstrate the technology developed to emplace seals in a nuclear waste repository in salt.

Table of Contents

C.2.1 Introduction C.2-1

C.2.1.1 Description C.2-1
C.2.1.2 Objective of Sealing Tests C.2-1
C.2.1.3 Purpose C.2-2

C.2.2 Materials C.2-2

C.2.2.1 Description C.2-2
C.2.2.2 Properties C.2-3

C.2.3 Emplacement Equipment and Instrumentation C.2-5

C.2.4 Seal Placement and Testing Operations C.2-6

C.2.4.1 Sequence of Operations C.2-6
C.2.4.2 Conditions of Emplacement C.2-13
C.2.4.3 Problems Encountered C.2-13
C.2.4.4 Quality Control C.2-13

C.2.5 Discussion C.2-13

C.2.6 References C.2-13

APPENDIX C.2
Waste Isolation Pilot Plant

C.2.1 Introduction

C.2.1.1 Description

The Waste Isolation Pilot Plant (WIPP) was created by the U.S. Department of Energy as a research and development facility to demonstrate the safe disposal of transuranic (TRU) radioactive waste generated from defense activities. Three main technological issues under investigation through laboratory tests, theoretical studies, and in situ tests are: safe handling of defense TRU wastes; adequacy of waste confinement systems, including waste containers and engineered barriers; and suitability of salt to provide for adequate underground access and long-term isolation. One of three sets of in situ experiments—the plugging and sealing tests—can be grouped in two major technical areas: (1) characterizing the Rustler and Salado Formations and (2) developing the seal materials and evaluating the seals. The focus of this case history deals with the second technical area, specifically the small-scale and large-scale seal performance tests (SSSPT and LSSPT) and the fracture-grouting test.

C.2.1.2 Objective of Sealing Tests

The SSSPTs are in situ sealing experiments that are intermediate in scale, between full-scale in situ seal tests and laboratory tests. The objectives of the SSSPTs are to

- determine in situ fluid flow performance for various seal systems,
- determine in situ mechanical performance of the host rock and seal materials,
- assess seal emplacement techniques, and
- support the development of numerical predictive capabilities.

The objective of the fracture-grouting experiments was to determine the efficiency of gas transmissivity reduction by the injection of microfine, cementitious grout through drill holes into marker bed 139 and the fractured, overlying halite.

The objectives of the LSSPTs are to:

- evaluate seal material emplacement techniques,

- evaluate time-dependent fluid-flow and structural performance of the seal system,

- compare in situ data with computer calculations to validate flow and structural models,

- demonstrate full-scale emplacement and seal performance in the host-rock environment, and

- develop the database for a WIPP underground seal design.

The majority of testing to date has involved the SSSPT. The fracture-grouting test, while completed, is currently being documented and the data evaluated. The LSSPTs at this writing are being developed; therefore, the majority of the discussion below deals with the SSSPTs.

C.2.1.3 Purpose

The purpose of the tests discussed above is to develop and demonstrate technology such that seals can be designed to limit the release of radionuclides to the accessible environment.

C.2.2 Materials

C.2.2.1 Description

SSSPT

Sealing design concepts presented by Nowak et al. (1990) include the use of cementitious materials (grouts and concretes) and "natural" materials (salt and bentonite). Crushed salt, consolidated to low permeability, is the principal long-term barrier to fluid flow. Short-term seal components are required until the creep consolidation is sufficient. Concrete, specifically developed for WIPP, and swelling clay material (e.g., bentonite) have been selected for the short-term seal components. The testing program for the SSSPT used expansive salt concrete seals, 100% bentonite seals, 50/50% crushed salt/bentonite seals, and 100% crushed salt block seals.

The emplaced seal materials consisted of blocks measuring 31.1 x 10.5 x 15.2 cm made in the same manner as those used in Test Series C. The blocks were comprised of a mixture of mined salt screened to 9.5 mm (0.375 in.) and 2% added water and were pressed in a block machine using approximately 26.2 megapascals (MPa) (3,800 pounds per square inch [psi]). A mixture of crushed salt and water was used to fill in the voids where it was not practical to put blocks.

Fracture Grouting

The proposed grout was a portland cement and pumice mixture composed of the following:

- 75% by mass, ordinary, sulphate, resistant portland cement (Canadian Type 50), Blaine fineness of 450 m^2/kg;
- 25% by mass, pumice;
- 2.5% superplasticizer (D-65); and
- mix water-tap water (water/cement + silica fume ratio 0.5).

C.2.2.2 Properties

SSSPT

Expansive Salt Concrete Seals

For the SSSPT-A and B tests, seal performance was summarized by Finley and Tillerson (1992) and is included below.

- Submicrodarcy initial permeability was measured during the brine flow testing (Peterson et al., 1987).
- A reduction in flow path size was observed by a decrease in tracer arrival times measured within 1 year of seal emplacement.
- Brine flow rates of about 200 mL/day corresponding to submicrodarcy permeabilities were measured for the 91-cm-diameter seals.
- Structural performance was satisfactory as evidenced by the seals withstanding 1.8 MPa back pressure during flow testing.
- The expansivity of the concrete (0.12% in the lab) provided sufficient interface between the seal and the rock to limit fluid flow at early times.
- There was little evidence of thermal cracking of the concrete during curing.

100% Bentonite Blocks

SSSPT-C Phase 2 and SSSPT-D Phase 2

- Microdarcy permeability was measured after about 2 months of brine testing.
- The only 100% bentonite seal tested showed a marked decrease (about two orders of magnitude) in brine flow due to bentonite swelling.
- Pressure measurements show a roughly twofold increase in the seal pressure after about 300 days of brine testing up to values near 0.7 MPa.

50/50% Salt/Bentonite Seals

SSSPT-C Phase 1

- Millidarcy permeability was measured after about 6 months of brine testing.
- Structural measurements by Stormont and Howard (1987) suggest that the salt/bentonite block seals do not behave significantly different from the 100% salt block seals over the time periods discussed.

100% Crushed Salt Block Seals

SSSPT-C Phase 1 and -D Phase 1

- Gas flow rate within a few months after seal emplacement exceeded the measuring capability of the equipment.
- Structural measurements, including seal pressure and borehole displacements, agree with the laboratory and modeling predictions that crushed salt seals will provide little resistance to closure and little resistance to flow until the crushed salt has achieved 90 to 95% of the intact salt density.

Fracture Grouting

Bleed: none

Viscosity: Marsh cone = 70 s

Setting time: final setting time 14.5 hours

Shrinkage: 1.5%

Strength: 32 MPa at 7 days

C.2.3 Emplacement Equipment and Instrumentation

SSSPT

Material blocks were pressed into workable sizes using a modified adobe block-making machine (Stormont and Howard, 1987). The blocks were pressed at about 26.2 MPa with 2 to 4% (wt) added water. As indicated by Finley and Tillerson (1992), the manufacture and emplacement of the seal materials for each of the test series were accomplished without much difficulty. However, the placement of the expansive salt-based concrete (SSSPT-A and SSSPT-B) required the use of chilled water and retarding admixture (sodium citrate) to maintain workability during transport underground.

Because performance of the seals was also being monitored, instrumentation of the seal was performed. As an example of the detail incorporated into the test, the instrumentation for the Test Series D, Phase 1, is briefly presented.

As discussed by Torres et al. (1992), only those gages proven to be reliable in Test Series A, B, and C were used in Series D. The following gages were used to monitor structural performance:

- Pressure cells were placed within the seal and are intended to measure the stresses internal to the seal.

- Protected hole-closure gages were placed within the interior of the seal and monitor hole closure within an emplacement borehole.

- Axial displacement gages are fixed axially in a seal spanning two block layers. They measure axial displacement within the crushed salt seal.

- Open hole-closure gages were placed in an empty hole to measure open hole closure.

- Extensometers monitor the extension in the surrounding host rock. They were placed in a satellite hole parallel to the emplacement hole.

A schematic of block placement in the D Series holes is shown in Figure C.2-1, while the associated instrumentation is shown in Figure C.2-2. Prior to actual instrument placement, "dummy" gages were fabricated and placed in a trial hole to demonstrate the feasibility of emplacing instrumentation within the seal.

Fracture Grouting

The schematic for the equipment used in the grouting operation is shown in Figure C.2-3. All metal portions of the grouting setup were 316 stainless steel. The pulverization mill and the agitation tank were chilled with cold water. The grout lines were 3/4 in. thermoplastic polypropylene lined with Nylon 11.

C.2.4 Seal Placement and Testing Operations

C.2.4.1 Sequence of Operations

The SSSPTs are a series of in situ experiments designed to evaluate the performance of various materials emplaced in boreholes from 6 in. (15.2 cm) to 38 in. (96.5 cm) in diameter. The holes were oriented both horizontally into the rib and vertically into the floor. A summary of the testing and the seal material involved is given in Table C.2-1. In general, an emplacement hole and an access hole are drilled either in the floor or the rib.

Figure C.2-4 illustrates a generic test configuration for the SSSPT. An emplacement hole is drilled into the rib or the floor. A small-diameter hole may also be drilled adjacent to the emplacement hole for rock instrumentation, and an additional access or injection hole is used to introduce gas or brine under a seal. This hole is typically drilled at an angle to the emplacement hole to intersect the bottom portion of the emplacement hole. A packer is placed in the injection hole to pressurize the portion of the hole beneath the seal.

The placement techniques used in the SSSPT emphasized conventional technology and the use of available personnel at WIPP. The materials emplaced included expansive salt concrete, blocks compressed from 100% crushed salt, 50/50% crushed salt/bentonite, and

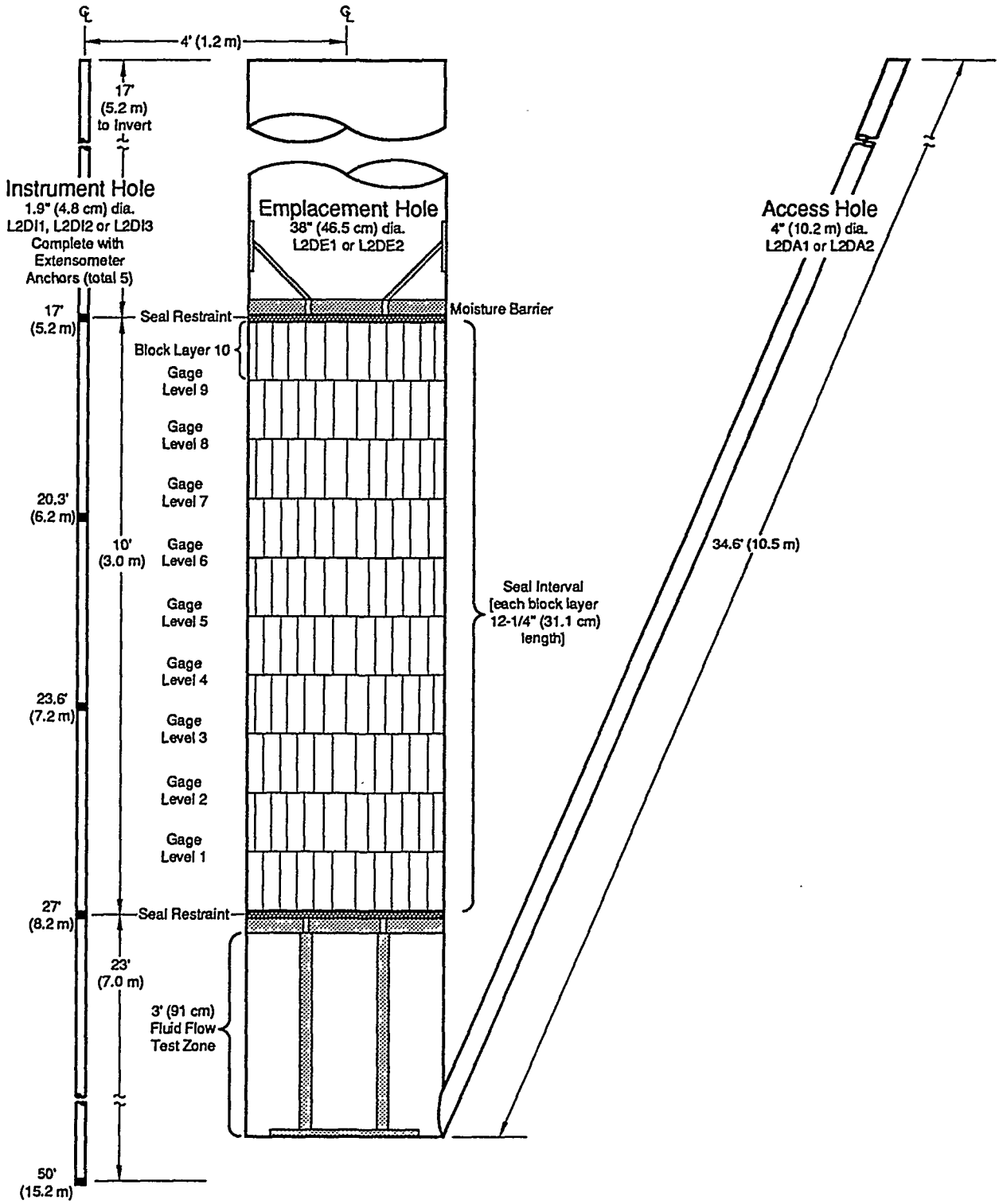


Figure C.2-1

Test Configuration for Holes L2DE1 and L2DE2

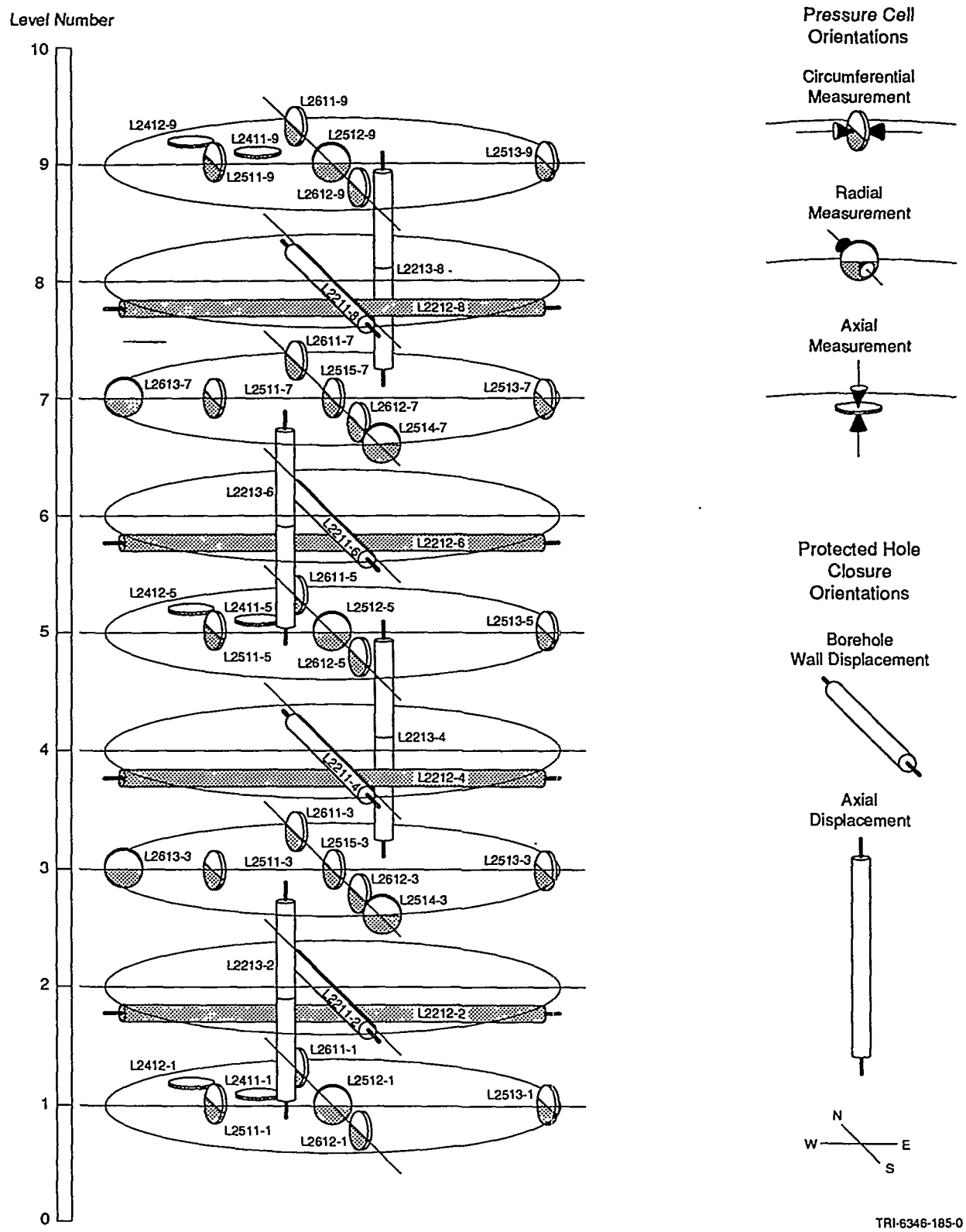
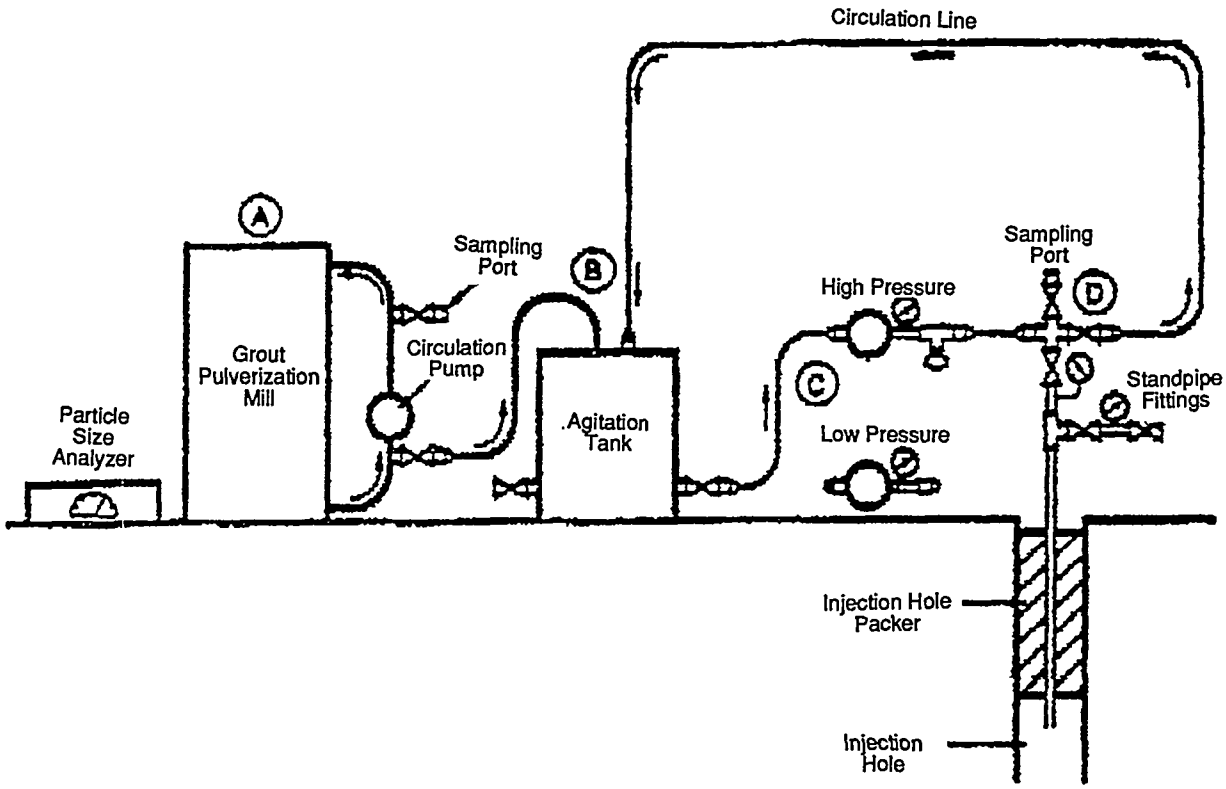


Figure C.2-2
Gage Layout for Seal Emplacement L2DE1



Legend

- Rupture Disk
- Pressure Gage
- Nipple
- Valve
- Quick Connect - Male
- Female "T"
- Low Pressure Hose
- High Pressure Hose
- Pump

System Description

(A)
- Grout mixed and pulverized (in the wet state) in mill.
- Pumped circulation aids pulverization and permits sampling for analyses on particle size.

(B)
- When particle size meets specifications, as defined in SOP472218, grout is pumped to agitation tank.

(C)
- Grout is pumped from agitation tank by one of two pumps. Pump selection determined by required pressure.
- Particle size is analyzed.

(D)
- Grout pumped to hole - valves control grout injection or permit bypass back to agitation tank.

Figure C.2-3
Schematic View of Grouting Equipment

Table C.2-1
Test Series Currently Planned for SSSPT

Test Series	Seal Material	Seal Emplacement Orientation	Emplacement Date	Measurements*
A	Salt-Based Concrete	Vertical	7/85	Seal Pressure; Displacement and Temperature; Gas and Brine Flow
B	Salt-Based Concrete	Horizontal	2/86	Seal Pressure; Gas and Brine Flow
C Phase 1	Salt and 50/50% Salt/Bentonite Block	Horizontal	9/86	Seal Pressure; Brine Flow
C Phase 2	Bentonite Block	Horizontal	12/90	Seal Pressure; Brine Flow
D Phase 1	Salt Block	Vertical	1/88	Seal Pressure; Hole Closure; Floor Heave; Gas Flow
D Phase 2	Bentonite Block (short-term)	Vertical	9/89	Seal Pressure; Brine Flow

*Note: Instruments include strain gages, stress meters, thermocouples, pressure cells, borehole displacement gages, Multiple Point Borehole Extensometers, and the Four Packer Fracture Flow Tool for fluid flow measurements.

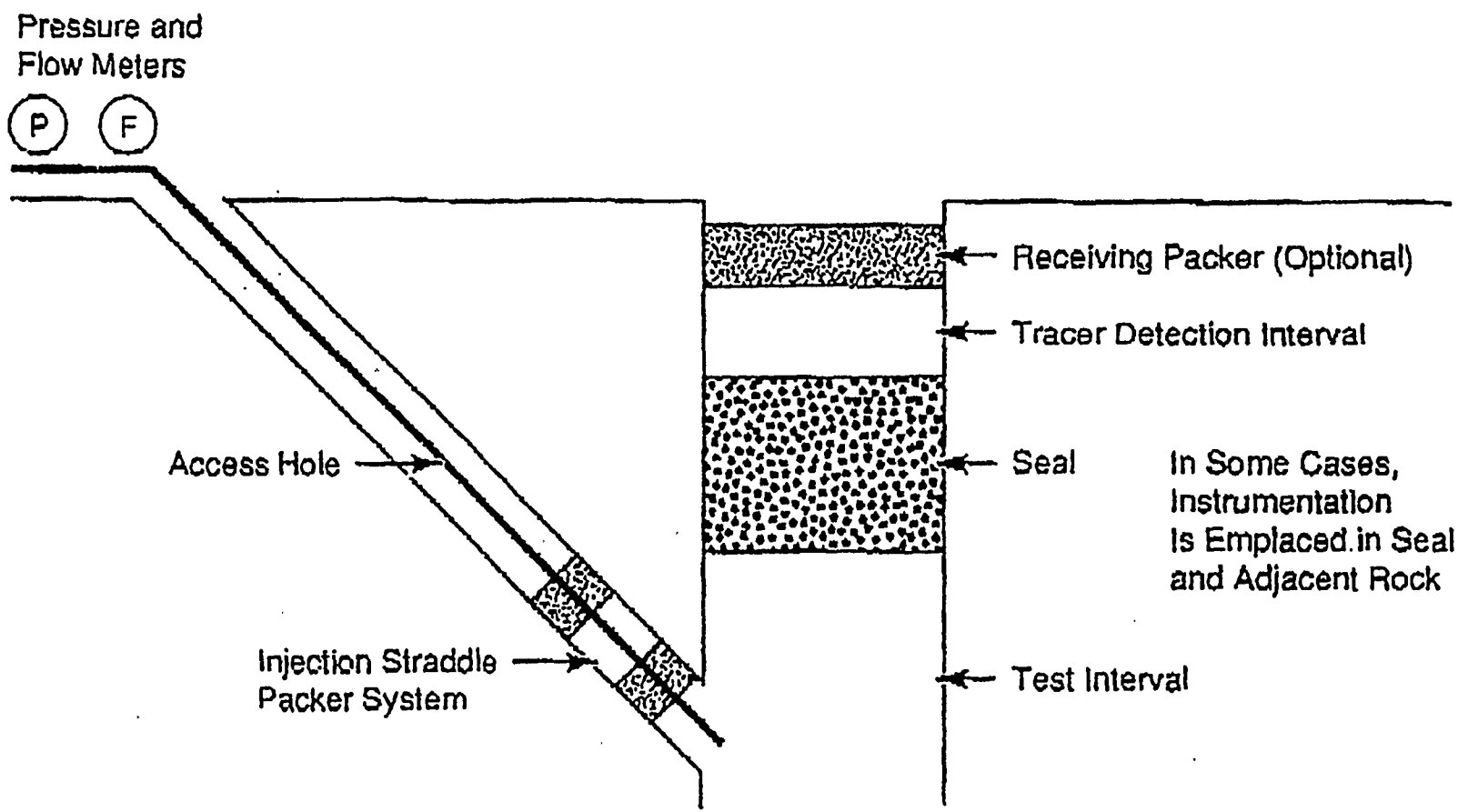


Figure C.2-4
Generic Test Configuration for the SSSPT

100% bentonite. A summary of the experience with the placement of these materials is given below.

Expansive Salt Concrete Seals

The SSSPT Series A and B used nine emplacement holes located in the floor of the D Room alcove and in the west rib of Room M. Emplacement of the seals was successfully demonstrated, as evidenced by the placement of nine seals in the vertical and horizontal orientations using standard equipment and personnel. The concrete met the emplacement criteria for slump, limited bleed, segregation, limited air entrapment, self-leveling behavior, and workability.

100% Bentonite Blocks

The SSSPT-C Phase 2 and the SSSPT-D Phase 2 used for seals constructed from the Sandia block machine. Emplacement of the 100% bentonite blocks was successfully demonstrated by the completion of the seals in a confined space, although labor was intensive. High-density (80% of intact bentonite) durable blocks could be consistently fabricated using modified technology, and a seal density of >75% of intact bentonite density can be achieved in a 3-ft bentonite core.

50/50% Crushed Salt/Bentonite Seals

SSSPT-C Phase 1 emplacement of 50/50% crushed salt/bentonite seals in horizontal boreholes was successfully demonstrated, although the placement was very labor-intensive. Also, high-density (>77% of salt/bentonite intact density) durable blocks of 50/50% crushed salt/bentonite could be consistently fabricated using the modified available technology.

100% Crushed Salt Block Seals

SSSPT-C Phase 1 and SSSPT-D Phase 1 used seven seals—four horizontally and three vertically. Emplacement of the 100% crushed salt block seals was successfully demonstrated in a confined space, although placement was very labor-intensive. High-density (>82% intact salt) durable blocks of 100% crushed salt could be consistently fabricated using modified available technology. Tamped in-place crushed salt can be emplaced to >75% of the host rock density, suggesting a possible alternative to block seals.

C.2.4.2 Conditions of Emplacement

The general stratigraphy of the WIPP site is bedded halite, with some interbeds of anhydrite, clay, and various mixed layers.

C.2.4.3 Problems Encountered

During the testing phase for test series A and B, embedded instruments in the seals showed obvious leakage along the instrumentation cable bundle during gas and brine testing.

C.2.4.4 Quality Control

All experimentation was performed under the Sandia National Laboratories quality assurance program for the WIPP. Multiple tests were run for redundancy. To evaluate the effects of closure, boreholes were left open to assess the effects of closure by itself.

C.2.5 Discussion

Seals using selected materials developed for the WIPP have been successfully constructed and emplaced with conventional technology and available personnel. Preliminary results suggest that expansive salt concretes and bentonite seals can provide seals against brine inflow during the early time periods, with permeabilities in the microdarcy range. The performance of the 50/50% crushed salt/bentonite mixtures is inconclusive at this time; however, permeabilities in the millidarcy range have been achieved.

C.2.6 References

Finley, R. E., and J. R. Tillerson (1992). "WIPP Small Scale Seal Performance Tests - Status and Impacts," *SAND91-2247*. Sandia National Laboratories, Albuquerque, New Mexico, 11 pp. (NNA.940415.0003)

Peterson, E. W., P. L. Lagus, and K. Lie (1987). "Fluid Flow Measurements of Test Series A and B for the Small Scale Seal Performance Tests," SAND87-7041, Sandia National Laboratories, Albuquerque, NM, Dec. 1987.

Torres, T. M., C. L. Howard, and R. E. Finley (1992). "Development, Implementation, and Early Results: Test Series D, Phase 1 of the Small-Scale Seal Performance Tests," *SAND91-2001*. Sandia National Laboratories, Albuquerque, New Mexico, 76 pp. (NNA.940415.0002)

APPENDIX C.3

Eagle Mine

Site Visit: July 22, 1992

Purpose: To document the use of cementitious plugs to seal abandoned horizontal mine openings.

Table of Contents

C.3.1 Introduction C.3-1

C.3.1.1 Description C.3-1
C.3.1.2 Objective of Sealing/Backfilling Operation C.3-4
C.3.1.3 Purpose C.3-4

C.3.2 Materials C.3-4

C.3.2.1 Description C.3-4
C.3.2.2 Properties C.3-4

C.3.3 Equipment C.3-4

C.3.4 Seal/Backfill Placement Operations C.3-4

C.3.4.1 Sequence of Operations C.3-4
C.3.4.2 Conditions of Emplacement C.3-5
C.3.4.3 Problems Encountered C.3-5

C.3.5 Discussion C.3-5

C.3.6 References C.3-9

APPENDIX C.3
Eagle Mine

C.3.1 Introduction

C.3.1.1 Description

The Eagle Mine is located near the town of Minturn, Colorado (Figure C.3-1). It was first developed at the turn of the century and became a consistent producer of lead-zinc ore for many years. The mine has passed through several changes in ownership. During the most recent active period from 1977 to 1983, the mine was operated by New Jersey Zinc, a subsidiary of Gulf and Western, Inc. The mine was accessed by shafts and adits constructed using drill-and-blast methods. Production for 1977 was reported to be 750 tons per day (tpd) zinc ore by underground timber stoping and room-and-pillar methods (Miller Freeman Publications, Inc., 1976). In 1983, the property was purchased by a Colorado entrepreneur interested in its gold/silver potential, which was subsequently found to be limited. The mine is no longer in operation, and the mining town of Gilman, Colorado, is largely abandoned.

The Gilman district lies near the center of the Colorado Mineral Belt, a rich mineralized zone stretching diagonally across the Colorado mountains from Durango to Jamestown (north of Denver). The Eagle Mine orebody is primarily a replacement-type deposit, with the lead-zinc ores occurring as galena and sphalerite in solution-collapsed breccias in layered Paleozoic carbonate rocks (Romberger, 1980). The stratified rocks overlie Precambrian crystalline rocks.

At the end of its active life, the mine pumped 300 to 400 gallons of water per minute (gpm), and its lowest levels extended some 300 ft below the Eagle River. Upon abandonment, the pumps were decommissioned, and water levels in the mine started to rise. Because of water quality problems and other environmental issues, the site was placed on the federal Superfund list. Dames and Moore was contracted to manage cleanup operations in 1985, under the supervision of the state of Colorado. In conjunction with other remediation activities, seals were constructed in the old mine openings in 1987, 1988, 1989, and 1990. A total of eight seals were installed during this period. Figure C.3-2 shows the location of some of Eagle Mine seals, including two inspected for this case study.

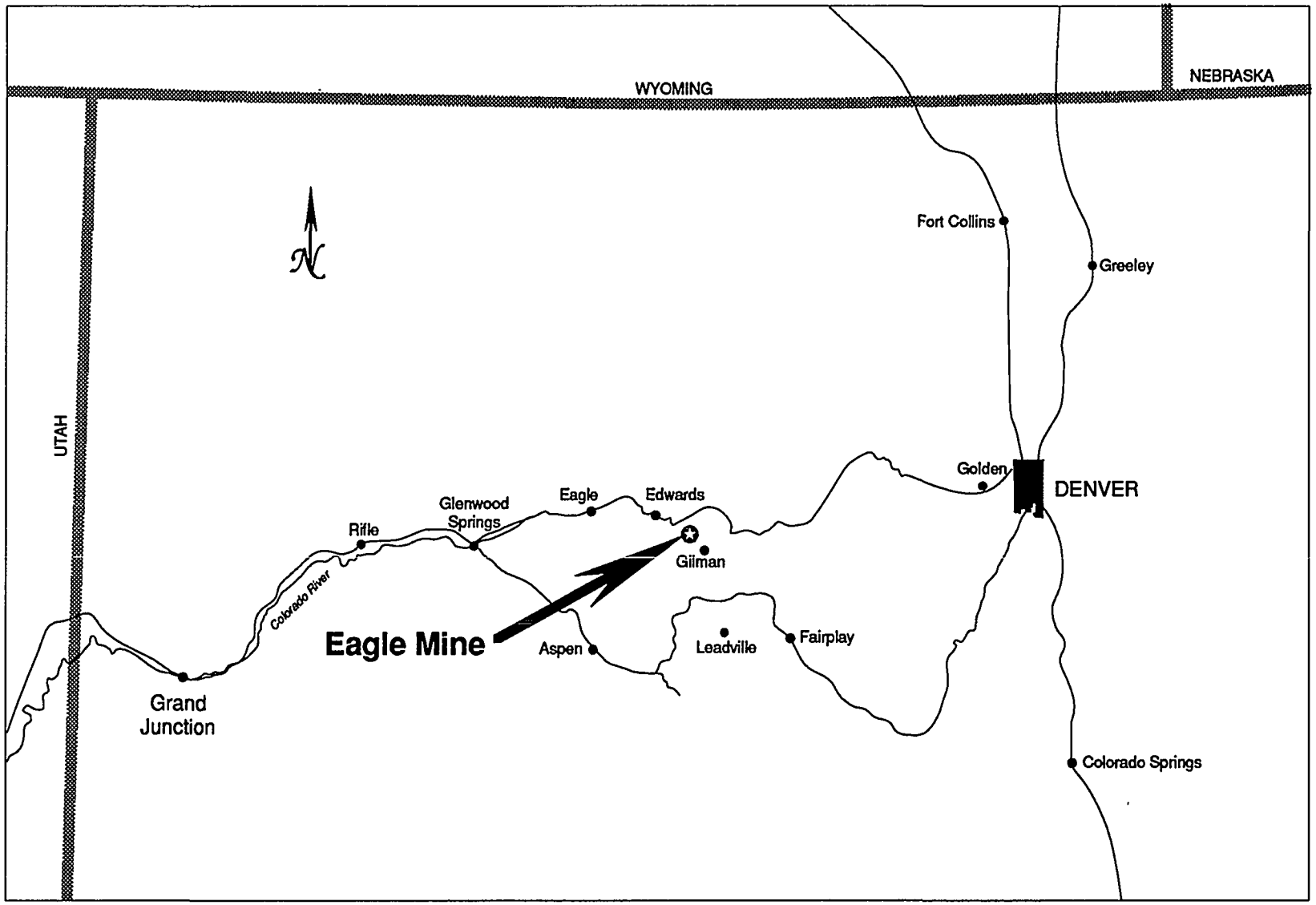
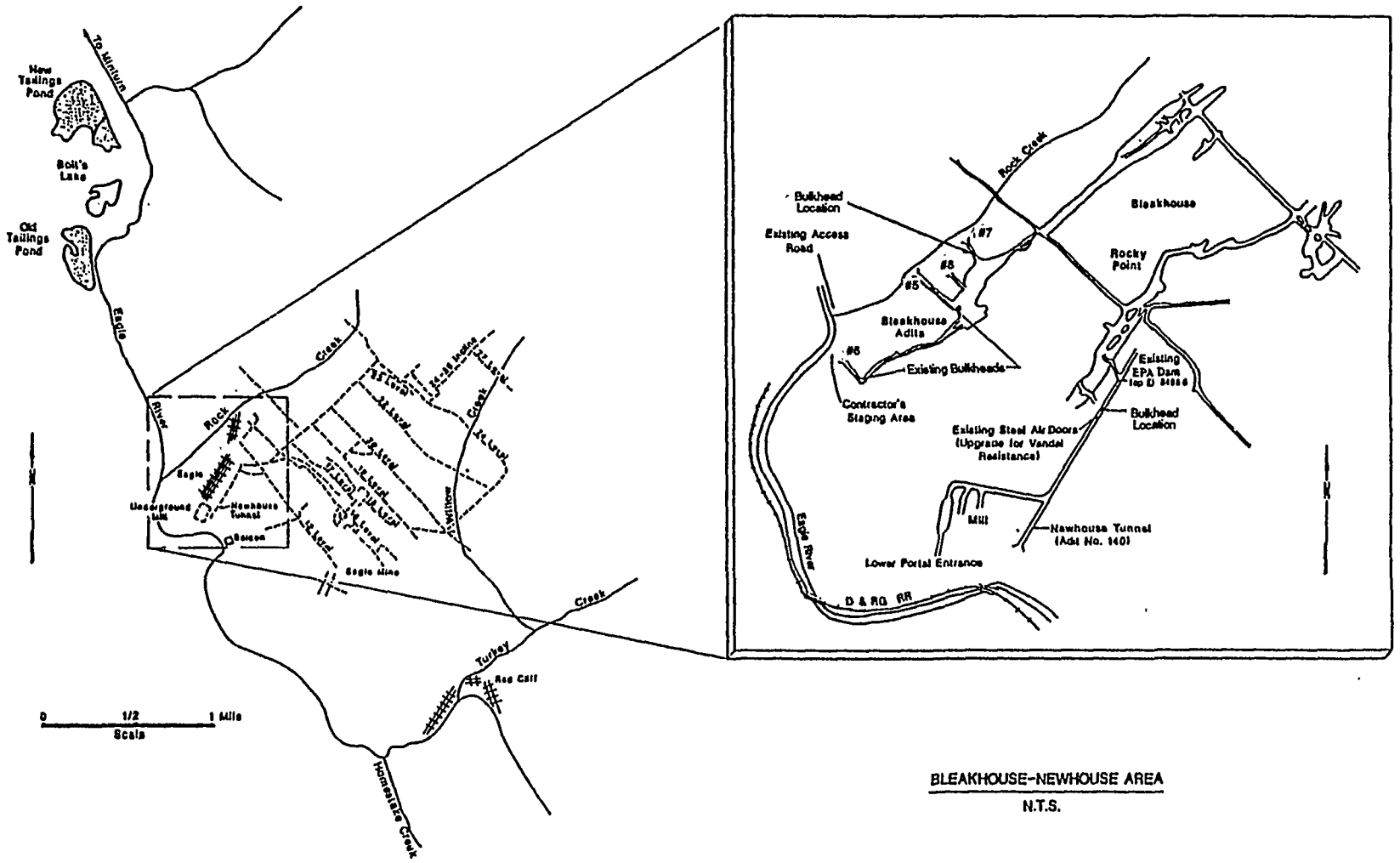


Figure C.3-1
Location Map for Eagle Mine



EAGLE MINE LOCATION MAP

BLEAKHOUSE-NEWHOUSE AREA

N.T.S.

Figure C.3-2

Location of Seal Nos. 6 and 7

C.3.1.2 Objective of Sealing/Backfilling Operation

After pumping operations were discontinued, contaminated water flowed from the abandoned mine workings into the Eagle River. The purpose of the sealing operation was to control these discharges by plugging the adits. Plugging was selected over backfill because the mine was already abandoned and partially flooded when sealing operations commenced, and the resulting safety and cost considerations were decisive. In this report, two representative plugs are described.

C.3.1.3 Purpose

Bulkheads or plugs are used for a variety of purposes in underground mining. The Eagle Mine case history documents the use of cementitious plugs to seal abandoned horizontal mine openings. Similar seals may be required at various locations in the potential Yucca Mountain repository as part of the overall sealing system, although the objectives will be somewhat different.

C.3.2 Materials

C.3.2.1 Description

All seals were constructed of concrete made from Type V sulfate-resistant portland cement with approximately 20% fly ash. The concrete was reinforced with polypropylene fibers.

C.3.2.2 Properties

The minimum specified 28-day strength of the concrete was 3,000 pounds per square inch (psi). The 28-day cylinder strength in the field was in excess of 4,000 psi, as determined by standard tests.

C.3.3 Equipment

Equipment consisted of off-the-shelf rubber-tired equipment, diamond drills, grout pumps, screw conveyors, concrete mixers, and jackhammers. No special equipment was used for any of the sealing operations.

C.3.4 Seal/Backfill Placement Operations

C.3.4.1 Sequence of Operations

For each plug installation, a suitable seal location was first established in the adit to be plugged. The location was selected near the portal, but far enough back to get away from

weathered and altered zones near the surface. At the selected location, the adit walls were scaled. A shallow, tapered keyway was constructed by drilling and barring. At least three rings of radial holes were drilled for pregrouting of the plug abutments. Grouting was performed using a portland cement-based grout.

Following pregrouting, forms were constructed and outlet piping installed in the forms. Concreting was done in one continuous operation to avoid cold joints. Each concrete placing operation took approximately 8 hours. After 28 days of curing, the front form was stripped. Contact grouting was performed to fill the remaining voids and seal the contact. Figures C.3-3 and C.3-4 show details of construction for plug No. 7.

C.3.4.2 Conditions of Emplacement

The adits to be sealed were located near the contact between underlying crystalline rocks and the ore-bearing quartzites and dolomites. Although all rock types were fractured, the stratified rocks were generally less competent and more highly fractured than the crystalline rocks. Plugs were emplaced in both geologic environments. Figure C.3-5 shows the No. 6 plug installation.

C.3.4.3 Problems Encountered

As the plugs were emplaced, the water level rose in the abandoned mine workings. The lower plug (No. 6) had approximately 150 ft of hydrostatic head acting at its base at the time of the site visit. The No. 7 plug had only about 10 ft of head. Although the plugs themselves were relatively watertight, some seepage escaped through naturally occurring fracture zones in the surrounding rock mass. Seepage bypassed plug No. 6 via a fracture zone that intersected the drift 40 ft out by the plug.

C.3.5 Discussion

The shaft and ramp plugs at the potential repository will be intended primarily to keep surface water out of the repository and to prevent gaseous releases from occurring. These functions are somewhat different than that for which the Eagle Mine plugs were intended: to retain water in the abandoned mine. However, the station plugs in the potential repository may serve to retain water in the shafts and ramps until it can dissipate into the formation.

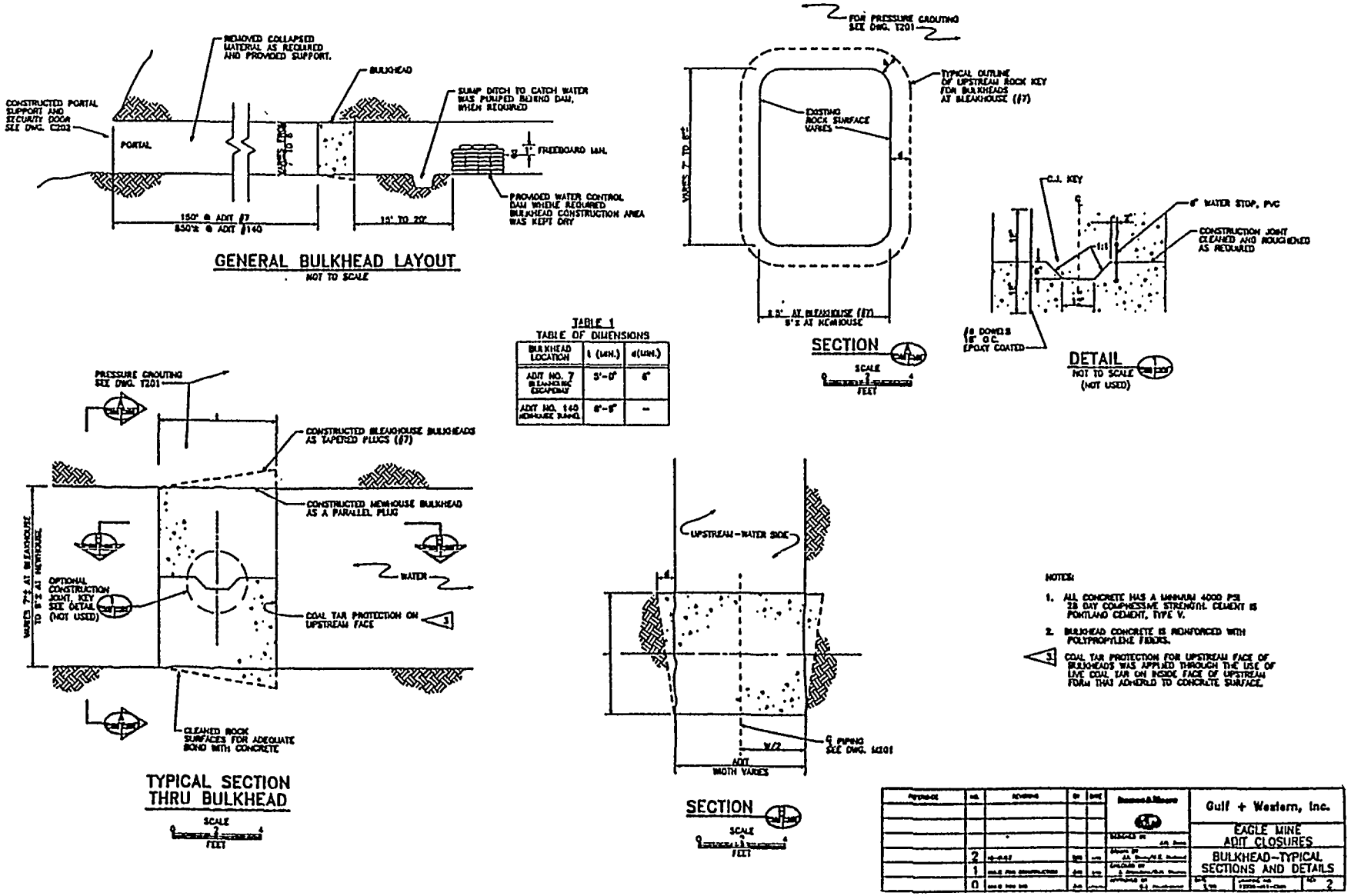


Figure C.3-3

Sections and Details for Plug No. 7

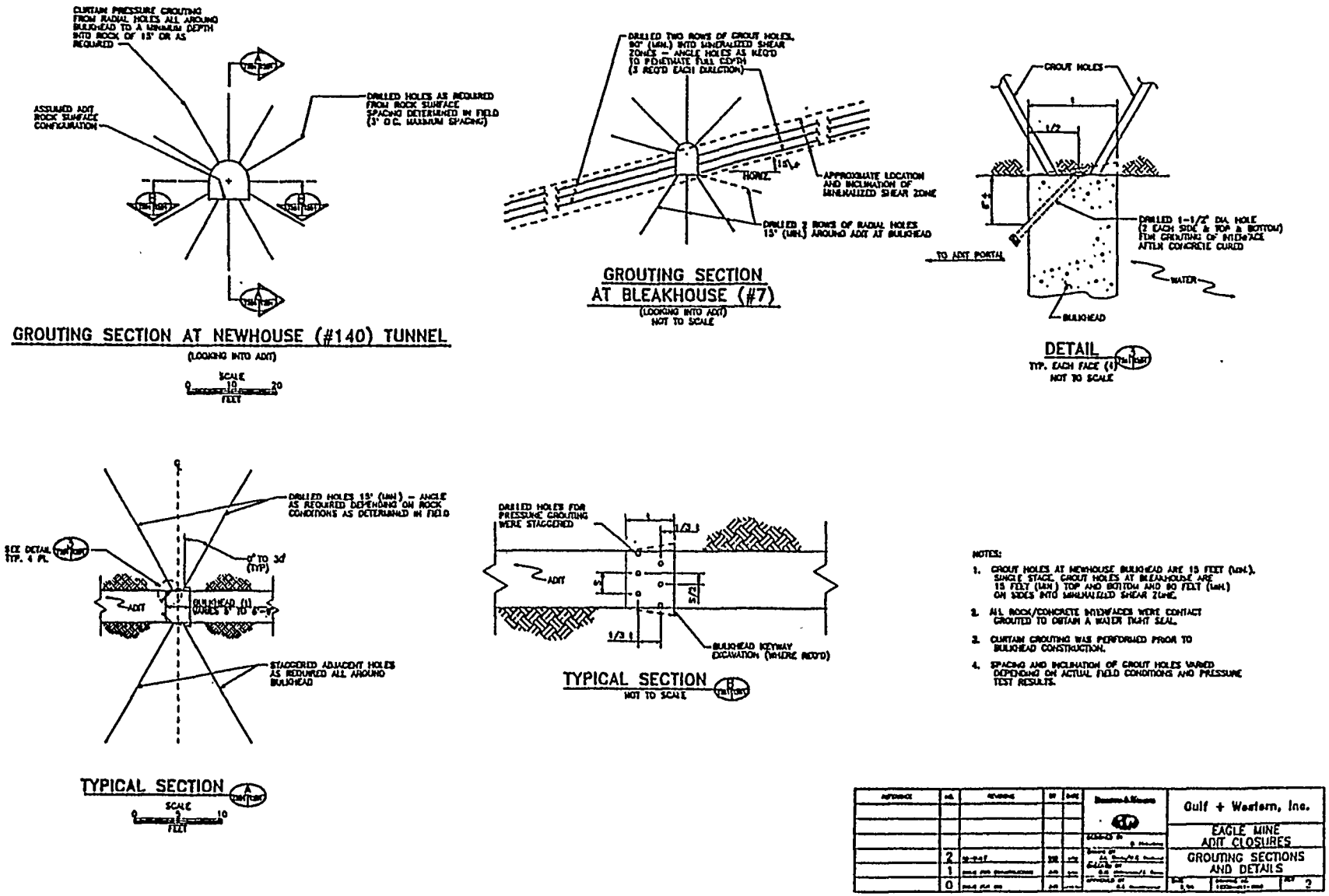


Figure C.3-4

Grouting Sections and Details for Plug No. 7

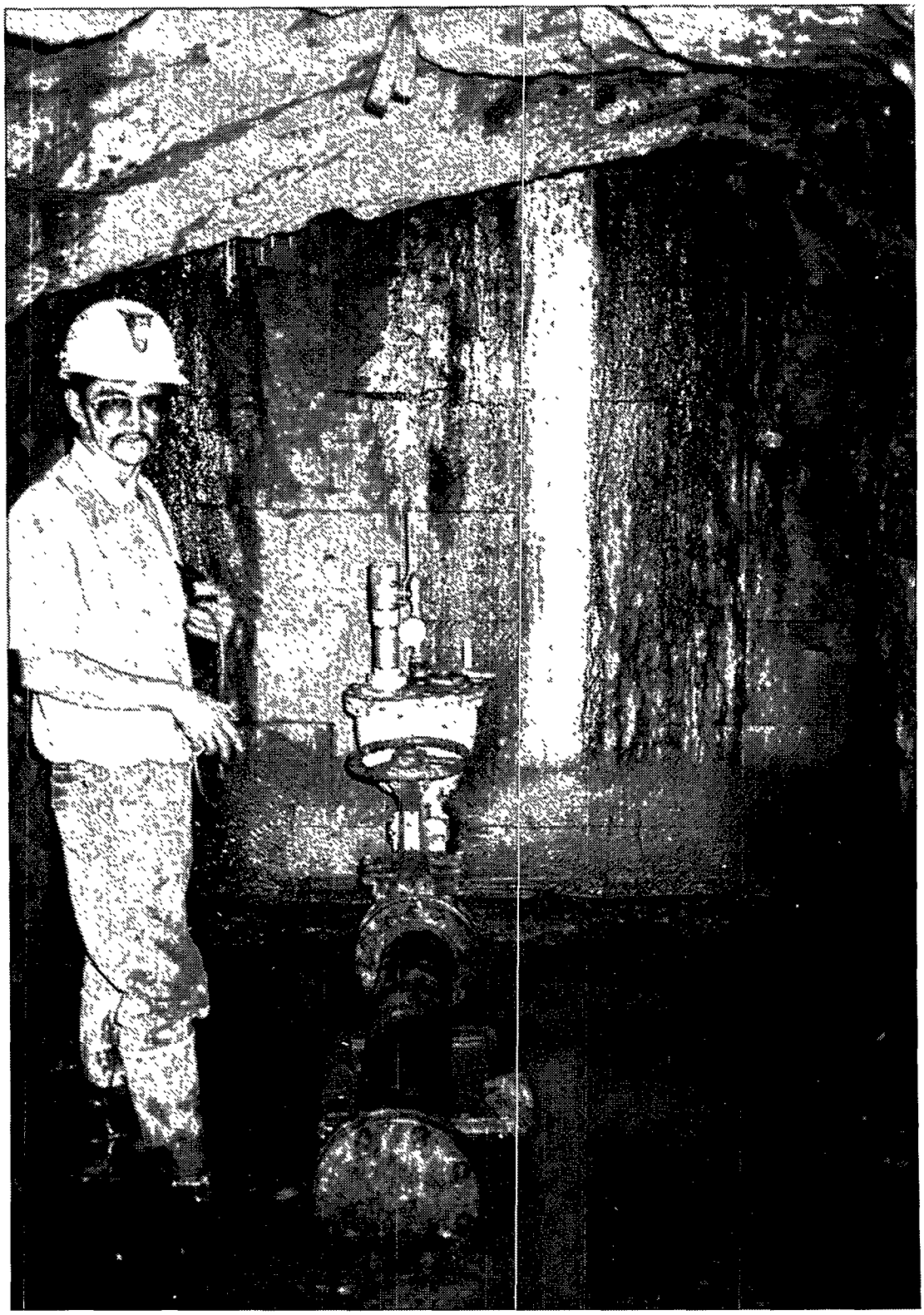


Figure C.3-5
Plug No. 6 Installation

Technology used to install the Eagle Mine plugs can be applied to some of the plugs at the potential repository. However, the conditions and criteria for the plugs will be different and may require different construction procedures.

C.3.6 References

Romberger, S. B. (1980). "Metallic Mineral Resources of Colorado," Colorado Geology. Rocky Mountain Association of Geologists, Denver, pp. 225-235. (NNA.940629.0005)

APPENDIX D

Estimate of Materials in Repository

APPENDIX D
Estimate of Materials in Repository

D.1 Estimate of Materials in Repository

Table D-1 shows potential materials remaining in the repository for the SCP-CDR design. Because of the present lack of detailed planning for Option 30, material quantities were not estimated for Option 30. Only materials not routinely removed from the underground operations are tabulated.

The total quantities of foreign materials were also allocated by area. Eight areas were identified, as shown in Table D-2. Distributions were made using engineering judgment where detailed takeoffs were unavailable. Table D-3 shows the distribution by area for each repository design option.

In addition to materials and substances, some amount of water may be introduced into the repository. The intent is to minimize the amount of service water used in drilling and tunneling operations, and the water used will be carefully collected and removed from the repository. Hence, the expected environmental impact of service water is expected to be minimal.

Large quantities of cementitious materials and steel will remain in the repository unless special procedures are implemented to remove them. Also, trace amounts of a variety of substances will remain. Further studies will be required to determine the potential impact of each of the foreign materials on the chemical environment of the waste package. Detailed repository design and planning must consider the effect of these foreign materials on the chemical environment.

Table D-1
Foreign Materials Remaining

	Materials Remaining In Repository	Estimated Quantities Remaining SCP/CDR	
	Rock Support		
1	Rock Bolts	21,400,000	lbs
2	Mats	NA	
3	Plates	3,600,000	lbs
4	Mesh	11,900,000	lbs
5	Grout	9,600	yds
6	Shotcrete	177,500	yds
7	Concrete	206,500	yds
	Bulkheads and Structures		
8	Roll up Door	700	pc
9	Air Door	100	pc
10	Gen Steel A36	47,600,000	lbs
11	Reinforcing Steel	200,000	lbs
	Consumables		
12	Diesel Fuel	200	gal
13	Engine Oil	300	gal
14	Hydraulic Fluid	600	gal
15	Gear Oil	200	gal
16	Lubricants	400	lbs
17	Tires	13,600	lbs
	Blasting		
18	ANFO Prill	372,300	lbs
19	Elect Blasting Caps	37,400	pc
	Misc. Unknowns		
20	Diesel Exhaust Residue	Trace	
21	Human Waste	Trace	
22	ANFO Fume Residue	Trace	
23	Spray Paint	Trace	
24	Misc. Organics	Trace	

Table D-2
Repository Areas by Usage

	Area	Drifts/etc.	Usage	Total Drift Lengths SCP/CDR (ft)
A	Ramps	North (Waste) South (Tuff)	Ventilation and Haulage	11230
B	Main Test Area	Misc.	Testing and Storage	0
C	Service Area	Shops, Warehouse, etc.	Storage and Service	7835
D	Ventilation	Perimeter Drift, Misc. Vent Drifts	Ventilation and Escapeways	33840
E	Mains	Waste, Service, Tuff, EW Crosscut	Haulage and Ventilation	33260
F	Emplacement	Emplacement Drifts	Haulage and Ventilation	524575
G	Exploration	Calico Hills Drifts	Testing and Observation	0
	Total			610740

Table D-3
Distribution of Materials in Repository

Materials Remaining In Repository	Estimated Percent of Quantities by Area SCP/CDR						
	A	B	C	D	E	F	G
Rock Support							
Rock Bolts	2%	NA	1%	6%	5%	86%	NA
Mats	NA	NA	NA	NA	NA	NA	NA
Plates	2%	NA	1%	6%	5%	86%	NA
Mesh	2%	NA	1%	6%	5%	86%	NA
Grout	2%	NA	1%	6%	5%	86%	NA
Shotcrete	2%	NA	1%	6%	5%	86%	NA
Concrete	1%	NA	1%	0%	2%	95%	NA
Bulkheads and Structures							
Roll up Door	0%	NA	60%	20%	20%	0%	NA
Air Door	0%	NA	60%	20%	20%	0%	NA
Gen Steel A36	20%	NA	60%	0%	20%	0%	NA
Reinforcing Steel	20%	NA	60%	0%	20%	0%	NA
Consumables							
Diesel Fuel	10%	NA	50%	10%	20%	10%	NA
Engine Oil	10%	NA	50%	10%	20%	10%	NA
Hydraulic Fluid	10%	NA	50%	10%	20%	10%	NA
Gear Oil	10%	NA	50%	10%	20%	10%	NA
Lubricants	10%	NA	50%	10%	20%	10%	NA
Tires	40%	NA	10%	0%	40%	10%	NA
Blasting							
ANFO Prill	2%	NA	1%	6%	5%	86%	NA
Elect Blasting Caps	2%	NA	1%	6%	5%	86%	NA
Misc. Unknowns							
Diesel Exhaust Residue	5%	NA	5%	80%	5%	5%	NA
Human Waste	10%	NA	40%	10%	30%	10%	NA
ANFO Fume Residue	10%	NA	10%	60%	10%	10%	NA
Spray Paint	20%	NA	50%	10%	10%	10%	NA
Misc. Organics	40%	NA	30%	10%	10%	10%	NA

APPENDIX

Information from the Reference Information Base Used in this Report

This report contains no information from the Reference Information Base.

Candidate Information for the Reference Information Base

This report contains no candidate information for the Reference Information Base.

Candidate Information for the Geographic Nodal Information Study and Evaluation System

This report contains no candidate information for the Geographic Nodal Information Study and Evaluation System.

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