EPA/600/R-05/071
April 2005

Mine Waste Technology Program

Underground Mine Source Control Demonstration Project

by:

Lynn McCloskey
MSE Technology Applications, Inc.
Mike Mansfield Advanced Technology Center
Butte, Montana 59702

Under Contract No. DE-AC09-96EW96405
Through EPA IAG No. DW89938870-01-1

Diana Bless, EPA Program Manager
Sustainable Technology Division
National Risk Management Research Laboratory
Cincinnati, Ohio 45268

This study was conducted in cooperation with
U.S. Department of Energy
Savannah River Operations Office
Aiken, South Carolina 29802

National Risk Management Research Laboratory
Office of Research and Development
U.S. Environmental Protection Agency
Cincinnati, Ohio 45268

Notice

The U.S. Environmental Protection Agency through its Office of Research and Development funded the research described here under IAG DW89938870-01-1 through the Department of Energy (DOE) Contract DE-AC09-96EW96405. It has been subjected to the Agency's peer and administrative review and has been cleared for publication as an EPA document. Reference herein to any specific commercial product, process, or service by trade name, trademark, manufacturer, or otherwise, does not necessarily constitute or imply its endorsement or recommendation. The views and opinions of authors expressed herein do not necessarily state or reflect those of the U.S. EPA or DOE, or any agency thereof.

Foreword

The U.S. Environmental Protection Agency is charged by Congress with protecting the Nation's land, air, and water resources. Under a mandate of national environmental laws, the Agency strives to formulate and implement actions leading to a compatible balance between human activities and the ability of natural systems to support and nurture life. To meet this mandate, EPA's research program is providing data and technical support for solving environmental problems today and building a science knowledge base necessary to manage our ecological resources wisely, understand how pollutants affect our health, and prevent or reduce environmental risks in the future.

The National Risk Management Research Laboratory is the Agency's center for investigation of technological and management approaches for preventing and reducing risks from pollution that threaten human health and the environment. The focus of the Laboratory's research program is on methods and their cost effectiveness for prevention and control of pollution to air, land, water, and subsurface resources; protection of water quality in public water systems; remediation of contaminated sites, sediments, and ground water; prevention and control of indoor air pollution; and restoration of ecosystems. The NRMRL collaborates with both public and private-sector partners to foster technologies that reduce the cost of compliance and to anticipate emerging problems. NRMRL's research provides solutions to environmental problems by developing and promoting technologies that protect and improve the environment; advancing scientific and engineering information to support regulatory and policy decisions; and providing the technical support and information transfer to ensure implementation of environmental regulations and strategies at the national, state, and community levels.

This publication has been produced as part of the Laboratory's strategic long-term research plan. It is published and made available by EPA's Office of Research and Development to assist the user community and to link researchers with their clients.

Sally Gutierrez, Director
National Risk Management Research Laboratory

Abstract

This report presents results of the Mine Waste Technology Program Activity III, Project 8, Underground Mine Source Control Demonstration Project implemented and funded by the U. S. Environmental Protection Agency (EPA) and jointly administered by EPA and the U. S. Department of Energy (DOE). Project 8 addresses EPA's technical issue of Mobile Toxic Constituents – Water – through a field demonstration at a remote, inactive underground mine.

This project was undertaken to demonstrate the feasibility of injecting a water-activated, expansive, flexible, and closed-celled source control material (grout) into a rock fracture system to reduce or eliminate the flow of acid mine drainage (AMD) into the underground workings of an abandoned mine. The Miller Mine, located in the Big Belt Mountains of Broadwater County, Montana, was selected for the field demonstration.

Grout injection was completed in two phases to reduce or eliminate AMD generated when groundwater contacts sulfide mineralization associated with the Miller Mine Reverse Fault and associated fractures. Phase I included drilling, coring, and grouting 10 approximately 624-feet drill holes in Precambrian sedimentary and Tertiary igneous rocks using core and Jackleg drilling methods. Results of Phase I work indicate that fracture flow was significantly reduced by approximately 77 +/- 5%, and the metals loading (particularly for arsenic, cadmium, copper, nickel, aluminum, and lead) was reduced by at least 80%.

Phase II grout injection included using Jackleg drilling and downstage grouting methods (approximately 400 feet) completed to seal rock fractures with observed flow that intercepted underground workings. Phase II work further reduced metals loading (zinc, iron, and copper) into the underground workings.

While the dissolved metal concentrations at the portal were not reduced, the mass loading of the major metals (iron, zinc, and aluminum) were reduced by an order of magnitude. This is the direct result of grout injection into the fracture system. The grouting reduced fracture flow and the metals load resulting from the diversion of fracture flow away from mineralized areas and encapsulation of the sulfide mineralization.

Contents

Contents (cont'd)

Contents (cont'd)

Note: Appendices A through F are available upon request from the MSE MWTP Program Manager. Please refer to document number MWTP-258. Email: mwtp@mse-ta.com, Phone: (406) 494-7100.

Figures

Page

Tables

Page

Acronyms and Abbreviations

Ag	silver
Al	aluminum
AMD	acid mine drainage
As	arsenic
Cd	cadmium
cm/s	centimeter per second
cps	centipoises
Cu	copper
DOE	U.S. Department of Energy
EPA	U.S. Environmental Protection Agency
Fe	iron
ft	foot
gpm	gallons per minute
HA	Hydro Active
ID	inside diameter
ICP-ES	inductively coupled plasma emission spectrometer
IDL	instrument detection limit
lb/d	pound per day
MCL	maximum contaminant level
MDL	method detection limit
Mg	magnesium
mg/L	milligrams per liter
mL/min	milliliter/minute
MMRF	Miller Mine Reverse Fault
Mn	manganese
MSE	MSE Technology Applications, Inc.
MWTP	Mine Waste Technology Program
Ni	nickel
OD	outside diameter
Pb	lead
ppm	parts per million
psi	pounds per square inch
QA	quality assurance
QAPP	quality assurance project plan
QC	quality control
RPD	relative percent difference
RQD	rock quality designation
TD	total depth
Tqd	tertiary quartz diorite
VLF	very-low frequency
Zn	zinc

Acknowledgments

This document was prepared by MSE Technology Applications, Inc. (MSE) for the U.S. Environmental Protection Agency's (EPA) Mine Waste Technology Program (MWTP) and the U.S. Department of Energy's (DOE) Savannah River Operations Office. Ms. Diana Bless is EPA's MWTP Program Manager, while Mr. Gene Ashby is DOE's Technical Program Officer. Ms. Helen Joyce is MSE's MWTP Program Manager.

Executive Summary

The Mine Waste Technology Program (MWTP), Activity III, Project 8, Underground Mine Source Control Demonstration Project was implemented and funded by the U.S. Environmental Protection Agency (EPA) and jointly administered by EPA and the U.S. Department of Energy (DOE). Project 8 addresses EPA's technical issue of Mobile Toxic Constituents – Water – through a field demonstration at a remote, inactive, underground mine. The Underground Mine Source Control Demonstration Project was performed to demonstrate the feasibility of injecting an innovative, nonrigid, source control material into the fractured rock system at an abandoned underground mine to reduce and/or eliminate the influx of acid mine drainage (AMD) into the underground mine workings.

In 1998, the MWTP selected the Miller Mine as a demonstration site for the field implementation and evaluation of an underground mine source control technology. Adit discharge ranged between 5 and 14.5 gallons per minute (gpm) and contained lead levels that exceeded the National Primary Drinking Water Regulations; aluminum, iron, manganese, pH, and sulfate at levels that exceeded the National Secondary Drinking Water Regulations; and nickel that exceeded the aquatic life standard for fresh water. The Miller Mine discharge is used as a source of drinking water for cattle and wildlife in the area immediately surrounding the Miller Mine.

Site characterization, source control materials testing/evaluation, and two phases of field source control emplacement were performed to define the optimal material and material application method for the demonstration. An expandable, closed-cell polyurethane grout material was selected. This material was designed and developed to act as a water-stop or water barrier system and to seal subsurface structures such as dams, tunnels, and sewers. This demonstration used Hydro Active Combi grout manufactured by De Neef, Inc. to seal the fractured rock system that was acting as a conduit for the discharge of AMD into the Miller Mine subsurface workings.

Site hydrogeological and geochemical conditions were monitored during the entire duration of the project. The main lower level of the Miller Mine consists of a 600-foot drainage/haul tunnel that forks approximately 70 feet from the end of the workings. During site characterization, it was determined that the main sources of influx occurred in the two drifts that fork from the main tunnel. The flow in one drift (W2), which contains a winze (a vertical workings starting in the floor of the workings), was fairly clean, containing minimal metals contamination and flows of approximately 1.5-gpm discharge from the drift. Before grouting, the second drift (W1 drift) contributed flows up to a maximum of 14.5 gpm, which exceeded most of the water quality regulatory levels. The main geologic structure controlling the flow at the end of the main workings was the Miller Mine Reverse Fault and the associated fracture system.

Phase I field source control technology injection was performed by core drilling grout injection holes over the top of the W1 drift and perpendicular to the fracture system to reduce the inflow of water. As a result of the Phase I grout injection, flow was reduced from the W1 drift by an average of approximately 77 +/- 5% over 10 months.

Equipment used during the Phase I grout injection was too large to use inside the W1 drift. After the initial grout injection, water still continued to seep from the walls through fractures in the W1 drift, especially on the northeastern wall of the underground mine workings. A decision was made to perform Phase II work by grouting small seeps flowing from close fractures in the mine wall to determine if the flow from the W1 drift could be further reduced or eliminated. The seeps were not eliminated, and the

influx through the small fractures migrated to different fractures after grouting was performed. The maximum reduction in flow after both injections was approximately 77 +/- 5%. From long-term monitoring results taken after grouting, the maximum recorded total flow from the Miller Mine adit was approximately 3 gpm, where 47% of the flow was from the W2 drift and 53% of the flow was from the W1 drift. Due to the reduction in flow from the W1 drift, the average percent reduction of dissolved metals loading over the duration of the project and the metals loading rates were significantly reduced. Average dissolved metal loading reductions of greater than 80% were obtained for cadmium, aluminum, zinc, and iron. Reductions of greater than 50% were obtained for manganese, lead, nickel, and copper. Iron loading, for example, was reduced from 7.5 pounds (lb)/day to 0.12 lb/day at sample port W1.

As a result of the technology application, only iron, lead, and manganese remain slightly above the National Secondary Drinking Water Regulation levels at 0.4, 0.04, and 3.26 parts per billion, respectively. As with the flows, these concentrations are much less than the original concentrations. An approximately 50% reduction in metals loading was achieved for all of the metals analyzed; for zinc and iron, the reduction was greater than 90%.

Overall, the application of the technology reduced the flow and the metals loading from the Miller Mine. However, most of the reduction in flow was achieved and resulted from the initial grout injection performed in 1999. The second grout application did not further reduce flows because the small, interconnected fractures controlling the seeps/flows into the underground workings were difficult to grout. Also, the flow would shift from one location to another because the rock was sheared and very weak. The main positive result from the second grout injection was the reduction of metals loading at sample point W1 and, to a lesser extent, at sample point W4 when comparing data from analogous seasons.

1. Introduction

1.1 Project Description

Mine Waste Technology Program (MWTP) Activity III, Project 8, Underground Mine Source Control Demonstration Project was funded by the U.S. Environmental Protection Agency (EPA) and jointly administered by EPA and the U.S. Department of Energy (DOE) through an Interagency Agreement. EPA contracted MSE Technology Applications, Inc. (MSE) through the MWTP to develop and evaluate a source control grouting technology that could be applied at abandoned mines to seal flows into the underground mine workings.

The objective of MWTP Activity III, Project 8 was to demonstrate the feasibility of source control materials for hydrogeological control to reduce water influx and minimize the production of acid mine drainage (AMD) at a nonferrous metal mine. The Miller Mine, located in the Confederate Mining District in Broadwater County, Montana, was the mine selected for implementation of the technology. The source control material selected for the demonstration was Hydro Active (HA) Combi grout, a closed-celled, expandable polyurethane grout that is flexible and can be injected under cold and submerged conditions. Details concerning the material selection process are provided in the MWTP Activity III, Project 8, Phase I, Site Characterization and Materials Testing Report (Ref. 1).

Application of this technology involved injecting the polyurethane grout into the rock-fracture system associated with the underground mine workings, thereby reducing the amount of ground and surface water infiltrating the mine workings and system. Reducing water inflow into the underground mine workings was expected to reduce the volume of impacted water discharging from the lower mine adit. However, these activities also have the potential to improve surface water quality downstream from the mine. The Miller Mine site, selected for this demonstration project, is located approximately 20 miles north of Townsend, Montana. Presently, slightly acidic waters containing elevated levels of heavy metals discharge from the mine adit directly into Greenhorn Gulch.

The Underground Mine Source Control Demonstration Project consisted of three major phases: (1) site characterization of the Miller Mine, (2) materials testing, and (3) field demonstration of the selected technology including verification monitoring and technology evaluation. This document is the final report and will address Phase III, Field Emplacement. However, all pertinent information pertaining to field emplacement that was obtained during other phases of the project will be addressed in this document. Information from all phases of the project was used to evaluate the effectiveness of the technology.

Work associated with Phase I included characterizing the Miller Mine site. The purpose of Phase I was to define characteristics of the mine site prior to technology application, including:

- sources of water infiltrating the underground workings;
- flow rates and water quality within the underground mine workings;
- the hydrogeological system of the mine site;
- historical, mineralogical, and structural geology of the mine system; and
- hydraulic connections within the underground mine.

Phase II involved performing materials testing of approximately 50 source control materials to determine which material would be viable for application during Phase III. Phase II testing was performed at MSE's testing facility and at IT Geotechnical Laboratory, Inc.
Phase III activities that are addressed in this report are given below.

- The site description and background information are presented in Section 1.

- A preinjection site characterization summary is presented in Section 2.

- The preliminary grout injection procedures and requirements are provided in Section 3.

- The general approach for the 1999 technology field emplacement and investigation is presented in Section 4.

- The general approach used for the 2001 technology field emplacement and investigation is presented in Section 5.

- Monitoring results and evaluation for the entire duration of the project are provided in Section 6.

- A summary of quality assurance project plan (QAPP) activities is provided in Section 7.

- The conclusions derived from the field program, previous work performed, and recommendations for future projects of this type are in Section 8.

- A list of references used in the document is recorded in Section 9.

1.2 Technology Background

1.2.1 Technology Background

The Miller Mine is an abandoned gold mine located in steep terrain on the western slope of the Big Belt Mountains in Broadwater County, Montana (SW1/4, SE1/4, Sec.13, T10N, R2E) between Greenhorn Gulch and Montana Gulch (Figure 1-1). Mine property is on the Superior Claim in the Confederate Mining District approximately 20 miles northeast of Townsend, Montana, at an altitude of 6,400 feet (ft). The mine includes two working levels, i.e., an upper and lower mine workings (Figures 1-2, Figure 1-3, and Appendix A, Plate 1). The lower adit drainage is slightly acidic while the upper workings are generally dry (Figure 1-3). The discharge from the Miller Mine flows into Greenhorn Gulch, a tributary of the creek in Confederate Gulch, which eventually empties into Canyon Ferry Reservoir (Figure 1-4).

Mine hazards at the Miller Mine site include two open adits, several waste dumps, numerous structures in various stages of disrepair, and discarded equipment. Approximately 12,000 cubic yards of disturbed material and waste rock are associated with the Miller Mine site (Figure 1-3.)

1.2.1.1 Site History

Lode claims in the Confederate Gulch area were first staked by Henry O. Miller in 1893. Mining continued at the Miller Mine until the early 1940s with the Miller Mine being the largest lode gold producer in the area. Approximately 4,000 ounces of gold ($80,000) were produced from the Miller Mine between 1910 and 1930. The mine was reopened after World War II and operated intermittently until the early 1960s. Except for exploratory drilling in recent years, very little activity or mining activity has been recorded in the area since the early 1960s.

1.2.1.2 Physiography

Terrain around and in the vicinity of the Miller Mine is steep and slightly wooded with some vegetation. Narrow, steep, and unpaved roads provide vehicle access to most areas of the mine. Winter access to the site is difficult due to deep snow and steep terrain, which impedes sampling efforts outside the mine during the winter months (Figure 1-3). During the winter months, the Miller Mine discharge freezes approximately 50 ft in from where the discharge exits the mine portal.

1.2.2 Project Objectives and Scope of Work

Most waters in the vicinity of the Miller Mine have a pH close to neutral and do not carry a large percentage of metals or suspended solids before infiltrating the subsurface fracture systems associated with underground mine workings. The

AMD forms when infiltrating water contacts an oxidized sulfide ore zone as it travels through the associated fracture system. Acid mine drainage typically is caused by the oxidation of sulfide mineralization. Especially problematic is when iron disulfide (pyrite) comes in contact with water. This chemical reaction results in increased acidity (lower pH) of the water and increased mobility of metals in water.

The objective of the Phase III field demonstration emplacement was to inject an innovative source control material into the rock fracture system and reduce or eliminate AMD by limiting the infiltration of water through hydraulic connections associated with the fracture system at the Miller Mine. The source control material was injected in order to fill and seal fractures that acted as hydraulic conduits, to reduce the amount of water infiltrating the underground workings, to prevent water from coming in contact with oxidized host rocks, and ultimately, to reduce or eliminate the generation of AMD and mobilization of heavy metals. Figure 1-5 provides a visual depiction of areas of AMD at the Miller Mine prior to the application of the technology.

This project demonstrated the effectiveness of an experimental, closed-celled, expandable, polyurethane grout as the innovative source control material. The grout was injected into the localized rock-fracture system to seal and reduce water flow into sections of the underground mine workings. The adit discharge and the discharge from each drift was continuously monitored before and after grout injection to help determine if the grout had sealed the localized fracture system and reduced the AMD in the mine.

1.2.3 Technology Criteria

Criteria for the success of the polyurethane grouting technology was established to define and measure the degree of success the technology application was able to achieve. Water-injection field tests were performed prior to establishing the hydraulic baseline conditions at the site before grouting. Postgrouting tests were performed, and results were compared to the baseline conditions to determine the performance of the polyurethane grout with regard to hydraulic sealing of the fracture system. The primary objective used to define the success of the project is noted below and is also addressed in the QAPP (Ref. 2). The specific Phase III objective was to show that the injection of the polyurethane grout reduced the cumulative volumetric flow by 95% at sample port W1 (a 60-degree trapezoidal flume). The initial volumetric flow at W1 was determined using flow data from the 10 months prior to grout emplacement, and the final volumetric flow was determined using the flow data from the 10 months following grout emplacement (Appendix A, Plate 1).

Because of seasonal variations in the flow at the W1 flume (i.e., high spring flow and low winter flow), the achievement of the objective was also evaluated on a seasonal basis. Cumulative flows for each 2-month period before grouting were compared to the corresponding 2-month period after grouting to show the near-term effect of the grout emplacement.

A secondary criteria for the success of Phase III included improving the quality of water exiting the Miller Mine adit by decreasing dissolved metals concentrations and increasing the pH.

Although the main objective of the demonstration was to control point-source influx, the evaluation of the project’s success also included the feasibility, cost-effectiveness, and flexibility of the technology to be used in other situations and other applications.

1.2.4 Demonstration History of Field Emplacements

Site characterization of the Miller Mine began in August 1998 and was completed October 1999. Fieldwork consisted of mine mapping, core and borehole drilling, water injection testing, and grout injection. Technology emplacement fieldwork was performed in two stages.

Stage I – Field Emplacement 1999 - The initial technology emplacement began during the week of September 13, 1999, and was completed by October 21,1999. For design and characterization prior to and during the emplacement, numerous water injection and dye tests were performed on several of the holes to determine the hydraulic conductivity of the fracture system and to help predict which fractures influenced the rock fracture system the most during grout injection. Fieldwork included drilling, coring, and grouting 10 core holes or drill holes (M1-M9 and MJ10) (Appendix A, Plate 2). A total of 624 ft of drilling was completed during this period that included 617 ft of coring. Injection grouting was associated with core holes M1 through M9 located between W1 and W2 in the lower working level of the mine. Borehole JK10 (7 ft in depth) was not core-drilled and is located northeast of W1 (Appendix A, Plate 2).

Stage II – Field Emplacement 2001 - The second technology emplacement began on April 16, 2001, and was completed by April 25, 2001. Fieldwork included Jackleg drilling and grouting 43 short holes that were drilled in a radial pattern. A total of 400 ft of drilling was completed during this period, and 65 gallons of grout were injected into these holes

Following the grout emplacement fieldwork, field monitoring, field sampling, and data evaluation continued through November 2003.

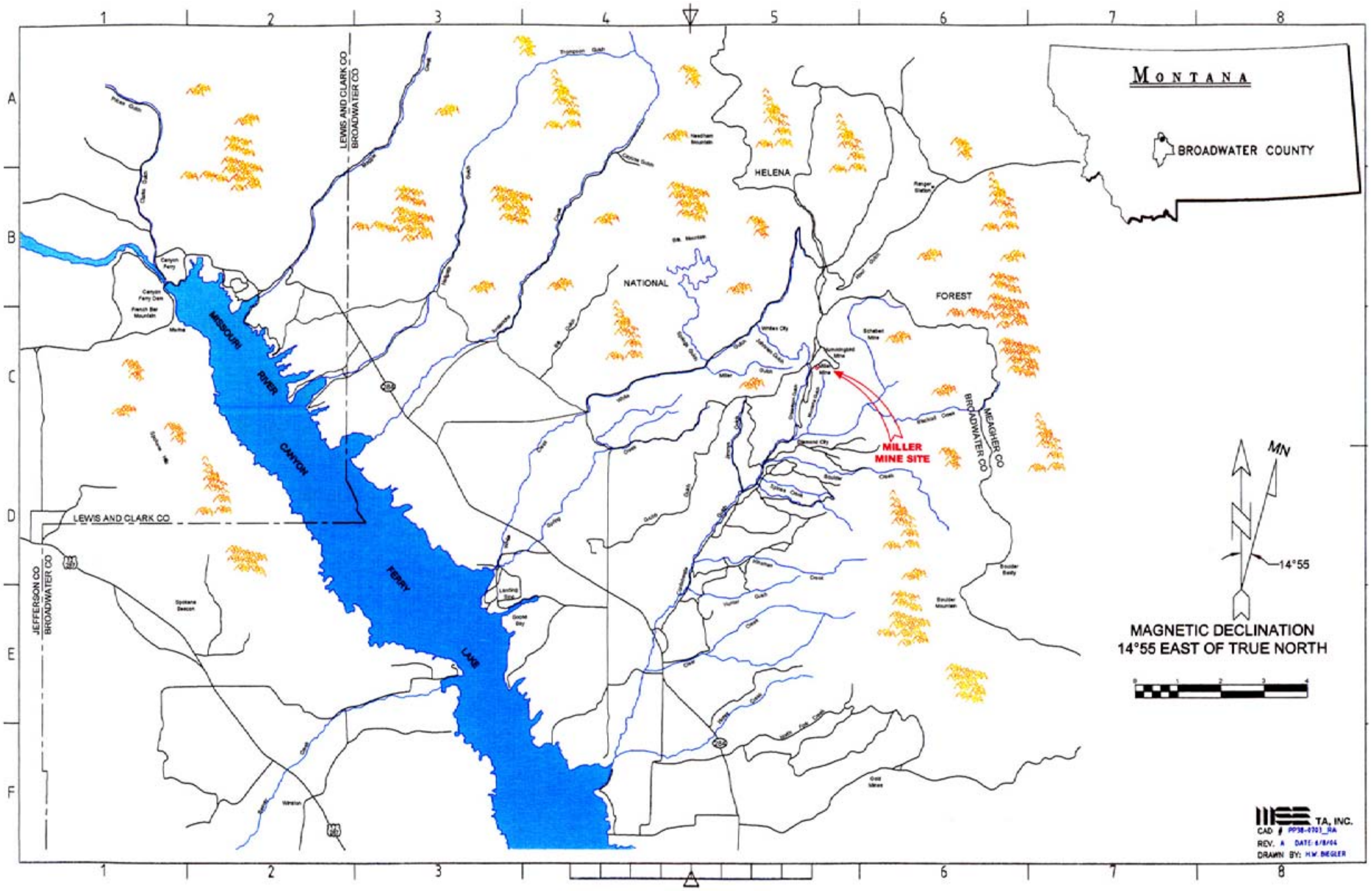


Figure 1-1. Miller Mine map.

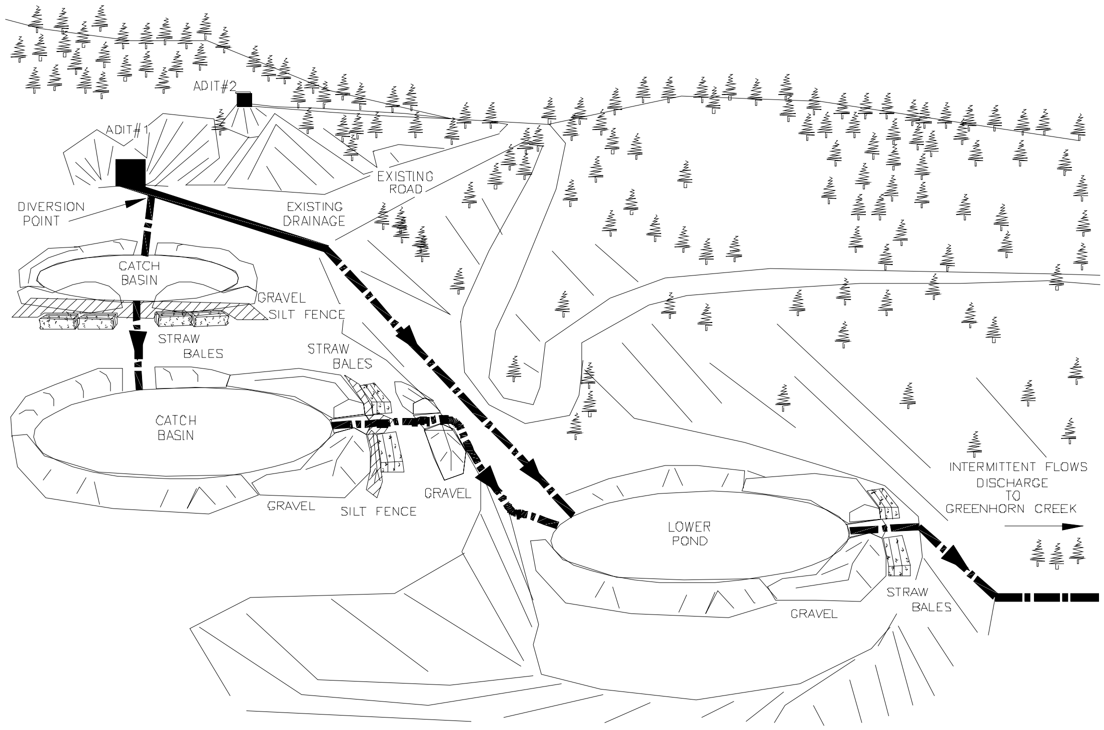


Figure 1-2. Generalized site map of the Miller Mine.

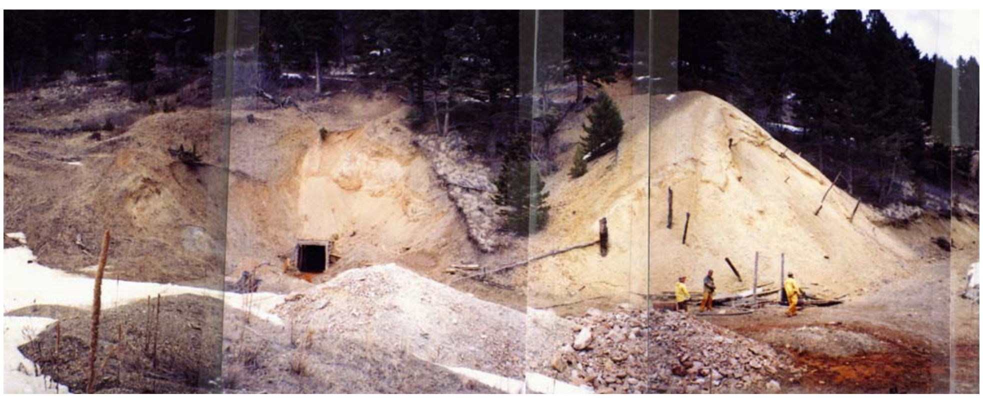


Figure 1-3. Photo of the Miller Mine upper and lower workings.

Figure 1-4. Photo of the Miller Mine lower adit discharge prior to implementation of the source control technology.

Figure 1-5. Areas of AMD at the Miller Mine prior to the application of the technology.

2. Preinjection Site Characterization Summary

The primary purposes for characterizing the underground mine workings were to provide baseline information to determine where inflows into the mine system occur, which structures control the inflow, and how much inflow was reduced as a result of the grout injection. Geological, hydrogeological, geophysical, and water quality information collected during the mine characterization were used for evaluation of the source control technologies, definitive design, and field emplacement of the source control material. The methods used to acquire both background information and information for the evaluation process are presented in the Activity III, Project 8, Site Characterization and Materials Testing Report (Ref. 1), the QAPP (Ref. 2), and in Sections 2 and 3 of this report.

2.1 General Information, Surveys, and Observations

A topographic map and an underground mine map were created from surveys completed at the Miller Mine site. Mine mapping included completing underground mine, geologic, and hydrogeologic surveys and constructing maps of both the upper and lower workings of the Miller Mine (Figure 2-1). A plan map showing the underground mine workings, the continuous hydraulic monitoring stations, and the main geological structure is provided in Figure 2-1 and Appendix A, Plate 3.

2.2 General Site Geology

Tertiary igneous intrusive rocks (generally quartz and granodiorites) intruded Precambrian limestones and shales of the Newland Formation (Belt rocks) to form Miller Mountain in the Confederate Gulch and White Gulch area of Broadwater County, Montana (Ref. 3). Structure and topography of this area are the result of the late Cretaceous and early Tertiary Laramide orogeny that gave rise to folding, faulting, and igneous intrusion. As a result, the dominant structure affecting Precambrian rocks in the area is the York Anticline (trend N35W). This structure has been modified by successive reverse faulting generally associated with the forced emplacement of Tertiary igneous rocks that formed Miller Mountain.

It is believed that the Miller Mountain intrusion may have been emplaced in at least two stages (Ref. 3). After the first period of intrusion, a steep reverse fault developed that may be related to the Miller Mountain Reverse Fault (Appendix A, Plate 1). A second period of intrusion occurred after the formation of the reverse fault that may account for the secondary fracture pattern described in this report. The secondary fracture-shear pattern is likely related to later magmatic injection into bedding planes and into preexisting fault planes. This magmatic activity is likely responsible for detachment, assimilation, and shearing of Belt rocks into relatively small remnants or residual masses of recrystallized Belt rocks enveloped by intrusive rocks.

Mineralization in the Miller Mine underground workings is associated with contact deposits along or near the igneous-sedimentary contact. Generally, the attitude of this contact within the Miller Mine is N25W, 30SW.

2.3 Miller Mine Hydrogeology

Acid mine drainage discharge from the lower adit likely originates from the Miller Mine Reverse Fault (MMRF), associated fractures, and from the secondary fracture-shear pattern related to detachment and assimilation of Belt sediments. These faults and associated fractures act as conduits transporting groundwater through overlying Precambrian Belt rocks and its weathered equivalent into the underlying fractured Tertiary quartz diorite (Tqd) (igneous intrusive rocks) and lower mine workings. Results of geologic mapping indicate that parts of the fault plane have been injected with Tqd (Appendix A, Plate 3). This likely occurred during the second stage of intrusive activity during magmatic injection of Tqd into bedding planes and the associated detachment and assimilation of Belt rocks. Subsequent faulting and fracturing is thought to have brecciated the Tqd and Precambrian Belt rocks, creating fractures and conduits through mineralized and altered rocks rich in pyrite and other metallic sulfides.

2.3.1 Underground Mine Flow Regime

The lower workings of the Miller Mine is a 600-ft haul tunnel designed for the removal of ore from the upper workings and two attached drifts. The drifts for this demonstration (W1 and W2) were designated by the sample locations, which were located at the mouths of the drifts (Figure 2-1). The W1 drift is 68 ft long and was mined mainly in the Tqd (Figure 2-1 and Appendix A, Plate 3). The last 30 ft of the W1 drift runs parallel to the Miller Mine fault system, and the flow was measured using a small, 60-degree trapezoidal flume manufactured by Plasti Fab, Inc.

The W2 weir was located at the mouth of the W2 drift and measured combined flows from the W3 weir as well as the flow from a winze that is located between sample ports W2 and W3. This winze is angled downward approximately minus 30 degrees to the north and was driven parallel under the length of the northernmost section of the W2 drift (Figure 2-1).

Pregrouting flow measurements show that only a fraction of the total discharge measured downstream near the adit portal at W4 is fromW2. The average flow recorded at W2 was approximately 1.15 gallons per minute (gpm) (Appendix B). It should be noted that the flow measurements at W4 could not be recorded during the winter months and records the total discharge from the Miller Mine adit portal, which is a combination of W2 and W1 flows.

2.3.2 AMD Mine Discharge and Water Quality

Metallic sulfide mineralization mapped in the upper working level shows altered Belt rocks replaced by pyrite and commonly altered to clay. Rock core, logged during coring operations, confirmed that sulfide mineralization and alteration were common in Belt and Tqd rock (Appendix C). As groundwater and oxygen migrate through the mineralized fracture zones, metals leach and acid forms, resulting in AMD flowing into the lower underground mine workings. As a result, elevated concentrations of dissolved metals (Table 2-1) and reduced pH levels are measured in the discharge water from the W1 drift.

In contrast, Table 2-1 shows that the dissolved metal concentration of water discharging through the W2 weir, especially zinc (Zn), iron (Fe), nickel (Ni), aluminum (Al), and manganese (Mn), were several orders of magnitude less than waters associated with the W1 drift. Using the information from the underground geologic and hydrogeologic surveys, it was determined that water flows over calcareous Precambrian Belt rocks (limey shale and siltstone) toward W3 and W2 and remains neutral, resulting in the water quality of flow from the W2 drift being below the maximum contaminant level (MCL) for most metals. As shown in Figure 2-1, it is not likely that any water infiltrating the northernmost upper workings comes in contact with mineralized rocks associated with the igneous-sedimentary contact zone; the mineralized rock is located below the upper workings between core holes M6 and M4 (Appendix A, Plates 1 and 3).

Comparing the water quality prior to technology emplacement between W1 and the adit discharge measured at sample port W4 shows a decrease in the concentration of dissolved metals (Zn, Mn, Fe) and an increase in the pH measurements. The data indicate that AMD and the concentration of dissolved metals were reduced during the transit between W1 and W4. Prior to implementation of the technology, AMD exceeded the MCL criteria for lead (Pb), and the secondary MCL for Fe at W4.

2.4 Geophysical Surveys

Geophysical surveys were conducted by DOE's National Energy Technology Laboratory to delineate water-filled fracture zones or surface-connected mine voids that may underlie the survey area at the Miller Mine. Initially, prior to technology application, very low frequency (VLF) and vertical-gradient magnetic surveys were conducted as a reconnaissance effort to delineate areas where surface water may be entering the underground mine workings. Geophysical data from the surveys was used to:

- identify fault/fracture zones or detectable mine voids (i.e., stopes, drifts, etc.);
- provide a baseline geophysical signature that could be used to evaluate subsequent intervention activities (e.g., grouting);
- determine the direction of water flow in the fracture along with the areas of high mineralization;
- identify areas where additional work would be needed; and
- identify the direction of groundwater flow after the grout had been injected into the underground workings.

The geophysical surveys performed prior to the technology emplacement were conducted above the underground gold mine using a 60- by 180-meter grid that was laid out on the steep slope overlying the mine. This work produced geophysical contour maps of the subsurface that correlated the mine workings to the areas that had the greatest potential for mineralization and water-bearing fractures (Appendix D).

After the technology was applied, vertical-gradient magnetic surveys were conducted, and data were collected using a Geonic EM34-#XL conductivity meter at intercoil spacings of 10, 20, 40, 50, 70, and 80 meters using both horizontal and vertical dipole orientations. The EM data was used to model the apparent conductivity for the block of earth underlying the surveyed grid. The modeling was done using Emigma software in which the conductivity images are configured as slices suspended at specific depths below and following the topography. Conductivity zones were interpreted to indicate areas of fractured rock where the increased conductivity was determined to be from increased water content (AMD), alteration (clay minerals that are conductive), or sulfide mineralization (conductive sulfide minerals) (Appendix D).

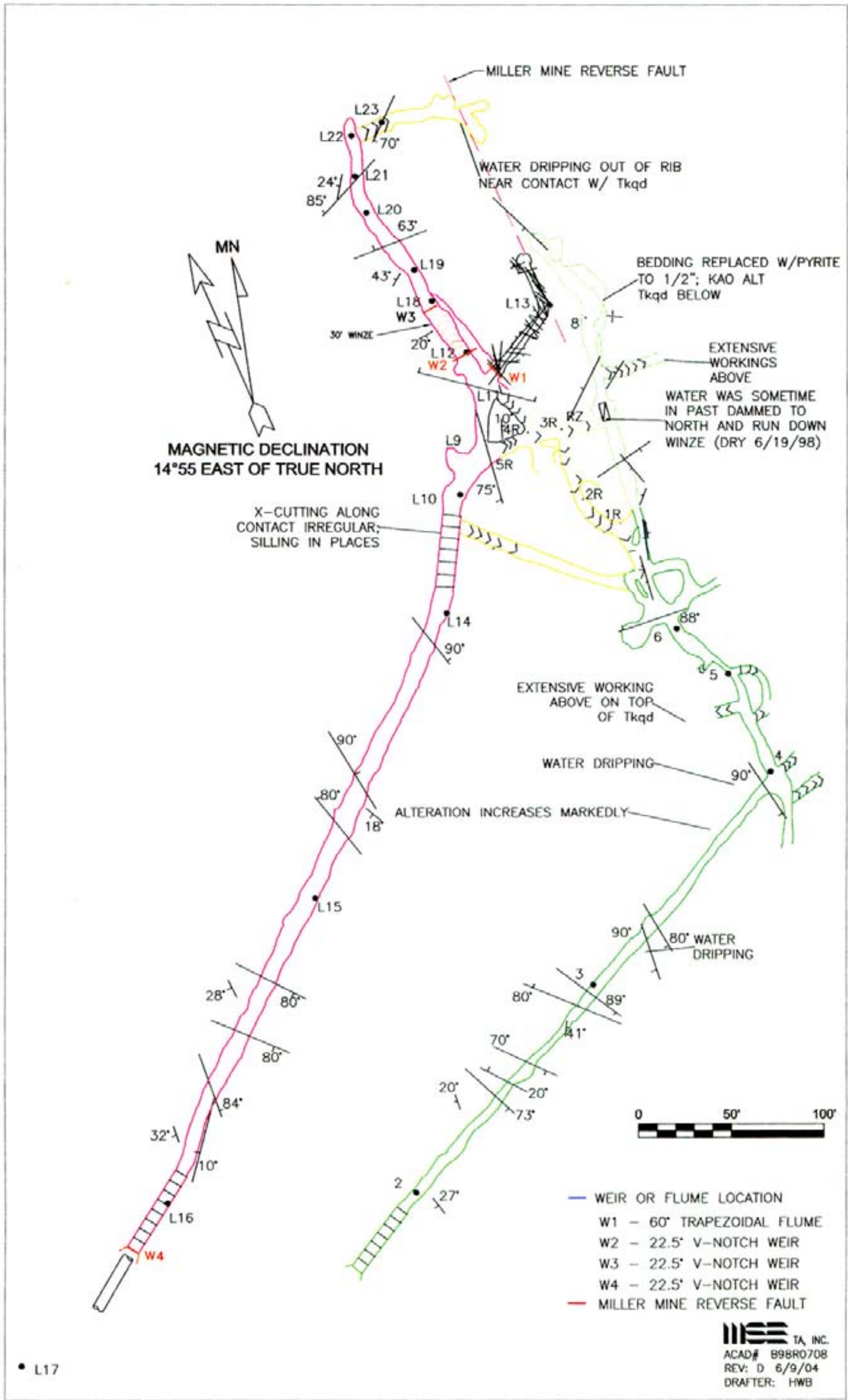


Figure 2-1. Initial Miller Mine underground workings plan map showing the weir, flume, and sample port locations W1, W2, W3, and W4 (AutoCAD drawing B98R0708)

Table 2-1. Background Water Quality for the Miller Mine Prior to Field Emplacement of the Source Control Technology

MWTP – Field Sampling Comparison
Site: Miller Mine
MWTP, Activity III, Project 8
Sample Location: Comparison of All Sample Locations

Sample Date:	7/8/1999	7/8/1999	7/8/1999	7/8/1999	MCL & Secondary MCL, mg/L
Sample #:	W1	W2	W3	W4	
Sample Matrix:	Water	Water	Water	Water	
Mine Inflow or Outflow	Inflow	Inflow	Inflow	Outflow	
pH (su)*	6.08	6.87	7.50	6.58	6.5 - 8.5
Sulfate*	828	295	445	539	250
Dissolved Metals (mg/L, ppm)					
Al	0.145	0.012	0.010	0.010	0.050 - 0.200*
As	0.025	0.025	0.025	0.025	0.050 until 1/06
Cd	0.003	0.003	0.003	0.003	0.005
Cu	0.002	0.002	0.002	0.002	1.000*
Fe	81.600	1.330	0.014	18.600	0.300*
Pb	0.023	0.023	0.023	0.023	0.015
Mn	5.970	0.594	0.072	3.220	0.050**
Ni	0.097	0.010	0.010	0.046	0.088**
Zn	0.915	0.057	0.013	0.388	5.000*
Ag	0.004	0.004	0.004	0.004	0.100*

* Secondary MCL

** Aquatic Life Standard for fresh water

3. Grout Injection Procedures

3.1 Introduction

Injection of a water-activated, expandable, polyurethane grout into the fracture system in the underground mine was completed in two different stages, first in October 1999 and then in April 2001. The 1999 technology application included core drilling and grouting 10 holes between the W2 sampling port and W1 sampling port (Appendix A, Plate 2). The April 2001 technology application complemented the first injection and was performed to further reduce AMD drainage from the W1 drift (i.e., drips, weeps). All October 1999 work involved drilling over and under the W1 drift. The April 2001 work involved Jackleg drilling and grouting holes drilled directly in the W1 drift adjacent and into the MMRF. The following sections describe core and rock drilling core recovery, water injection tests, grout formulations, the grout injection system, the grout injection process, and the monitoring performed to observe the affects of grout injection. A discussion of the performance of the test, the water and dye injection observations, and the grout injection are presented in Section 4.

3.2 October 1999 Core Drilling and Recovery

Drilling and coring was accomplished using a Hagby 1000 core drill and Gardner-Denver Model 83 Jackleg drill. Rock core from nine core holes was recovered and described (M1, M2, M3, M4, M5, M6, M7, M8, M9) (Appendix A, Plate 2). An additional hole was drilled using the Jackleg drill (MJ10). Rock core was not recovered from MJ10.

Approximate drill locations were marked along the eastern rock face between mine survey points L11 and L12 (core-drill locations) and between L11 and L13 (MJ10) (Appendix A, Plate 2). Respective bearings of each core hole were determined by orienting the longitudinal axis of the Hagby core drill parallel to a string stretched between two spads attached on the eastern and western faces. The compass bearing between respective spads was determined using a Brunton compass prior to the setup and arrival of the core drill and associated equipment.

Setup for each core hole included fastening the front frame of the drill rig to the rock face using two Dywidag rock bolts (approximately 1-inch-diameter threaded steel) embedded approximately 4 ft deep into the rock face with Fastloc epoxy. This setup provided stability for the core drill and ensured that coring operations would be successful and safe.

Water circulation was established and used as the drilling and coring fluid. Water was pumped from the flooded winze located north of the W2 weir into a plastic reservoir and then drawn using a hydraulically powered reciprocating pump and injected through the tools of the drill string and bit. Fluid pressure and flow rate were regulated on the Hagby control panel to achieve the desired flow circulation, feed rate of bit, bit pressure, and rotation rates of the drill rods and diamond bit. These parameters, depending on the size and type of drill tools, were optimized for the best core recovery. Drill cuttings were flushed from each hole by spent drilling fluids.

3.2.1 Recovery of Orientated Core

Continuous and orientated core was recovered from three core holes (M1, M2, M3) using the clay-impression method and associated tools (Appendix C). Orientated core was recovered using a clay core orientor tool constructed from the inner tube of a core barrel. This tool is weighted with Pb the full length of one side of the barrel, which forces the weighted side down. Orientation was accomplished by placing potter's clay on the shoe of the orientation tool, inserting it in the hole, and then forcing the clay to make an imprint of the base of the hole. This impression was used to align the orientation of the next core sample taken with that of the last core run. Recovered core was subsequently laid out in a split metal tray and its "up" orientation was scribed the full length of the core run. Occasionally, it was not possible to orientate or scribe the entire core run due to intervals that were severely fractured, altered, or weathered.

3.3 Water Injection and Dye Tests

Water injection tests were performed in four core holes (M1, M2, M3, and M8), and dye tests were performed in two core holes (M3 and M4) using a blue food-grade dye. These tests were done for design purposes to determine the relative hydraulic conductivity and the interconnection of the fracture system (Appendix A, Plate 4).

Generally, water was pumped into a completed core hole through a packer system configured for the desired test interval(s) for water injection testing (Ref. 4). Pressure and flow rate were regulated to achieve the desired test parameters for each unique testing interval. Water used for testing was pumped from the winze located north of Weir 2 into a plastic reservoir tank. Water was drawn from the reservoir tank with a hydraulically powered pump and transferred into the respective core holes through the packers (behind or between) and injected through 1/2-inch inside diameter (ID) perforated steel pipe (perforations on 6-inch centers) (Figure 3-1). Pressure was regulated at the Bean 20 pump with a built-in pressure control valve. A hydraulic motor control on the Hagby control panel regulated the flow rate of the reciprocating pump for water testing, and a Great Plains Industries, Inc. electronic digital meter was used to record total water volume per test. A glycerin-filled pressure gauge was used to record test pressure.

3.4 Grout Formulations and Emplacement

Grout was injected into core holes as specified through a grout delivery system that consisted of a plastic reservoir tank "fill-rite" flowmeter connected in-line to the reservoir outlet (suction side of pump), and a hi-con dual-action piston pump (Figure 3-2). The grout delivery system was used to inject grout into the respective core holes through the packer assemblies and at the desired interval. An additional pumping system was used to supply and mix water (necessary for grout activation) in tandem with grout emplacement at the core hole standpipe.

The water delivery system consisted of a sump pump placed in the winze located north of weir 2, which conveyed water into a plastic tank. Water was drawn from this tank with an air-powered Bean pump and plumbed to a tee located at the core hole standpipe. Grout was injected through the run of the tee with water injected at the branch of the tee. Generally, grout and water were injected simultaneously into the core hole; the grout delivery valve was open, and water (supplied by the additional pumping system) was injected into the core hole. The "fill-rite" flowmeter was used to record total grout volume injected into the core holes. A pressure gauge located at the standpipe was used to record injection pressures of both water and grout. The Great Plains Industries flow totalizer recorded the volume of water added to the core hole.

3.5 April 2001 – W1 Drift Drilling and Grouting

A Gardner-Denver Model 83 Jackleg drill was used to drill the radial pattern of grout injection holes used to seal the fracture system during the April 2001 technology emplacement.

Approximately 400 ft of hole were drilled and 45 gallons of grout were injected into the Miller Mine fracture system at pressures between 60 and 300 pounds per square inch (psi). The objective was to determine if the small seeps that existed after the initial grout injection would be minimized and the flow from the Miller Mine adit effectively reduced further. The Jackleg drill was used because it would fit into the small W1 drift (4-ft by 6-ft drift). Mechanical packers were used instead of pneumatic inflatable packers because of the ease of use and cost effectiveness.

3.6 Geophysical Investigation Methods

The geophysical techniques used for this demonstration project included VLF terrain conductivity and field gradient magnetometry. The field gradient magnetometry was performed in the underground mine workings of the Miller Mine. The survey was performed in the upper and lower mine workings. The gradient magnetometry was used to distinguish anomalies due to mineralized zones, which the gradient magnetometry can detect, from anomalies due to water-filled fractures, which gradient magnetometry cannot detect. The VLF is sensitive to conductive areas at depth. These areas include anomalies due to mineralization and water-filled fractures. This combination of techniques provided a quick, comprehensive approach for delineating conductive zones.

Figure 3-1. The drilling/grouting subcontractor placing an inflatable packer system into the grout hole for dye and water injection testing.

Figure 3-2. Photograph of the grout pump, tanks, and flowmeter system for the Stage I grout injection in 1999.

4. Phase I - Field Emplacement Results

Field emplacement and associated fieldwork was performed by MSE and Bush Drilling Incorporated in two phases. Phase I fieldwork consisted of drilling, coring, logging, testing, and grouting 10 boreholes placed in a radial pattern at select locations in the lower working level. Support equipment included a generator, compressor, core and Jackleg drills, mucker, various pumps, tanks, lines, ancillary tools, equipment, and supplies.

The HA Combi grout, manufactured by De Neef Construction Chemicals, Inc., was used for the demonstration. The HA Combi grout is available in 5-gallon, closed-head pails sealed under dry nitrogen since the grout is a water-activated material. The HA Catalyst Accelerator was used at 2% of grout volume. The volume of catalyst can vary depending on the designed activation time allotted for the project. For this project, 2% catalyst was used.

Bush Drilling provided all personnel (driller and helper), equipment, and tools necessary to drill, test, recover rock core, and perform grouting operations. MSE site personnel included a field team leader/site safety officer, site security, and technical personnel.

4.1 Preliminary Design (1999)

In Phase I, Technology Deployment (1999), drilling, coring, water and dye testing, and grouting of the associated fracture systems in the lower working level of the Miller Mine was accomplished in three design segments and at three drill stations (Appendix A, Plate 1). Design Segment 1 included drilling, coring, water testing, and grouting M1 through M4 at the first drill station located between survey points L11 and L12 (Appendix A, Plate 1). Design Segment 2 included drilling, coring, and grouting M5, M6, and M7 from the first drill station. Design Segment 3 included drilling, coring, and grouting M8 and M9 from a second drill station located just east of survey point L11, and MJ10 was drilled at a third drill station located approximately 33 ft northeast of weir 1 in the W1 drift (Appendix A, Plate 2).

The respective core holes were drilled to intercept the fracture system associated with the W1 drift and the MMRF. The core holes were drilled perpendicular to the fracture system. Only three drill stations were used to minimize mobilization and movement of the drill rig. The fracture system intersected by the core holes was mapped in the W1 drift during the initial underground survey prior to drilling.

Typically, the work sequence for each of the nine core holes included drilling and recovering core, descriptive logging, performing water injection and dye tests in three core holes, and grouting. Core was not recovered from the last borehole (MJ10) since Jackleg drilling methods did not allow for the recovery of core. Approximately 624 ft of drilling was completed, which included 617 ft of coring. Core recovery ranged between 94% and 100% for each of the nine core holes (Table 4-1).

The primary goal of Design Segment 1 work was to shut off or significantly reduce fracture flow of AMD into the W1 drift and make work associated with Design Segment 2 unnecessary. However, after grouting all the grout holes at Design Segment 1, the results indicated that the flow in the W1 drift had only a 75% reduction when averaged over a10-month period and compared to the original flow volume. As a result, Design Segment 2 drilling and grouting was performed to further reduce flow from the W1 drift in an attempt to achieve the 95% reduction criteria. While performing the Design Segment 1 work, the local direction of fracture flow into the W1 drift was identified and mapped. During Design Segment 2, it was determined that the main inflow into the W1 drift occurred on the right-hand rib or eastern side of the drift, and therefore Design Segment 2 grout injection was performed to reduce the inflow on the eastern side of the W1 drift. To further reduce the inflow from the east side of the Miller Mine W1 drift, Design Segment 3 work was undertaken and entailed performing Jackleg drilling and grouting drill hole number MJ10.

4.1.1 Design Segment 1 – Grout Injection

Design Segment 1 work consisted of setting up the core drill at Drill Station No. 1 and drilling a radial pattern of four core holes (M1-M4) from four pivot points all located within 5 ft of each other. These initial core holes radiated from N53E to S81E and were inclined up from the horizontal (9 degrees to 13 degrees). Each borehole was capped with standpipes, cored, pressure tested with water, and grouted as described below. This primary phase of drilling and coring included a total depth of nearly 296 ft (Table 4-1 and Appendix C)

Design Segment 1 grouting initially reduced the average fracture flow discharge from 5.88 gpm to 1.24 gpm as measured at the W1 sample port (Appendix B). In addition, data from this work indicated that an increased flow occurred in the southeastern section of the area to be grouted (Appendix A, Plate 4). This became evident after M1, M2, and M3 were drilled and cored. Discharge progressively increased from zero in M1 (northernmost core hole) to approximately 4 gpm in M3 located south of M1.

4.1.2 Design Segment 2 - Grout Injection

Design Segment 2 grout injection work consisted of drilling three core holes (M5, M6, M7) from the first drill station. These core holes radiate from due east to N85E and were completed to reduce the flow beyond that observed from Design Segment 1. However, significant reductions of AMD discharge were not observed after completion of the work. Influx into the W1 drift varied between 1.53 gpm and 1.03 gpm (0.10 and 0.08 W1 vertical head measurement) directly following completion of grouting at M5, M6, and M7.

4.1.3 Design Segment 3 – Grout Injection

Design Segment 3 grout injection was performed to reduce the flow entering the right rib (east side) of the W1 drift. The work consisted of drilling two core holes from the second drill station (located just east of survey point L11; grout core holes M8 and M9) nearly parallel to the W1 drift but with different inclinations. Additionally, one short hole (MJ10), located approximately 33 ft northeast of the W1 flume, was drilled and grouted.

4.2 Rock-Core Drilling and Recovery

The Hagby core drill was used to recover rock core from nine core holes (M1, M2, M3, M4, M5, M6, M7, M8, and M9) located on the east rib of W2 between mine survey points L11 and L12 (Appendix A, Plate 1) within the lower workings of the Miller Mine. An additional borehole (MJ10) was drilled with the Jackleg drill in the east rib of the W1 tunnel approximately 33 ft northeast of the W1 weir.

In general, the rock core was described as a fractured and altered chloritized quartz diorite or granodiorite (Miller Mountain Intrusive rocks) with minor amounts of fractured silicified claystones and mudstones (Precambrian Belt rocks). Core recovery and details about each hole are shown in Table 4-1, and core logs are included in Appendix C.

Two sizes of core were recovered from each core hole (HQ-NQ3 or BQ-AW34); slightly larger core (HQ, BQ) was recovered the first 10 ft of each core hole (Table 4-1). The larger size core (HQ or BQ) was drilled and recovered in order to install a 10-ft standpipe with a threaded collar at the face. Core holes M1, M2, and M3 had HQ size standpipe installed [3.764-inch outside diameter (OD)]; M4 through M9 had BTW standpipe installed (2.346-inch OD).

All core was logged, photographed, and stored in water-resistant core boxes. Core logs included lithologic descriptions of the rock core and drawings of the respective fracture patterns

(Appendix C). Orientated core was recovered in the first three core holes (M1, M2, M3) (Table 4-2) using the clay-impression method. Clay impressions were taken at the base of the core hole after each core run, matched to recovered core, and used to determine the orientation of the core and the fractures that were to be grouted.

4.2.1 Downhole Camera Survey

A Sperry-Sun single-shot borehole deviation tool was used to confirm the bearing and inclination of M1, M2, and M3 completed during Design Segment 1. This tool used a downhole camera to provide a photograph at discrete depths of the magnetic orientation (compass bearing) and inclination of the borehole. Generally, data from this borehole deviation tool confirmed the initial bearing and inclination determined by the Brunton compass. This tool provided the compass bearing relative to magnetic north. A correction factor of 17 degrees was added to correct for the local magnetic declination of 17 degrees east of true north. Table 4-2 details the results of the downhole survey.

4.2.2 Core Hole Location Survey

A theodolite was used to determine the location of each initial drill point upon the rock face above the sill (northing, easting, and elevation). This information is shown in Table 4-3. Generally, the initial drill point for M1 through M6 was approximately 5 ft above the sill. The initial drill point for M7 was approximately 3 ft above the sill and over 6 ft for M8 and M9.

4.2.3 Geology – Recovered Rock Core

Approximately 617 ft of rock-core (sedimentary rocks of Precambrian age and Tertiary igneous rocks) were recovered, described, and logged during drilling and coring operations. The dominant rock type included a quartz or granodiorite likely emplaced during the Tertiary period by forceful injection into Precambrian limey shales and mudstones. Many of the rock fractures logged and described in the rock core are likely related to intrusive emplacement and associated faulting and deformation.

Generally, the granodiorite was slightly to extremely altered, weathered, and had a slight greenish cast due to chloritization. Previous work confirms that hydrothermal alteration of biotite and hornblende to chlorite is common (Ref. 3). Numerous, but generally small intervals of core, were extremely altered, soft, weak, and clayey. This is due to the complete alteration of orthoclase to kaolinite and plagioclase to sericite (Ref. 3). Many of these altered intervals included sheared fracture planes with slickenside textures that often occurred in the vicinity of the MMRF. Generally, these rocks are extremely fractured and cut by numerous quartz veins and veinlets that are commonly mineralized with Fe and Cu sulfides. Disseminated sulfide mineralization was common throughout the core.

Silicified and fractured shales, mudstones, and claystones were logged and described in the rock core. These rocks comprised approximately 52 ft of the total potential core of 617 ft and generally were creamy white to creamy pale brown, hard, and suffused by numerous quartz veins and veinlets to give a contorted banded appearance.

These rocks are mapped in Appendix A, Plate 3, which shows in plan view intrusive and silicified sedimentary rocks and associated faults and fractures. Appendix A, Plate 3 also shows, based on rock core data, the relationship between Belt and igneous intrusive rocks. Generally, Plate 3 supports the concept of pressurized magmatic injection along zones of weakness as well as local magmatic assimilation of Belt sedimentary rocks during periods of intrusion. Remnants of silicified Belt rocks are enclosed or otherwise surrounded by igneous intrusive rocks. Fracturing and faulting are concentrated in areas associated with the contact between the sedimentary and igneous rocks.

4.2.4 Structure

Detailed geologic and structural interpretation within the Miller Mine is based on the work of Mr. E. A. Johnson (Ref. 3) and recent work by MSE during coring and grouting operations within the mine. Recent structural data was gathered from the nine radiating core holes between mine survey points L11 and L12. In general, these core holes were drilled normal to the main structural trend associated with the MMRF having a trend approximately N30W with a dip approximately 80 degrees west. Numerous fractures and faults are associated with the contacts between the igneous and sedimentary rocks (Figure 2-1 and Appendix A, Plate 3).

4.2.5 Fracture/Shear System – Mine Map

Fractures in the granodiorite mapped in the W1 drift dip steeply toward the west and show two major strike patterns of N15W to N60W and N65W to N40E. Generally, fractures mapped in the W1 drift are located 2 to 7 ft apart and become more abundant and closely spaced toward the northeast (near the MMRF). Fracture origin is likely due to movement along faults and the development of stresses within the zone of the MMRF.

4.2.6 Fracture/Shear Systems – Rock Core

Plate 3 of Appendix A, which was developed from core data, shows two fracture systems associated with the MMRF. The first of these systems has a strike pattern of N10W to N45W while the second has a strike pattern of N30E to N45E. Core data indicated that both the granodiorite and silicified sedimentary Belt rocks are highly fractured and slightly to extremely altered by the actions of hot water

Plate 3 of Appendix A, shows several areas of undifferentiated Precambrian Belt rocks enclosed by Tqd and generally bordered on the east and west by fractures. It is believed that this fracture pattern may be one of the primary water conduits into the W1 drift from the MMRF. Several fractures shown on Plate 3 intersect the W1 drift and are associated with water-bearing intervals. The presence of this group of fractures could not be determined from visual observations or mapping conducted in the W1 drift.

Average fracture density described from core samples varies between two to nearly seven fractures per foot of core. This is much greater than the fracture density mapped in the W1 drift and is possibly due to the masking effects of the abundant coating of Fe oxides and other alteration products along the drift. Generally, fracture density associated with core samples increased with drill hole depth and in the direction of the MMRF (Figure 2-1 and Appendix A, Plate 3). These closely spaced fractures are generally steep, dipping between 60 degrees to near vertical, especially near the MMRF. Core sections near the MMRF are extremely soft, weak, altered, and clayey. By contrast, core sections taken at a distance from the MMRF, but associated with the secondary fractures, are generally fragmented with a striated texture (slickensides) and not altered to clay.

4.2.7 Observed Water-Bearing Fractures

Water-bearing intervals were recorded while drilling and coring in the lower workings of the Miller Mine. Details are included in Table 4-4 and Appendix A, Plate 4. Results of core drilling indicated that water-bearing fractures occur southeast of M1, and flow increased from virtually zero in M1 to over 4 gpm in M3 and M4 near the zone of the MMRF (Plate 4). Water flow was observed from all core holes except for M1 and M5 (Table 4-4, Table 4-5, and Appendix A, Plate 4). It is likely that a significant part of this water is associated with water-bearing fractures associated with the MMRF and possibly the contact between igneous and silicified Belt country rocks. Core hole data indicated that local water flow within the fractured rocks generally increases to the southeast.

Water-bearing intervals of fractured rock, recorded from field data, are presented in Table 4-4 and mapped in Appendix A, Plate 4. Generally, all recovered core was fractured and slightly to extremely altered. Water-bearing fracture systems generally were found to occur beyond 40 ft within core holes M2, M3, M4, M6, M7, M8, and M9 in the vicinity of the MMRF. Appendix A, Plate 4 shows water-bearing intervals, the mapped location of each core hole and potential pathways between core holes.

4.2.7.1 Description – Water-Bearing Fractures

Water-bearing fractures occurred in seven core holes (M2, M3, M4, M6, M7, M8, and M9) and are described below. Except in M4 and M9, water-bearing fractures were found to occur in the quartz-granodiorite. Many fractures associated with M4 and M9 are related to the diorite-Belt rock contact. Grout hole M5 was primarily a dry drill hole; however, fractures in M5 were connected to the fractures in M4 and M6. Cross-communication between M5, M6, and M4 occurred during grout injection.

4.2.7.2 M2 Water-Bearing Fractures (44 to 49 ft)

Fractures in this bore hole interval are the result of shear associated with faulting (slickenside texture-surface smooth, glassy finish with parallel striations) and are narrow, open, and slightly coated or stained with Fe oxide, chlorite, or clay (kaolinite). Much of the core from this interval can be described as a very altered diorite with a closely spaced fracture pattern that is highly broken and fragmented (Log Description of M2, Appendix C).

4.2.7.3 M3 Water-Bearing Fractures (53 to 63 ft and 68 to 73 ft)

Rock fractures in the 53- to 63-ft interval of this bore hole occur in chloritized diorite. Fractures are partly the result of shear (described above) and are narrow to moderately wide (0.05 to 0.5 inch in width), unstained to partially filled with Fe oxide, Mn oxide, clay, chlorite, and quartz. These fractures are open and permeable.

Rock core from the 68- to 73-ft interval is sheared with abundant slickensides, soft, weak, and extremely altered to clay. Fractures are often partially filled with chlorite, clay, and quartz.

4.2.7.4 M4 Water-Bearing Fractures (23 to 31 ft and 45.5 to 46.5 ft)

Rock fractures in the 23- to 31-ft interval of this bore hole include a sequence of fractured silicious Belt rocks (M4 Log, Appendix C). Water-producing fractures in this interval area are believed to be associated with the diorite-Belt rock contact. Typically, fractures near the contact are closely spaced and have produced fragmented rubble, especially between 24 and 25 ft. These fractures are narrow to moderately wide and stained and partially infilled with Fe oxides and clay. The entire interval has been shot through or suffused with quartz veins to 0.5 inch in thickness and mineralized by disseminated Cu and Fe sulfides.

Water-bearing fractures in the 45.5- to 46.5-ft interval of this bore hole are associated with fractured diorite, narrow with a striated texture (sheared – slickensides), and slightly coated with clay, chlorite, or manganese oxide.

4.2.7.5 M6 Water-Bearing Fractures (43 to 44 ft)

Fractures associated with this bore hole interval have fragmented the core and are stained with oxides of Fe.

4.2.7.6 M7 Water-Bearing Fractures (52 to 57 ft)

Fractures associated with this bore hole interval are similar to fractures described above in M6, except that these fractures are stained or partially filled with clay and manganese oxides. As described above, the core is chloritized, suffused with quartz veins to 2 inches in thickness, and mineralized with disseminated sulfides of Fe and Cu.

4.2.7.7 M8 Water-Bearing Fractures (48 to 53 ft)

Fractures associated with this bore hole interval are associated with shear, especially near 49 ft. These fractures are generally narrow and stained to partially infilled with clay and chlorite. This interval is chiefly chloritized diorite as described above, but includes a small interval, between 51 and 52 ft, of silicified Belt rocks.

4.2.7.8 M9 Water-Bearing Fractures (31 to 32 ft and 43 to 58 ft)

Fractures within the 31- to 32-ft interval of this bore hole are associated with the contact between altered diorite and silicified Belt rocks. This interval is rubblized with fractures that are narrow, open, and stained with oxides of Fe.

Fractures within the 43- to 58-ft bore hole interval are narrow to moderately wide and stained to partially filled with oxides of Fe, chlorite, and kaolinite. Much of the core within this interval is extremely altered, soft, and clayey ; water-bearing fractures between these extremely altered sections are rubblized and completely fragmented.

4.3 Water/Dye Injection Tests

Water/dye injection and testing of select intervals (stages) was done in several core holes before grout injection (M1, M2, M3, and M8) using packers inflated with nitrogen to pressures of approximately 150 psi to 200 psi. Water/dye injection tests were conducted to:

– determine the approximate hydraulic conductivity of the fractured rock;
– confirm that specific intervals and logged fracture patterns were interconnected between core holes;
– confirm that fractures were able to take water; and
– confirm that fractures had the potential to accept grout.

Water mixed with food-grade blue dye was injected into M3 and M4 as part of the water injection testing phase. Blue dye injection testing was used to:

– more easily see any cross communication or interconnection between core holes;
– determine where water entered into the W1 drift;
– determine the approximate travel times from injection points to discharge points; and
– determine the approximate volume of discharge from respective discharge points.

4.3.1 Lugeon Water Injection Testing

The Lugeon method was used to calculate the relative permeability of the fractured interval during water injection testing (Ref. 5). This method provides information on whether or not a fractured interval will easily take water based on the amount of water injected per unit length of the interval per unit time (liter/meter/minute). Generally, a value of 1 Lugeon is a low permeability area where grouting is not necessary; 10 Lugeons warrant grouting for most seepage reduction jobs. A Lugeon value of 100 indicates areas where fractures are common and open and grouting is necessary to control seepage.
The water testing Lugeon values and borehole permeabilities are presented in Appendix E. Lugeon values shown in Figures 4-1, 4-2, and 4-3 were calculated as a function of injection pressure: the Lugeon value is 2 (liters per meter per 5 minutes) when injection pressure was 1 bar (15 psi); when pressure was other than 1 bar, the value was liters/meter/minute)(10)/actual pressure in bars.

High Lugeon values indicate relatively high fracture permeability for a given interval. For example, Figure 4-1 shows a comparison of Lugeon values and the total number of fractures of several cored intervals in bore hole M1. The interval between 40 ft and 70 ft shows a relatively low Lugeon value (8) associated with a high fracture density (6 fractures per foot).

By contrast, the interval near the face of M1 (10 to 18 ft) has a high Lugeon value (greater than 100) associated with a low fracture density (2 fractures per foot). This data illustrates that fractures in the 40- to 70-ft interval are relatively tight and impermeable compared to fractures associated with the 10- to 18-ft interval. Overall, the Lugeon value of bore hole M1 decreases with hole depth as the fracture density increases.

The Lugeon value of bore hole M2 is relatively constant through the first 52 ft of core (61-66) and then drops significantly between 52 and 70 ft of bore hole to a value of approximately 12. The fracture density is similar throughout the bore hole with an increase from 4.2 to 5.5 in the 32- to 52-ft section. Figure 4-3 shows that the Lugeon value for bore hole M3 drops from greater than 100 to an average of approximately 50 after the first 20 ft of bore hole. The fracture density for this hole is nearly constant through the first 68 ft at a value near 4.5. The fracture density increases in the last section of the bore hole to a value of 6.8. Generally, Figures 4-1, 4-2, and 4-3 indicate a decrease of fracture permeability with depth (decreasing Lugeon values) associated with a slight increase of fracture density. This may indicate that an open and more permeable fracture pattern was developed near the rock face during blasting and mucking associated with the excavation of the mine working.

4.3.2 Lugeon Fracture Hydraulic Conductivity Results

Fracture hydraulic conductivity values were calculated based on the Lugeon values calculated from water-injection tests. Lugeon hydraulic conductivity values calculated for the fracture pattern associated with core holes M1, M2, M3, M4, and M8 ranged between 10^{-3} and 10^{-4} centimeters per second (cm/s). These values were calculated based on the approximate relationship of 1 Lugeon unit equals 1.3 x 10^{-5} cm/s (Ref. 5). Specific values are listed in Table 4-5.

Generally, these values indicate that the conductivity of the fracture system using the Lugeon method is similar to the conductivity of sandy silt. However, the hydraulic conductivity of the system is also a function of a fluid's density and viscosity. Lower hydraulic conductivities of the fracture pattern may be expressed as density and viscosity of a given fluid increase. This is the case since the grout selected for injection has a viscosity (500 to 700 centipoises (cps) at 77 °F), which is two orders of magnitude greater than the viscosity of the mine waters (1.4 cps at 41 °F) used during the injection tests. Due to the high viscosity of the grout, the hydraulic conductivity value for the fracture system is reduced, thus limiting the effective radius of the grouted area (Table 4-6). This is especially true since the great majority of fractures logged and described were narrow and tight (Appendix C), and very little grout was observed in fractures within the core from previously grouted holes as discussed below and shown in Appendix A, Plate 4.

Preliminary calculations indicate that the hydraulic conductivity of the fracture system was approximately two to three orders of magnitude lower when a viscous grout was substituted for water. For example, all hydraulic conductivity values noted in Table 4-5 are reduced to approximately 10^{-6} cm/s when calculating the hydraulic conductivity of the fracture system using a viscous grout, thus the radial extent of the grout emplacement is reduced.

4.4 Phase I - Grout Emplacement

Grout injected into each core hole (M1-M9) consisted of an HA Combi polyurethane grout combined with a HA Flex Cat Accelerator (fast/slow activator) that was designed to fill voids and fractures. This material activates and expands on contact when mixed with water. Field observations of this material indicated that this material had a common expansion ratio of at least 5 to 1 when thoroughly mixed with water.

This grout was chosen based on its performance during laboratory tests. Criteria critical to the use of this material was its ability to withstand and solidify under cool temperatures known to occur in the Miller Mine and when submerged in acidic waters. This grout passed all test requirements and was selected from among 50 other grouts subjected to identical tests.

The HA Combi Grout is a resin activated by a urethane activator (HA Flex Cat); lines, hoses, tanks, grout pumps, and sundry equipment were cleaned with a special washing agent after each grouting operation. This washing agent is a nonflammable solvent mixture necessary to clean the injection equipment and prevent plugging by residual activated grout.

All grout injected into each respective core hole was activated by the fast urethane activator except for the 32- to 70.5-ft interval in M2. Each batch of grout was mixed with a 2% portion of activator. For example, each 5-gallon batch of grout was mixed with 0.10 gallons of fast activator or 0.38 liters. Table 4-7 details the amount of grout injected into each core hole, amount of water injected with the grout, average injection pressure, time required to inject, and date of injection. A grout sample was recovered from each grout batch prior to injection. These samples, each approximately 3 ounces, were thoroughly mixed with a similar amount of water to determine reaction times. Generally, grout reaction and associated expansion began 2 to 3 minutes after being mixed with water between 65 °F and 70 °F. Observed grout expansion under these conditions ranged between 4 and 5 times the original volume. Table 4-6 gives general properties of the grout, and Table 4-7 reports grout injection data associated with each core hole.

This data includes the injection intervals, amount of grout and water injected, and average injection pressure. Using Table 4-7, the sequence of grout injection for each core hole between October 4 and October 20, 1999, can be followed.

4.4.1 Core Hole Fracture Cross Communication

Water testing and later grout injection confirmed that the fracture system, described and logged at each core hole, would accept grout, and preferential communication existed between core holes. The water and grout injected into each respective core hole followed a fracture path of least resistance, and many of the fractures described in the core were too narrow and tight to allow the viscous grout to fill and seal the fracture system. Fracture communication and fluid transport were observed during the water injection testing and the grout emplacement. The communication between fractures is greatly influenced by the viscosity of the fluid (water or grout) and perhaps by capillary forces associated with tight and narrow fractures. Cross communication between core holes was faster and more widespread as the viscosity of the fluid was reduced. For example, communication between core holes or discharge into the W1 drift was much less when a viscous grout was substituted for water during grout injection. Fracture communication and interconnection between core holes was inferred while drilling and coring, water testing, and during grouting. Appendix A, Plate 4 shows details of fracture communication recorded in the field.

Fractures observed in the core were logged and described in detail. With few exceptions, fractures were narrow (0.05- to 0.1-inches in width) to very narrow (less than 0.05 inch in width). Fractures occurred in all recovered core. Generally, these fractures were narrow, stained with Fe, and not in-filled. Maximum fracture density ranged from over four fractures per foot to nearly seven fractures per foot in the first three core holes (M1, M2, M3) (Figures 4-1, 4-2, and 4-3). These fracture densities are representative and similar to fractures logged in M4, M5, M6, M7, M8 and M9 (Appendix C).

4.4.1.1 Fracture Communication – While Drilling and Coring

Fracture communication between core holes was observed between M1 and M2 while drilling M2. In general, water flow (chiefly from drilling fluid) increased slightly in all core holes and from the W1 drift while drilling and coring M2-M7 (Tables 4-4 and 4-8). Appendix A, Plate 4 shows the direction, distance, and packer intervals where communication of grout or water occurred during application of the technology or while drilling grout holes.

Congealed grout was found as a fracture filling in five different, very small intervals among four closely spaced core holes (Appendix A, Plate 4). However, this grout was gelatinous and did not resemble or have the texture of grout samples left to congeal in a nonconfined environment. This indicates that much of the grout did not travel through the tight and narrow fracture pattern but took a preferred path. The single exception was the grout injected into MJ10 (drilled with a Jackleg drill). This grout sealed a nearby weep hole having an inflow of 1 to 2 gpm in the W1 drift immediately after injection and had the same texture as grout left to congeal in a nonconfined environment. Apparently, fractures associated with this weep hole were relatively large, open, and allowed for grout transport.

Grout, likely from M3, was observed in small intervals of core from M4 and M5 (Appendix A, Plate 4). Similarly, grout was described as a fracture filling in a small interval of core from M9 that likely originated from M8 or M6 (Plate 4). Likewise, two small intervals of grout were described as a fracture filling in M6 that likely originated from M5 (Plate 4).

An aqueous discharge of approximately 4 gpm was observed from grout hole M3 immediately after completion drilling. This observation correlated to a reduction of discharge from the W1 weir and indicates that there is a direct relationship between flow measured in the W1 weir and fractures intercepted by M3.

Similarly, an increase in the discharge to the W1 drift was observed immediately after M4 was cored. This was a direct result of M4 skimming and breaking through the back of the W1 drift roof and intercepting a water-producing fracture that previously had very little affect upon measurements of the W1 weir. This resulted in an increase in the discharge from the W1 weir as water (approximately 4 gpm) rained from the back where M4 had broken through the roof of the mine workings. This affect was transitory; discharge from the W1 weir was reduced after M4 was grouted.

4.4.1.2 Fracture Communication While Water Testing

Water testing was performed on M1, M2, M3, M4, and M8. This included injecting blue dye (food coloring) mixed with water into M4 and several intervals of M3 (Table 4-8). Fracture communication between core holes was observed between M2 and M3 during water injection testing of M3 (Table 4-8). Increased water flow was observed from M2 while testing the 40- to 70-ft M1 interval. Similarly, increased water flow was observed from M2 while testing several shallow intervals of M3, and blue dye from M1 and M2 was observed while testing deeper intervals of M3 (Table 4-8 and Appendix A, Plate 4). No increase in water flow was observed from M3 or M1 while testing M2. Also, because there was communication between M3 and M1 grout holes, it was apparent that there was a large flow path in the deeper intervals setting close to the MMRF.

4.4.1.3 Fracture Communication – While Grouting

Fracture communication between core holes while grouting was observed. An increase of water flow from M1 was evident while grouting M2 (M2 was grouted prior to M1). Similarly, blue water was observed coming from the W1 weep hole in the W1 drift while grouting M1. A flow of grout and water from previously grouted core holes M3, M4, and M5 was observed while grouting M9. The W1 rib leaked grout while grouting the interval of M8 from the face to 42 ft (Table 4-8). Finally, grout was observed to be leaking from the W1 weep hole after grouting MJ10.

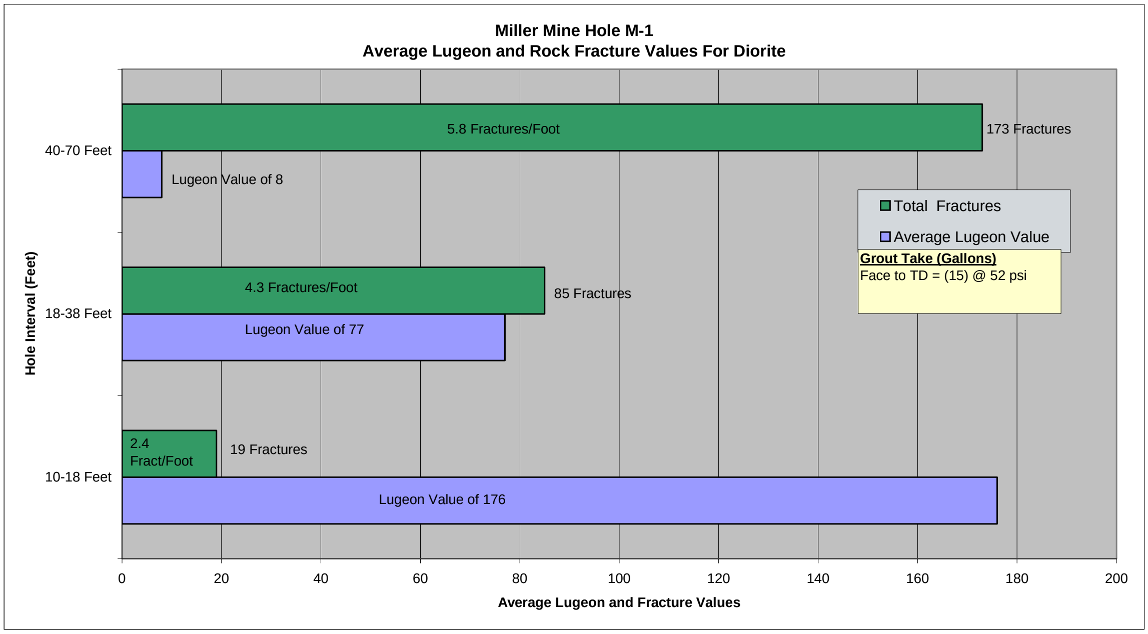


Figure 4-1. Plot of the average Lugeon and fracture density for grout hole M1.

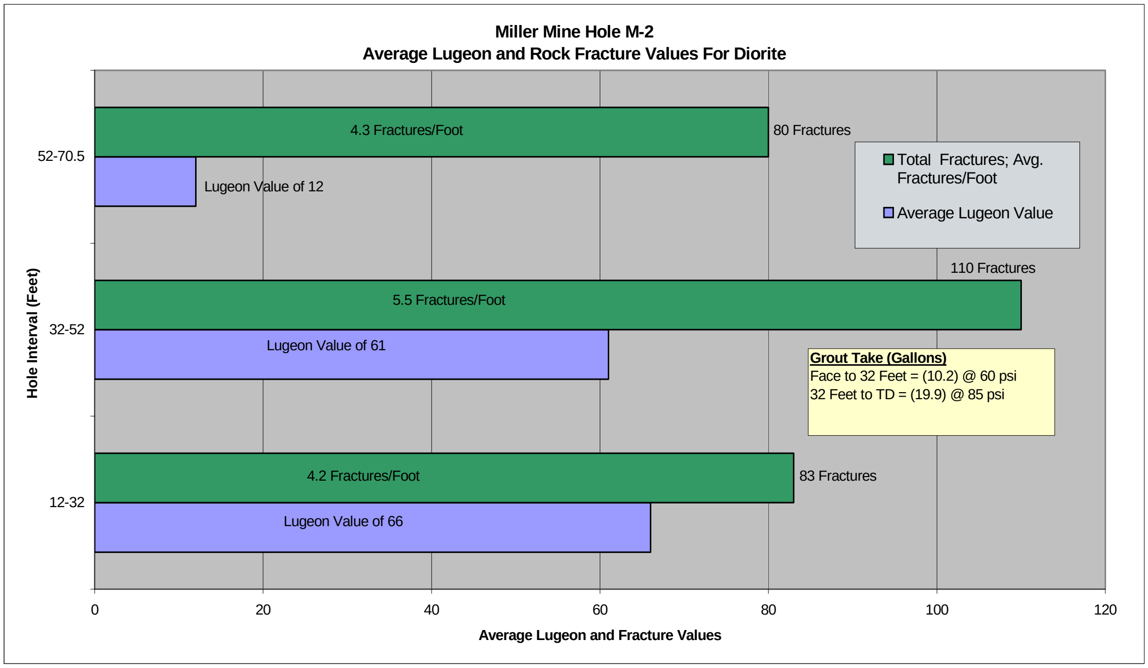


Figure 4-2. Plot of the average Lugeon and fracture density for grout hole M2.

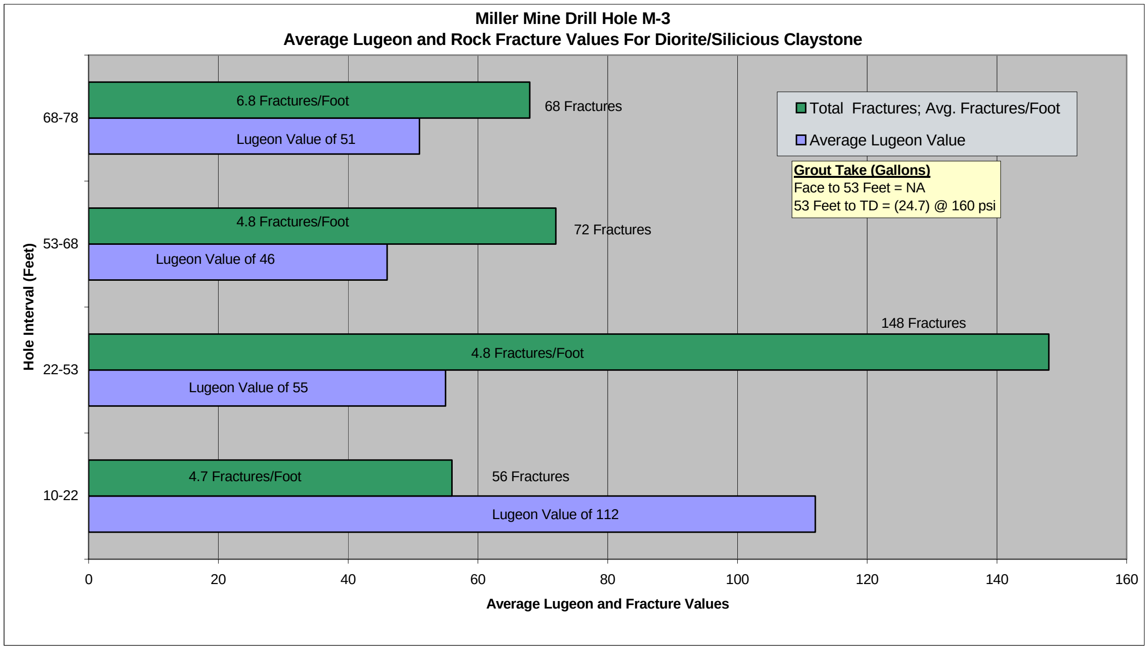


Figure 4-3. Plot of the average Lugeon and fracture density for grout hole M3.

Table 4-1. Rock Core and Drill Data

Core Hole (ft above sill)	Drill/ Method[1]	Core Size[2]		TD[3] (ft)	Core Recovery		Mean RQD[4]	Bearing	Inclination Degrees from Horizontal[5]	Core Orientated[6]
		0-10	10-TD		Total Core (ft)	%				
M1	Hagby	HQ	NQ3	70	66	94	28	N53E	+13	Yes
M2	Hagby	HQ	NQ3	70.5	66	94	34	N70E	+8	Yes
M3	Hagby	HQ	NQ3	78	76	97	27	S85E	+10	Yes
M4	Hagby	BQ	AW34	77.2	77.2	100	37	S81E	+9	No
M5	Hagby	BQ	AW34	76.3	76	99	34	S89E	+10	No
M6	Hagby	BQ	AW34	72	71	99	37	N85E	+10	No
M7	Hagby	BQ	AW34	57	57	100	36	N90E	-8.5	No
M8	Hagby	BQ	AW34	58	57	99	30	N70E	+1	No
M9	Hagby	BQ	AW34	58	57	99	18	N67E	+6.5	No
MJ10	Jackleg	NA	NA	7	NA	NA	NA	S88E	+2	NA

[1] Hagby 1000 core drill; Gardner-Denver, Model 83 Jackleg pneumatic-percussive drill
[2] HQ core diameter 2.500 inches, hole diameter 4.625 inches; BQ core diameter 1.432 inches, hole diameter 2.965 inches; NQ3 core diameter 1.875 inches, hole diameter 2.980 inches; AW34 core diameter 1.062 inches, hole diameter 1.890 inches
[3] Cored TD (total depth) except for MJ10
[4] RQD (rock quality designation) is a modified core recovery percentage in which all pieces of sound core over 4 inches in length are summed and divided by the length of the core run (Appendix C)
[5] Inclination and bearing measured with Brunton Compass
[6] Orientated core was recovered using the clay-impression method (Appendix C)

Table 4-2. Comparison of Downhole Camera Survey Data

Hole Number	Brunton Survey at Face		Camera Survey		
	Bearing	Inclination (Degrees)	Bearing	Inclination (Degrees)	Depth-Ft (TD)
M1	N53E	+13	N58E	+13	70
M2	N70E	+8	N71E	+7	70.5
M3	S85E	+10	S85E	+10	78

Table 4-3. Survey – Drill Point Location

Drill Point	Northing	Easting	Elevation	Ft Above Sill
M1	11547.8	11244.9	6306.4	5.3
M2	11547.0	11245.7	6305.8	4.7
M3	11544.6	11247.7	6306.0	4.9
M4	11543.8	11248.1	6305.7	4.6
M5	11545.2	11247.4	6306.1	5.0
M6	11545.8	11247.1	6306.1	5.0
M7	11545.3	11247.9	6304.5	3.4
M8	11530.6	11257.4	6307.2	6.1
M9	11531.5	11256.9	6307.9	6.8

Table 4-4. Water-Bearing Fractures (Intervals)/Communication

Core Hole No.	Water-Bearing Intervals	GPM	Core Hole Communication During Drilling
M1	NA	NA	M1 and M2 (M2 drill water from M1)
M2	43.8-48.7	0.1	Water from M1 while drilling M2 at 29 ft
M3	53-58	2.4 to 4.1 total	Drilling fluid in the W1 drift
	58-63	0.2	Drilling fluid in the W1 drift
	68-73	1.5	
M4	23-31	4	Water flowing from back after drill skimmed and broke into the W1 drift
	45.5-46.5	1	
M5	NA	NA	At 55.9-56.1, observed grout from M3; blue dye seen at 37.8 in M5, likely from M4
M6	43-44	Approximately 2	At 37.1 and 43.5-43.8, observed grout likely from M4
M7	52-57	1.5	
M8	48-53	2-3	
M9	43-58	1.1	At 42.4, observed grout from M8/M6
	31.4-32.4		
M4	23-31	3-4 gpm	At 57.6, observed grout from M3; water flow increased through back (M4 core hole skimmed W1 back) while drilling M5
	46		

Table 4-5. Average Fracture Hydraulic Conductivity Values (cm/s) from Water Tests (K); Based on Lugeon Values

Core Hole M1		Core Hole M2		Core Hole M3		Core Hole M4		Core Hole M8	
Interval (Ft)	**K**	**Interval (Ft)**	**K**	**Interval (Ft)**	**K**	**Interval (Ft)**	**K**	**Interval (Ft)**	**K**
10-18	2.29×10^{-3}	12-32	8.58×10^{-4}	10-22	1.45×10^{-3}	32-77.2	1.69×10^{-4}	42-58	1.82×10^{-4}
18-38	1.00×10^{-3}	32-52	7.96×10^{-4}	22-53	7.09×10^{-4}				
40-70	1.04×10^{-4}	52-70.5	1.50×10^{-4}	53-68	5.98×10^{-4}				
				68-78	6.63×10^{-4}				

Table 4-6. Grout Properties

Grout	Viscosity	Color	Density	Toxicity	Tensile Strength
Uncured	500-700 cps at 77 °F	Opaque-amber	8.75-9.17 lbs/gal	Non Toxic	NA
Cured	NA	Pale brown	8.75-9.17 lbs/gal	Non Toxic	89 psi

Table 4-7. Grout Injection Data

Core Hole No.	Grout Injection Intervals (Ft)	Amount of Material Injected (gal)		Average Injection Pressure (psi)	Time Required For Injection (minutes)	Date of Injection
		Grout	Water			
M1	Face – 70	15	9.5	52	8	10/5/99
M2	32 - 70.5	19.9	5	85	7	10/4/99
	Face – 32	10.2	7	60	6	10/5/99
M3	53 – 78	24.7	5	160	12	10/5/99
	[1]Face – 53	NA	NA	NA	NA	
M4	32-77.2	18	15	100	18	10/8/99
M5	32-76.3	15	15	80	14	10/8/99
M6	Face – 72	20	23	60	12	10/13/99
M7	Face – 57	18	20	60	10	10/13/99
M8	42 – 58	13.5	14.8	80	11	10/18/99
	Face – 42	2	1.7	NA	1	10/20/99
M9	Face – 58	17.9	16.6	80	7	10/20/99
Total		174.2	131.6			

[1] M3 interval face - 53 not grouted; grout came around packer from interval 53-78 and came to face

Table 4-8. Water Test/Grout Communication Between Core Holes

Core Hole No.	Water Injection Intervals/Stages (ft)	Flow of Injected Water Observed From	Grout Injection Intervals/Stages	Grout/Water Observed From
M3	10-22	M2		
	22-53	M2		
	53-68	M3 flow ceased once packer set at 53	53-face	
	68-78 blue dye		53-78	
	53-78 blue dye			
	30-78	M1 and M2		
M2	12-32			
	32-52		Face-32	
	52-70.5		32-70.5	Water from M1
M1	10-18		Face-70	Blue water – W1 weep hole
	18-38			
	42-70	M2		
M4	32-77.2		Dye not observed in M1, M2, and M3; dye observed in the W1 drift; flow from W1 back increased	32-77
M5			32-76.3	Grout leaking from back & rib of the W1 drift; used polyfill
M6			Face-72	
M7			Face-57	
M8	42-58	W1 weep hole flow increased	42-58	Increase in the W1 weep hole flow
			Face-42	W1 rib leaked grout; plugged leaks with polyfill
M9			Face-58	Grout/water from M5, M4, M3
MJ10			0-7	Plugged W1 weep hole

5. Phase II – Field Emplacement

As a result of the 1999 grout injection, a majority of the flow into the W1 drift at the Miller Mine was eliminated. However, the grout injection had only eliminated flow that entered the mine from either above or below the mine workings. Due to the small size of the W1 drift (4 ft by 6 ft), it was impossible to place the Hagby core drill in the drift. Flow was visibly entering the mine from the northeastern rib of the underground mine workings as drips and weeps. A decision was made that additional grout injection work would be performed in an effort to eliminate flow into the W1 drift. The work was scheduled for April 2001, and a Jackleg drill was used to access the W1 drift.

5.1 Preliminary Design

A Gardner-Denver Model 83 Jackleg drill was used to drill the grout holes in the W1 drift (Figure 5-1). Geological information gathered during the October 1999 grout injection was used to map and design the second field implementation of grout. Five areas within the W1 drift were designated for grout injection; the areas were denoted as Zone A, B, C, D, and L13 and were located 18, 12, 6.9, 5 and 0 inches from survey point L13, respectively. The grout holes were drilled upward in a radial pattern at a shallow angle to the back of the workings (between 10 and 16 degrees) at each zone location. Holes were drilled and grouted on an alternating pattern to reduce communication between grout holes.

Initially, all grout holes were drilled to a depth of 8 ft and had a 1.75-inch ID. Some holes were extended in the field, but this decision depended on the fracture density, flow, and communication between the core hole and the W1 workings. Once a grout hole was drilled, the flow from each hole was denoted; mechanical packers were then placed in each hole to reduce the amount of water entering the mine workings. Zone A was the first scheduled location drilled and grouted, followed by Zone B (Appendix A, Plate 5).

The same grout materials (HA Combi grout and the HA Flex Cat Accelerator) were used for the April 2001 grout injection phase (Ref. 1). The amount of catalyst was varied depending on the number of holes to be grouted per grout run, water temperature, and amount of water flowing from each grout hole. The amount of grout varied for each grout run. Consideration was taken into account for the grout hoses, grout wasted between setups, and length of the hole (1 gallon grout per an 8-ft grout hole).

The activated grout containers and the grout injection equipment were transported to the grout stations using a small Kubota tractor (Figure 5-2). Once the grout equipment was set up, the grout was poured into a 30-gallon grout tank and activated using the catalyst. Grout injection began using a Moyno screw pump powered by a generator. Grouting pressures ranged from 60 to 300 psi, depending on the tightness of the fracture.

5.1.1 Geology and Drift Segments (A, B, C, D, and L13)

Due to the limiting size of the W1 drift workings (4 ft by 6 ft), a Jackleg drill was used to drill the grout holes. The cuttings were not collected since the workings were so confining, allowing only one individual to work in the W1 drift. As a result, the geology of each hole was not defined.

The geology mapped during the 1999 grout emplacement showed that the majority of the water entering the mine was either associated with the contact zones between the Tqd and the silicified limey shales of the Precambrian Newland Formation or with the fracture/fault zones associated with the MMRF. In the W1 drift, there were select areas that produced water. These areas were mapped before and after the first grout emplacement, and five distinct zones were selected for further grout emplacement using Jackleg drilled short holes. The five zones were as follows.

Zone A – At zone A, water influx into the underground mine workings was at the contact between the quartz diorite and the silicified limey shales. At this zone, water drips from the roof and the south mine wall. Fractures were at a lower angle (61 to 53 degrees from vertical). Lower angle fractures were not very tight (compressed) and usually produced or allowed increased water movement compared to steeper angle fractures and faults.

Zone B – Zone B was located at the contact between the quartz diorite and the silicified limey shale; however, Zone B was much tighter and the fractures are near vertical to the mine workings. This zone had minimal influx, and the influx was from drips in the wall that dampened the wall surface.

Zones C, D, and L13 – Zones C, D, and L13 were all associated and located in the MMRF zone (Appendix A, Plate 5). In the area of survey point L13, the mine workings followed the strike of the fault and the zone of numerous (less than 1 inch) quartz stringers; however, the quartz stringers were discontinuous and low grade, which led to the abandonment of the W1 drift. The rock in this area was very broken, and the northeastern and eastern rib of the W1 drift was inaccessible to the 1999 grout emplacement due to the size of the drilling equipment used during that emplacement. However, this area was grouted during the 2001 grout emplacement.

5.2 Summary of April 2001 Grout Emplacement

Jackleg grout holes (1.75-inch ID) were drilled in a radial pattern to intercept either the petrologic contact or the fracture system associated with the MMRF (Appendix A, Plate 5). Drilling and grouting was initiated at Zone A and progressed to the zones further in the W1 working.

The planned drill pattern was produced by drilling every other hole first and then completing the planned pattern by infill drilling between the initial holes. The flow of water from each hole was measured directly after each hole was drilled. Mechanical packers were then placed in each hole to reduce the amount of water entering the working area (Figure 5.1). The average volume of grout, estimated on a per foot of hole basis, was 0.23 gallons of grout per foot of hole grouted. The HA Combi grout had an assumed expansion factor of approximately 5 to 7 times its original nonactivated volume. However, the expansion factor of the grout is inversely proportional to the confining pressure (e.g., bearing pressure in fractures) in the system where it is being injected.

During injection, water was added so that the grout would activate. This procedure was used more in holes that did not produce a significant volume of water after they were drilled than in holes that communicated with the W1 drift (Table 5-1).

Table 5-1 provides the date holes were drilled and grouted, hole designation, total depth and bearing, grout volumes, injection pressures, water volumes, and comments. For the design of the 2001 field injection, there were several water-bearing zones targeted for grout injection. The zones were designated Zones A-D and Zone L13, located close to the survey point L13. The zones were drilled and then grouted (Figure 5-2), and the characteristics of each zone are as follows.

Zone A – Approximately 25 gallons of water-activated, polyurethane grout was injected into Zone A at pressures ranging between 200 to 330 psi. Grout holes on the west side of the workings were tight and fairly dry (slow drips) (Appendix A, Plate 5). However, the east side of the workings was very fractured and soft, allowing an increased influx of water into the mine workings. Reducing the inflow of water to this zone was difficult. Grout holes were drilled having an eastern orientation, as well as a western direction. Results show that water moved from the grouted area to nongrouted areas.

Zone B – This zone was tight, and the fractures were high angle. All grout holes drilled at Zone B were dry. Only three holes were drilled and grouted in this area. The total grout take for the three holes was 2 gallons.

Zone C – All grout holes drilled in Zone C had water flowing from them. C5ALT and C7 did intercept a high flow zone in the fracture/fault system. These holes were placed at low angles to the walls of the workings and drilled to intercept the MMRF.

Zone D – Zone D was drilled from underground survey point L13. D4, located in the east side (rib) of the mine workings, intercepted a high flow zone and the MMRF. Flow was greater than 3 gpm. The maximum grout injection pressures for D4 was approximately 50 psi. The grout holes drilled in the roof and on the west side of the workings were dry. All holes were grouted for closure.

Zone L13 – At underground survey point L13, water dripped from the roof and east side of the mine workings. Drill holes L13A and JK10R, drilled above the collar of hole number MJ10, provided the most inflow (1.5 gpm) into the workings of this group of holes The fault/fracture system associated with the MMRF was contacted by Jackleg drill holes at locations Zones D and L13. Using the underground mine map (Figure 2-1) and the geological maps created from the core drilling that occurred during the 1999 grout emplacement, it was apparent that these holes would contact the main water-bearing zone that contributed to the generation of AMD in drift W1 (Figure 5-3).

5.2.1 Grout Emplacement

Upon finalization of the April 2001 grout emplacement, approximately 400 ft of hole had been drilled and grouted. The total volume of HA Combi grout injected was 69 gallons, and of that, approximately 24 gallons were wasted. Wasted grout included the gallons of grout used to fill the grout injection lines and to recirculate the grout through the grout tank system. Some grout was wasted/lost when switching connections between the mechanical packers.

On average, approximately 0.17 gallon of grout was injected per foot of hole. However, grout holes with increased permeability took more grout on a per foot basis. The locations that took increased amounts of grout were located on the east and northeast side of the workings.

Figure 5-1. Stabilizing the underground mine working of the Miller Mine with roof bolts and screen using a Gardner-Denver Model 83 Jackleg drill.

Figure 5-2. Kabota tractor that transported the grout and grout equipment into the abandoned underground workings of the Miller Mine.

Figure 5-3. Iron oxide stalactites in the W1 drift, which were areas where flow through the fractures required grouting.

Table 5-1. Summary of the Grout Injection Sequence at the Miller Mine, W-1 Drift

Date	Grout Hole	Total Depth (ft)	Compass Bearing Drilled	Grout Injection Volume per Hole (gallons)	Maximum Pressure during Grouting (psi)	Volume Water Flowing from the Open Hole	Comments
4/16/01	A1	10.5	N54E	1	300	Drip	Hit fracture at 5 ft from collar
4/16/01	A2	12	N54E	2		Dry	Shallow angle drilled
4/16/01	A3	8	N67E	3		Dry	Did not hit fractures
4/18/01	A4	10	N69E	1.2	280	1 gpm	Steady flow, fracture at 6-7 ft from collar
4/19/01	A4ALT	10	N72E	1	200	Drip	Located between A4 and A5
4/16/01	A5	8	N80E	4		Steady drip	Hit fracture at 5 ft from collar
4/16/01	A6	6	N80E	1		Small drip	Hit fracture at 6 ft from collar
4/20/01	A11 (AB1L)	4	S60W	1	200	Steady flow	Water pushed packer out of wall – rough ground
4/20/01	A12 (AB2V)	5	S62W	2.5	200	Drip	Drilled into right rib of drift
4/20/01	A13 (AB3)	5	S50W	1.5	200	Steady drip	Drilled into right rib of drift
4/24/01	A14 (ARRV)	6	S70W	1.75	300	Drip	Drilled toward mine entrance
4/20/01	A15 (A3ALT)	10	S70W	1	200	Small steady flow	Located between A3 and A4
4/20/01	A16 (A2ALT)	10	S80W	2.5	200	Drip	Located between A2 and A3
4/24/01	AC (ARRC)	6	S70W	0.25	330	No water	Drilled toward mine entrance
4/25/01	ARRM	6	S27W	0.75	220	Drip	Drilled toward mine entrance
4/25/01	ARRM2	4	S27W	0.50	260	Dry	Drilled toward mine entrance
4/17/01	B2	17	N30E	2	300	Dry	Dry, grouted to fill hole
4/19/01	B3	8	N45E			Dry	Closed mechanical packer
4/17/01	B4	15	N80E			Dry	Closed mechanical packer
4/17/01	C1	8	N30E	1		Dry	No fractures hit during drilling
——	C2	——	N30E	——	——	——	Not drilled, dry area
4/17/01	C3	17	N42E	2	300	Drips	Packer has bad valve, in roof
——	C4ALT						Drilled 3 ft over weep
4/20/01	C4	19	N59E	2	140	Small drip	
4/17/01	C5	15	N65E	2	Out hole	Drips	Block from roof dropped, 2 ft of hole damaged
4/17/01	C5ALT	15	N65E	2	300	Steady flow	Drilled 3.5 ft back from C5
4/20/01	C6	17	N70E	2	150	Drip	Packer fell out of hole
4/17/01	C7	19	N75E	2.5	280	High flow	Low angle hole, Miller Mine Fault
4/19/01	C7ALT	19	N75E	2.5	200		
4/19/01	C7ALT RR (C7B)	21	N78E	3	200		Hit fracture at 15-19 ft; hard drilling past 19 ft

Date	Grout Hole	Total Depth (ft)	Compass Bearing Drilled	Grout Injection Volume per Hole (gallons)	Maximum Pressure during Grouting (psi)	Volume Water Flowing from the Open Hole	Comments
4/19/01	D1 (DMB)	12	N11W	2	300	Dry	
4/20/01	D2	10	N2E	2	300	Dry	Roof hole, left rib
4/20/01	D3	12	N3E	2	100 – 60	Dry	Roof hole, center
4/20/01	D4	12	N5E	4	50	3 gpm	Hit high flow zone, fault zone
4/20/01	D5	12	N5E	1.5	300	Drip	Plugged drill, grout at 6 ft
4/17/01	1L13H (U)	15	N85E	1	300	A few drops	Above MJ10, fracture at 3 ft from collar
4/17/01	1L13RR	8	—	Not grouted	Not grouted	Dry	Roof, +10 degree hole
4/24/01	2L13H	15	N11E	2	200	Drops	Fracture at 5 ft from collar
4/17/01	2L13RR	8	N65E	1	300	Drops	Fracture at 5 ft from collar
——	2L13R	——		——	——	——	Not drilled
4/25/01	L13A	15	N30E	1.5	250	Steady flow	Lost bit and striker
4/25/01	L13B	15	N11E	0.50	300	Steady flow	Lost half gallon at packer
4/24/01	JK10R	6	N2E	2	220	Water	Drilled into fault zone
4/24/01	CLR	3	S50W	0.50	300	Some water	Drilled over weep

6. Long-Term Monitoring and Mine Maintenance Results

6.1 Mine Maintenance

The Miller Mine project started in 1998 and was extended through 2003. During this period, the mine was opened with a 3-year project access requirement. As the project progressed, there were two technology applications rather than the originally planned single technology application, and the long-term monitoring was extended through 2003. As a result, the mine was restabilized several times; originally it was reopened and stabilized with the thought that the mine only needed to remain open for 3 years.

During each technology application, the mine required stabilization, and in April 2001, there was significant stabilization of the underground mine workings using screens and roof bolts (Figure 5-2). However, during the July 2001 sampling event, it was discovered that some of the timbers in the lower portal of the Miller Mine had collapsed and an area inside the mine had experienced some movement either due to freeze/thaw events, increased water from spring runoff and storm events, or localized seismic movement. As a result of this finding, access to the Miller Mine underground workings was determined to be unsafe, and further sampling was performed only at the Miller Mine portal (W4) and not in the underground workings at W1, W2, and W3 sample ports (Figure 2-1 and Appendix A Plate 1).

6.2 Monitoring History and Methods

For this demonstration, flow from the W1 drift was monitored at the W1 sampling port by a small, 60-degree trapezoidal flume located at the front of the W1 drift (Figure 1-5 and Figure 2-1). Flow from the winze and W3 drift was monitored at the W2 and W3 sample locations using 22.5-degree V-notch weirs. The total flow from the Miller Mine was monitored at the portal initially using a 22.5-degree, V-notch weir. In addition, a volumetric bucket test was performed at the 10-inch effluent discharge pipe. Because of mine instability, measurements at W1, W2, and W3 were discontinued. The last sample taken inside the mine was on November 8, 2001. Monitoring was continued at the Miller Mine portal, sample port W4, until November 2003. The flow and water quality results are presented in Appendix B.

6.3 Long-Term Monitoring Results

The primary criteria for the success of the project was to reduce the cumulative volumetric flow at the W1 sample location by 95% using flow data from 10 months before and after grout was injected into the Miller Mine fracture system (Ref. 2).

The first phase of grouting occurred September 24, 1999, to October 23, 1999, and the second phase of grouting occurred April 4, 2001, to April 16, 2001. Figures 6-1 and 6-2 show the flows with the approximate dates of grout completion.

The average flow at W1 prior to the first grout emplacement was 5.88 gpm, and the flow after both grout emplacements averaged 1.24 gpm, an approximately 80% reduction.

For statistical evaluation of the flow from the W1 drift, plotting the flow measurements [gallons per day (gpd)] versus time (days) and the area under the curve prior to grouting and after grouting were compared and used to calculate the percent reduction in flow (gallons) (Appendix F). During the comparison, corresponding seasonal time periods were evaluated to minimize the effect of seasonal fluctuations in the flow comparison. Table 6-2 provides the flow measurements taken at the W1 drift over the duration of the project. The flow was categorized into flow prior to, after Phase I, and after Phase II grout injection. Note the number of days elapsed for each time period is listed in Table 6-2.

Data from Table 6-2 were entered into MATLAB software as separate data strings; the area under each curve was then calculated to determine the total volume of water that flowed through the W1 flume during each time period (Appendix F).

Volumes between the pregrouting phase and Phase I grout injection were compared to determine the percent reduction in flow volume due to the Phase I grouting. Flow values between Phase I and Phase II were also compared to determine if the additional grouting reduced flow. Table 6-3 provides the results from the MATLAB calculations. Because the timeframes from the grout phases did not match exactly, date values after the Phase I grouting were matched with the compared grout phase, and flow values were then interpolated.

Table 6-3 reveals that the Phase I grouting reduced flow volumes during the December 1999 to September 2000 time period by 77%. The additional grouting of Phase II did not reduce flows any further. The flow volume during the July to November 2001 time period increased by 21% when compared to the July 2000 to November 2000 time period. However, flow calculations did not consider the effect of precipitation. During the specified time period, the total precipitation between 2000 and 2001 increased by 34% (Ref. 6). This increase in precipitation was not taken into account in the flow calculations at the Miller Mine.

Considering that the trapezoidal flume at W1 had an accuracy of +/- 5%, the flow reduction ranges were calculated by using the +5% maximum and -5% minimum flow values for each phase. Table 6-4 shows that the flow from the pregrout phase to after Phase I grouting was reduced by 75.1% to 79.6%. The flow difference between the Phase I grouting and the Phase II grouting was increased by 9.8% to 34.1%.

Long-term flow and monitoring results indicate that the grouting program has reduced the flow from the Miller Mine adit (W4), as well as the concentrations of dissolved metals. Flow and analyses of dissolved metals were recorded for 4 years after completion of the primary grouting program in 1999.

The influent flow (cumulative monthly flows at W1 and W2) and the effluent flow at the Miller Mine adit (W4) were compared and evaluated (Figure 6-1). Results indicate that the percent reduction in flow at the Miller Mine adit (W4) ranged between approximately 50% and 84% depending on the season (Figure 6-1).

Prior to the application of the source control technology, flow results and visual observation revealed that most of the flow into the workings occurred in the W1 drift (Figure 6-2). After grouting, flow in W1 was reduced; however, flow recorded from the W2 and W3 drifts indicate that those workings were not affected by the grout emplacement even with the close proximity of the workings.

The last flow measurements taken inside the Miller Mine workings were on November 8, 2001. On that date, 53% of the flow was from the W1 drift and 47% was from the W2 drift. As the flow at W4 increased from 1.9 gpm to approximately 3.0 gpm in September 2003, it is unknown where the increase in flow originated because access to the underground mine workings was restricted. However, since there was an increase in precipitation of approximately 34% from the prior year, then part of the increased flow could be a result of increased precipitation. The effect of the increased precipitation on the in-mine flow regime is not known.

6.4 Water Quality Results at the Miller Mine

The secondary criteria for success at the Miller Mine included improving the quality of the water discharging the adit; decreasing the dissolved metals concentrations, and increasing the pH of the water discharging from the W1 drift and the mine portal at sample port W4. The analyzed dissolved metals were Al, arsenic (As), cadmium (Cd), Cu, Fe, Pb, Mn, Ni, Zn, and silver (Ag); sulfate concentrations were also analyzed. Because of the wide range in concentration values, the dissolved metal concentrations and metals loading graphs were plotted using a logarithmic scale (Figure 6-3). The dissolved metal concentrations for As and Ag were at or below the instrument detection limit (IDL) and will not be discussed in detail.

6.4.1 Water Quality and Metals Loading at W1

During evaluation of the dissolved metal concentrations from July 1998 to October 2001, it was determined that application of the technology did not significantly affect the water quality in the W1 drift. It was thought that by grouting the exposed fracture surfaces (much of which is sulfide material) and eliminating water or oxygen (required for the formation of AMD) that the dissolved metals concentrations in the W1 drift would be reduced. Water quality results from W1 indicate that there was a slight decrease in pH from 6.08 to 5.95 and a minimal increase in dissolved metals concentration for Al, Cd, Cu, Fe, and Pb (Table 6-5). Conversely, a decrease in the concentrations of Zn, Mn, Ni, and sulfate were recognized (Figure 6-3 and Table 6-5).

However, due to the reduction in flow in the W1 drift, the metal-loading rates (both dissolved and total) were reduced significantly, even though the water quality (as concentration) did not fluctuate (Figures 6-4 and 6-5). Average dissolved metal load reductions of greater than 80% were calculated for Cd, Al, Zn, and Fe. Reductions of greater than 50% were recognized for the Mn, Pb, Ni, and Cu dissolved loads (Figure 6-4). Specific metals-loading reductions calculated for the W1 drift (Figure 6-5) indicate:

- Fe load was reduced from 7.50 pounds per day (lb/d) to 0.12 lb/d;
- Mn from 0.45 lbs/d to 0.09 lb/d; and
- Zn from 0.08 lbs/d to 0.004 lb/d.

It should be noted that the data used in the calculation of load reduction were taken from the same season (fall–early winter) as the flow of water and the concentration of metals in that water (both load calculation variables) are known to be strongly effected by seasonal variations.

6.4.2 W4 Miller Mine Portal Water Quality and Metals Loading

The water quality of the water discharging at the Miller Mine portal (W4) improved as a result of the technology applications. The water quality improvement is directly related to the reduction of influx into the W1 drift. The metal concentrations from W1 are being diluted as the water from W1 merges with water from the W2 drift. The amount of dissolved metal in the portal discharge significantly dropped after each application of the technology (Figure 6-6). Iron and Zn concentrations were reduced the most when comparing 1999 pregrout and 2003 postgrout analytical results. The dissolved concentrations for Pb did increase; however, due to the decrease in flow, Pb loads were reduced by 50% on average (Figure 6-7).

As calculated, the average percent reduction of dissolved metal loads was greater than 50% for all metals analyzed. Both Zn and Fe loading was reduced as much as 90%, and the Phase II grouting assisted with further reduction in the metals loading (Figure 6-8). The most significant reduction in metals loading at the Miller Mine portal, located 600 ft from the W1 drift, include:

- Fe from 3.05 lb/d to 0.02 lb/d;
- Mn from 0.46 lb/d to 0.09 lb/d; and
- Zn from 0.06 lb/d to 0.004 lb/d.

Most of the analyzed dissolved metals load was reduced by an order of magnitude or more.

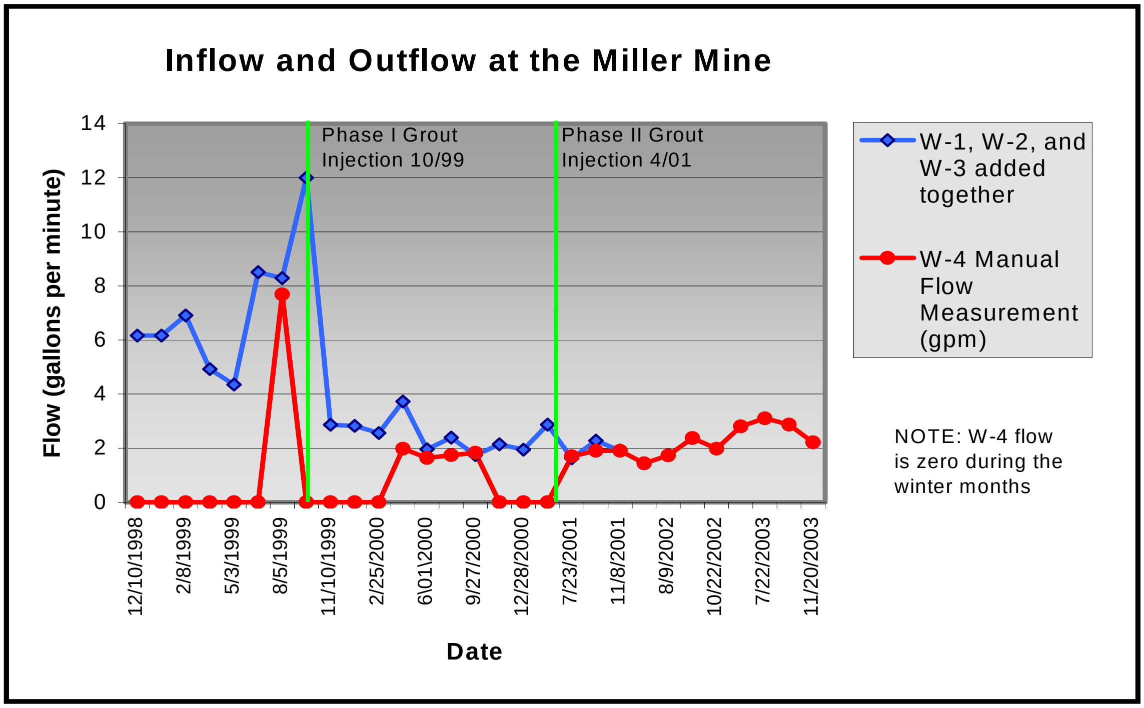


Figure 6-1. Inflow (W1, W2, W3 added together) and outflow (W4) measurements compared over time to evaluate losses and fluctuations due to grout injection.

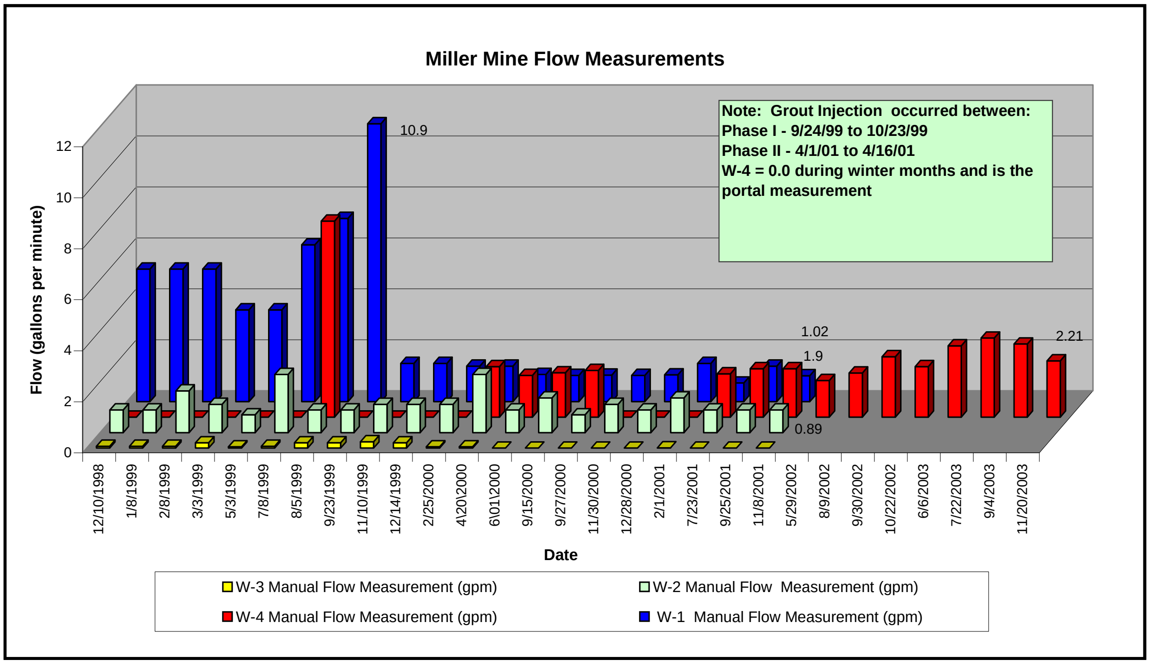


Figure 6-2. Miller Mine flow measurement taken at the respective locations throughout the mining system. Note that the Miller Mine became unstable, and measurements for W1, W2, and W3 were discontinued in November 2001.

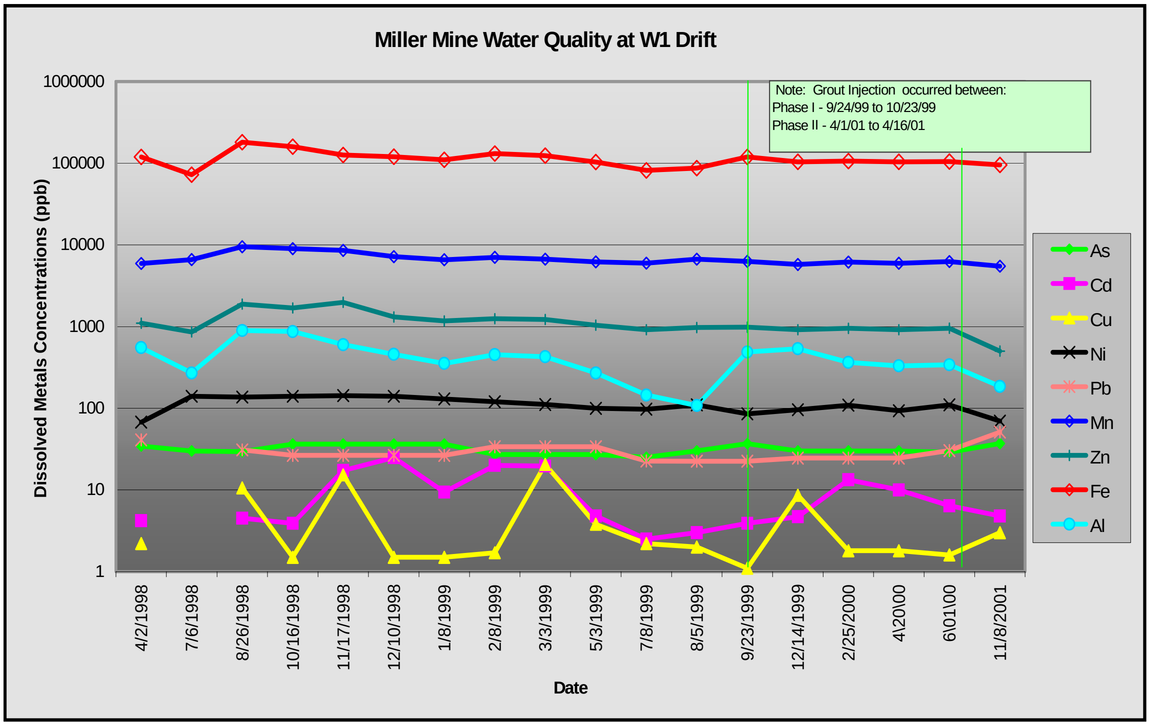


Figure 6-3. Miller Mine water quality results for the W1 drift, the area where most of the dissolved metals were originating. The water quality results reflect the dissolved metals concentrations at W1 micrograms per liter.

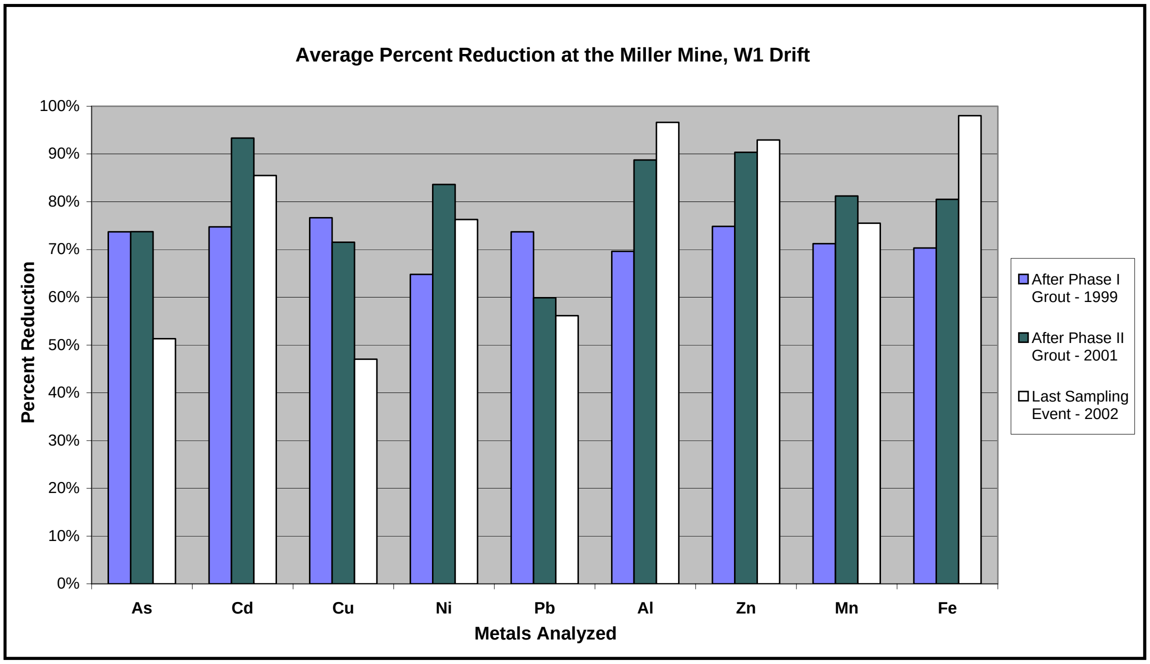
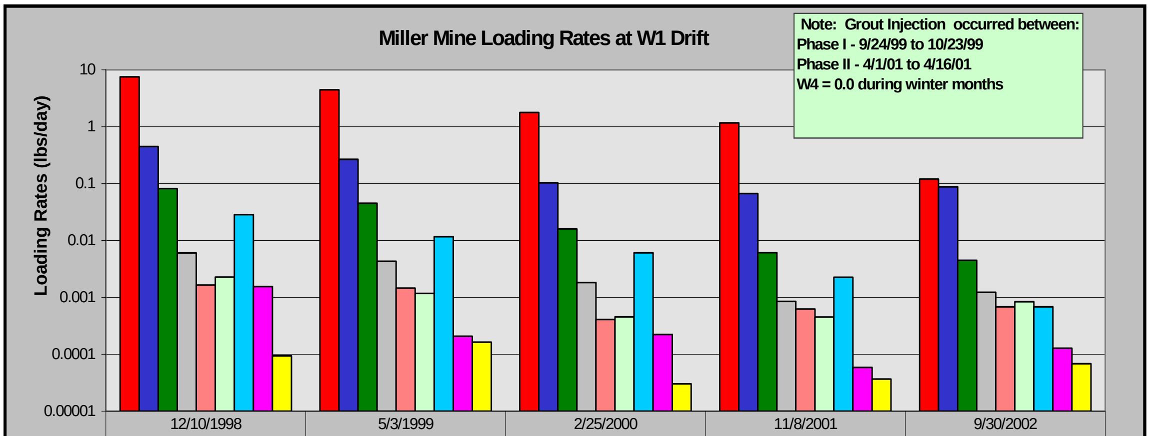


Figure 6-4. Average percent reduction for metals loading at the W1 drift prior and after each phase of grouting.



	12/10/1998	5/3/1999	2/25/2000	11/8/2001	9/30/2002
Fe (lbs/d)	7.492E+00	4.452E+00	1.774E+00	1.166E+00	1.195E-01
Mn (lbs/d)	4.483E-01	2.676E-01	1.031E-01	6.723E-02	8.764E-02
Zn (lbs/d)	8.179E-02	4.495E-02	1.595E-02	6.123E-03	4.496E-03
Ni (lbs/d)	6.056E-03	4.309E-03	1.824E-03	8.499E-04	1.229E-03
Pb (lbs/d)	1.655E-03	1.461E-03	4.101E-04	6.246E-04	6.829E-04
As (lbs/d)	2.279E-03	1.171E-03	4.536E-04	4.531E-04	8.394E-04
Al (lbs/d)	2.853E-02	1.167E-02	6.109E-03	2.266E-03	6.829E-04
Cd (lbs/d)	1.555E-03	2.075E-04	2.226E-04	5.878E-05	1.281E-04
Cu (lbs/d)	9.365E-05	1.643E-04	3.013E-05	3.674E-05	6.829E-05

Figure 6-5. Metals loading at the W1 drift sample port located in the underground workings of the Miller Mine.

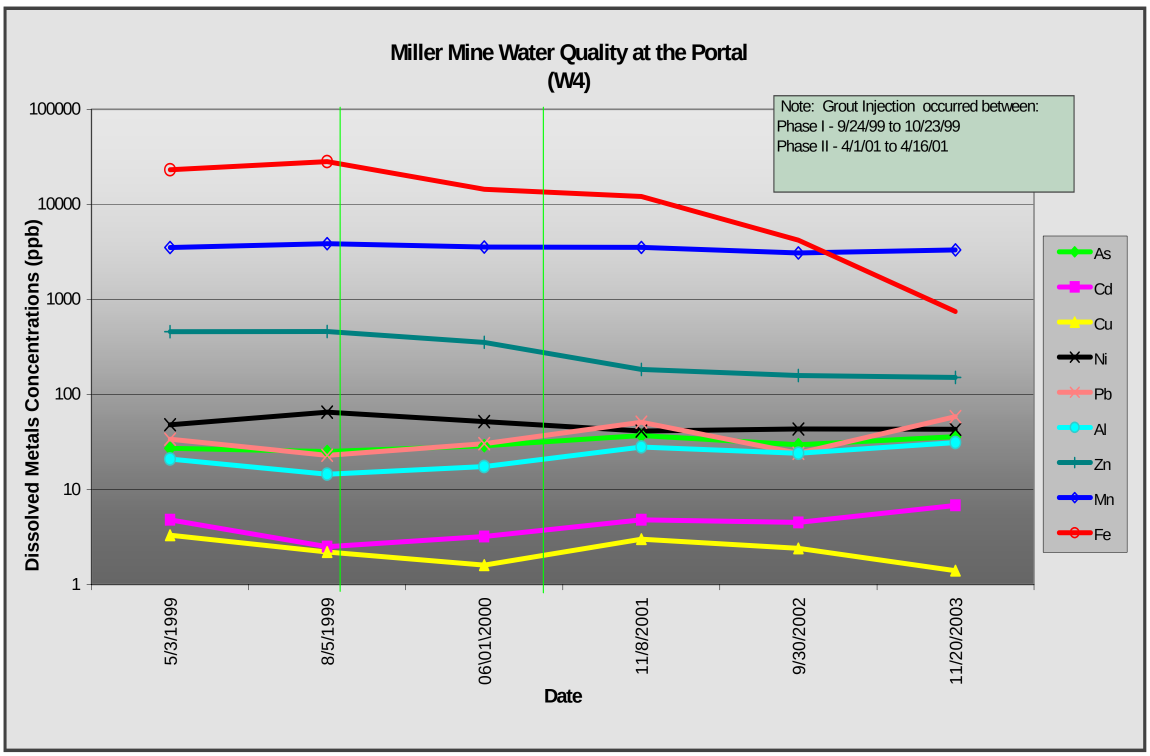


Figure 6-6. Water quality at the Miller Mine portal (W4 sample port).

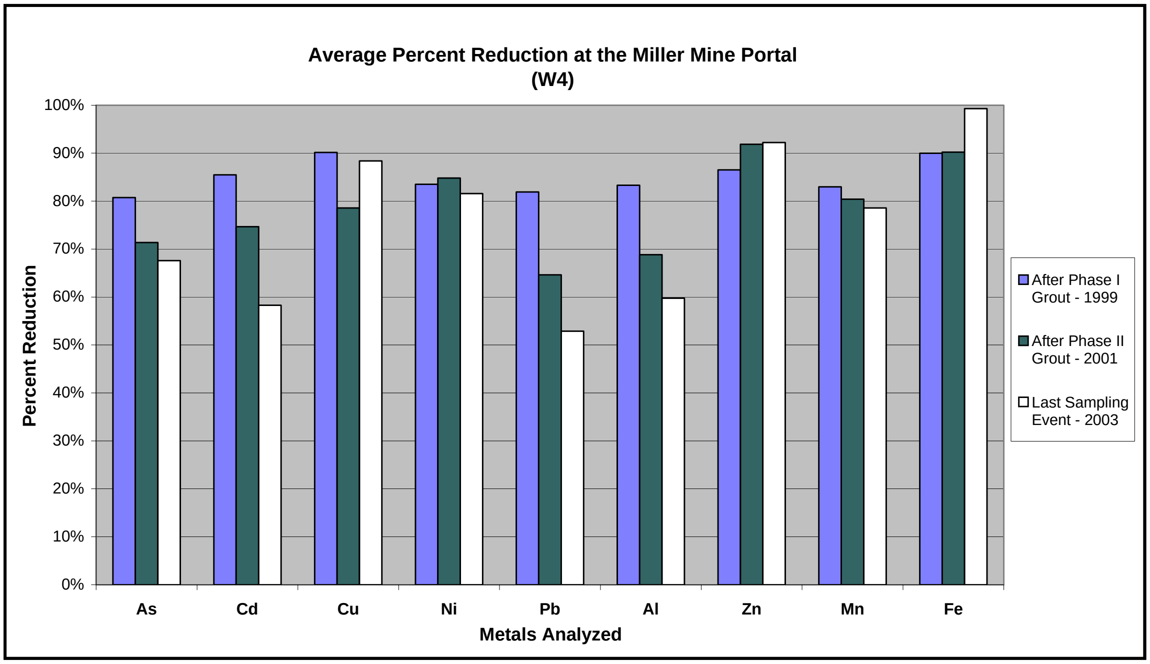
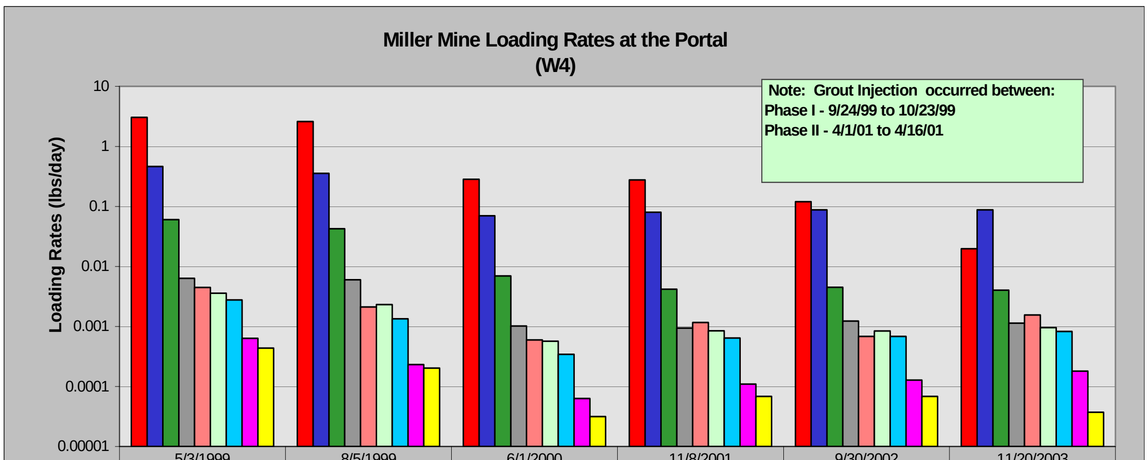


Figure 6-7. The average percent reduction in metals loading at the Miller Mine portal after grout injection.



	5/3/1999	8/5/1999	6/1/2000	11/8/2001	9/30/2002	11/20/2003
Fe (lbs/d)	3.051E+00	2.594E+00	2.829E-01	2.760E-01	1.195E-01	1.974E-02
Mn (lbs/d)	4.636E-01	3.555E-01	6.973E-02	8.030E-02	8.764E-02	8.783E-02
Zn (lbs/d)	6.036E-02	4.238E-02	6.934E-03	4.175E-03	4.496E-03	4.007E-03
Ni (lbs/d)	6.339E-03	6.011E-03	1.017E-03	9.376E-04	1.229E-03	1.138E-03
Pb (lbs/d)	4.464E-03	2.114E-03	5.952E-04	1.163E-03	6.829E-04	1.550E-03
As (lbs/d)	3.579E-03	2.318E-03	5.677E-04	8.441E-04	8.394E-04	9.552E-04
Al (lbs/d)	2.760E-03	1.339E-03	3.418E-04	6.388E-04	6.829E-04	8.252E-04
Cd (lbs/d)	6.339E-04	2.308E-04	6.286E-05	1.095E-04	1.281E-04	1.804E-04
Cu (lbs/d)	4.358E-04	2.031E-04	3.143E-05	6.844E-05	6.829E-05	3.715E-05

Figure 6-8. Metals loading for the Miller Mine portal (sampling results from port, W4).

Table 6-1. W1 Drift Recorded Flow Data Taken for the Duration of the Technology Demonstration

Date	GPD	Date	GPD
12/10/98	7,488	4/20/00	2,002
1/8/99	7,488	6/1/00	1,536
2/8/99	7,488	9/15/00	1,482
3/3/99	5,184	9/27/00	1,499
5/3/99	5,184	11/30/00	1,482
7/8/99	8,856	12/28/00	1,512
8/5/99	10,354	2/1/01	2,160
9/23/99	15,696	7/23/01	1,051
11/10/99	2,160	9/25/01	2,002
12/14/99	2,160	11/8/01	1,469
2/25/00	2,007		

Table 6-2. The volume of Flow from the Miller Mine, Drift W1, Prior to and After Each Phase of Grouting on a Gallons per Day Basis

Prior to Grouting		
Date	**Day**	**GPD**
12/10/98	0	7,488
1/8/99	29	7,488
2/8/99	60	7,488
3/3/99	83	5,184
5/3/99	144	5,184
7/8/99	210	8,856
8/5/99	238	10,354
9/23/99	287	15,696
After Phase I Grouting		
Date	**Day**	**GPD**
11/10/99	0	2,160
12/14/99	34	2,160
2/25/00	107	2,007
4/20/00	162	2,002
6/1/00	204	1,536
9/15/00	310	1,482
9/27/00	322	1,499
11/30/00	386	1,482
12/28/00	414	1,512
2/1/01	449	2,160
After Phase II Grouting		
Date	**Day**	**GPD**
7/23/01	0	1,051
9/25/01	64	2,002
11/8/01	108	1,469

Table 6-3. Flow Reductions at the W1 Drift

W1 Flow Calculations			
Grout Phase	**Time Frame**	**Total Gallons**	**Percent Reduction of Flow**
Pregrouting	December 1998 to September 1999	2,282,036	77% reduction between Pre-Grouting and Phase I
Post Phase I Grouting	December 1999 to September 2000	514,481	
Post Phase I Grouting	July 2000 to November 2000	143,490	21% increase of flow between Phases I and II
Post Phase II Grouting	July 2001 to November 2001	174,058	

Table 6-4. W1 Drift Flow Reductions with Error Ranges

Grout Phase	Time Frame	Flow Volume	+5% Volume	-5% Volume	Max % reduction	Min % reduction
Pregrout	12/98 - 9/99	2,282,036	2,396,138	2,167,934	79.6%	75.1%
Phase I	12/99 - 9/00	514,481	540,205	488,757		
Phase I	7/00 - 11/00	143,490	150,665	136,316	-9.8%	-34.1%
Phase II	7/01 - 11/01	174,058	182,761	165,355		

Table 6-5. Water Quality Results for the Miller Mine after Both Phases of Field Emplacement of the Source Control Technology

MWTP – Field Sampling Comparison

Site: Miller Mine

MWTP, Activity III, Project 8

Sample Location: Comparison of All Sample Locations

Sample Date:	7/8/1999	11/8/2001	11/8/2001	7/8/1999	11/8/2001	11/20/2003	MCL & Secondary MCL, mg/l
Sample Location:	W1	W1	W2	W4	W4	W4	
Sample Matrix:	Water	Water	Water	Water	Water	Water	
Mine Inflow or Outflow	Inflow	Inflow	Inflow	Outflow	Outflow	Outflow	
pH (su)*	6.08	5.95	6.4	6.58	6.65	6.75	6.5 - 8.5
Sulfate*	828	648	941	539	510	453	250
Dissolved Metals (mg/L, ppm)							
Al	0.145	0.185	0.028	0.010	0.028	0.031	0.050 - 0.200*
As	0.025	0.037	0.037	0.025	0.037	0.036	0.050 until 1/06
Cd	0.003	0.0048	0.0048	0.003	0.0048	0.0068	0.005
Cu	0.002	0.003	0.003	0.002	0.003	0.0014	1.000*
Fe	81.600	95.200	17.600	18.600	12.100	0.744	0.300*
Pb	0.023	0.051	0.051	0.023	0.051	0.058	0.015
Mn	5.970	5.490	0.956	3.220	3.520	3.310	0.050*
Ni	0.097	0.069	0.020	0.046	0.041	0.043	0.088**
Zn	0.915	0.500	0.0949	0.388	0.183	0.151	5.000*

* Secondary MCL

** Aquatic Life Standard for Freshwater

7. Summary of Quality Assurance Activities

7.1 Background

The following is a summary of the quality assurance (QA) activities associated with MWTP Activity III, Project 8, Underground Mine Source Control. Analytical samples and field data were collected according to the schedule outlined in the approved project-specific QAPP. All field and laboratory data available was evaluated to determine the usability of the data. Phase I critical analyses were surface water flow rate manually, surface water flow rate by weir, field pH, and dissolved metals (Al, As, Cd, Cu, Fe, Pb, magnesium (Mg), Mn, Ni, K, Na, and Zn). Phase III critical analyses were surface water flow rate manually and surface water flow rate by weir. Phase IV critical analysis was surface water flow rate manually. A critical analysis is an analysis that must be performed in order to determine if project objectives were achieved. Data from noncritical analyses were also evaluated.

7.2 Project Reviews

An external technical systems audit of the project field activities and the HKM Laboratory was performed by Science Applications International Corporation, subcontractor to EPA, on September 27, 2000. There were no findings, two observations, and one additional technical comment identified during the audit.

The observations included deviations from the quality objective for surface water flow manually and inadequate calibration verification checks on the pH meter. Amendments were made to the QAPP to correct the observations. The additional technical comment concerned the difference (30% or greater) between the manual head reading on the weirs and flumes and the pressure transducer reading. The reason for the inaccurate readings from the pressure transducers was investigated. The QAPP was amended to state that manual readings would be the sole head measurements.

7.3 Data Evaluation

Data that were generated throughout the project were validated. The purpose of data validation is to determine the usability of data that were generated during a project. Data validation consists of two separate evaluations: an analytical evaluation and a program evaluation.

7.3.1 Analytical Evaluation

An analytical evaluation of all data was performed to determine the usability of the data that were generated by HKM Laboratory for the project. Laboratory data validation was performed using *USEPA Contract Laboratory Program National Functional Guidelines for Inorganics Data Review* (Ref. 7) as a guide. The data quality indicator objectives for critical measurements were outlined in the QAPP and were compatible with project objectives and the methods of determination being used. The data quality indicator objectives were method detection limits (MDLs) accuracy, precision, and completeness. Control limits for each of these objectives are summarized in Tables 7-1 and 7-2. The quality control (QC) criteria were also used to identify outlier data and to determine the usability of the data for each analysis.

Measurements that fall outside of the control limits specified in the QAPP, or for other reasons were judged to be outliers and were flagged appropriately to indicate that the data were judged to be estimated or unusable. All data requiring flags are summarized in Table 7-3.

7.3.2 Program Evaluation

Program evaluation includes an examination of data generated during the project to determine that all field QC checks were performed and within acceptable tolerances. Program data that were inconsistent or incomplete and did not meet the QC objectives outlined in the QAPP were viewed as program outliers and were flagged appropriately to indicate the usability of the data.

7.3.2.1 Surface Water Flow Rate

Surface water flow rate was measured with weirs, a flume, and manually. The flume/weir water depth was measured with pressure transducers and staff gauges installed at each weir and flume. During the field activities, it was discovered that the pressure transducer measurements and staff gauge measurements differed by as much as 30% or greater. It was apparent that the pressure transducers did not stay constant; therefore, the manual reading of the staff gauges became the sole measurement for flume/weir water depth.

As outlined in the QAPP, manual checks of surface water flow rate were performed at each available weir location during sampling events at the site. Manual flow rate measurements were performed using a graduated cylinder and a stopwatch. Water was collected in a 1-L graduated cylinder. The time necessary to fill the graduated cylinder was measured with a stopwatch. The flow rate was calculated by dividing the volume collected by the time elapsed. The test was then repeated to ensure the precision of the method. The results of the duplicate test were to be within ± 200 milliliters per minute (mL/min) for the flow rate measurement to be valid. If the results of the duplicate test were not within ± 200 mL/min, a third manual flow rate check was performed and compared to the other flow rate measurements. If the third measurement was within ± 200 mL/min of one of the first two measurements, an average of the two comparable measurements was used for reporting purposes.

The surface water flow rate measurements were obtained in accordance with the procedures outlined in the QAPP. No surface water flow rate data was judged to be outlier.

7.3.2.2 pH

The pH measurements were a critical analysis for Phase I. The pH measurements were collected using a pH meter with automatic temperature compensation capable of measuring pH to ± 0.1 pH units. The pH probe was calibrated daily using two fresh buffer solutions (4 and 7). Meter calibration was verified following initial calibration and every 10 samples using a third buffer solution (pH 5) within the calibration range. The QAPP required that the calibration verification standard be within 0.2 pH units of the true value and that duplicate pH readings be conducted every 10 samples or 1 per batch, whichever is more frequent. The calibration verification standard was within acceptable limits. The QAPP also required that a sample duplicate be conducted every 10 samples or 1 per batch, whichever is more frequent. Sample duplicates were collected for each batch, but three of the samples did not have a pH measurement recorded in the logbook or field notes; therefore, the pH data for those three sampling events was flagged "J" as estimated. A summary of the flagged data is presented in Table 7-3.

7.3.2.3 Dissolved Metals

Dissolved metals analysis was a critical analysis for Phase I. Aqueous samples were collected from the four sampling locations during each sampling event, as well as a duplicate sample from a predetermined sampling location. Sampling procedures for the collection of the aqueous samples outlined in the QAPP were followed. The samples were taken to HKM Laboratory for analysis by inductively coupled plasma emission spectrometer (ICP-ES). No dissolved metals data was judged to be outlier.

7.4 Summary

Although field log sheets were used to collect specific data during sampling events, critical activities should be documented in the field logbooks. In addition, one logbook should be used for recording sampling activities; this project used nine logbooks. The information provided in the logbook needs to be expanded and better organized so that anyone reviewing the logbook can clearly understand what occurred at each sampling event. The importance of logbook protocol should be reiterated to sampling personnel.

Table 7-1. QA Objectives for Accuracy, Precision, MDL, and Completeness

Measurement	Units	MDL	Precision[1]	Accuracy	Completeness[2]
Flume/weir water depth	Inches	0.03	N/A[3]	± 5%[4]	95%
Surface water flow rate (weir/flume)	Gpm	1	N/A[3]	± 5%[4]	95%
Surface water flow rate (manual)	mL/min	200 mL/min	± 200 mL/min[5]	N/A[4]	95%
PH	S.U.	2	± 0.2[6]	± 0.2[7]	95%
Dissolved metals	mg/L	See Table 6-2	≤ 20% relative percent difference (RPD)	75%-125% spike recovery	95%

[1]Precision will be determined by the RPD of duplicates, unless otherwise indicated.
[2]Completeness is based on the number of valid measurements, compared to the total number of samples.
[3]Duplicate measurements of field process measurements will not be taken. All equipment is calibrated against National Institute of Standards and Technology (NIST) traceable standards.
[4]Accuracy of weirs/flumes will be ensured by installing flumes and weirs according to Standard Operating Procedure (SOP) H6-6 and avoiding installation locations that could adversely affect weir/flume accuracy (approach conditions do not allow uniform velocity distribution, damage to weirs or flumes, and changes in weir or flume dimensions). In addition, manual flow rate measurements will give an indication of whether the weirs and flumes are returning reasonable flow rate measurements.
[5]Precision of manual surface water flow rate measurements will be determined by the absolute difference between consecutive measurements.
[6]Precision of pH measurements will be based on the absolute difference of duplicate readings.
[7]Accuracy of pH measurements will be based on absolute difference of reading compared to standard buffer solution.

Table 7-2. Instrument Detection Limits (IDLs) for ICP Analysis of Dissolved Metals

Analyte	IDL (ppm)[1]	MDL (ppm)[2]
Al	0.017	0.05 – 2.0
As	0.026	0.05
Cd	0.004	0.01
Calcium	0.010	N/A
Cu	0.002	1.3
Fe	0.014	0.3
Pb	0.024	0.05
Mg	0.009	N/A
Mn	0.003	0.05
Ni	0.015	N/A
Potassium	0.017	N/A
Sodium	0.007	N/A
Zn	0.009	5.0

[1] These IDLs are considered sufficiently low for the characterization of this site.
[2] Based on National Primary and Secondary Drinking Water Regulations.

Table 7-3. Summary of Flagged Data for Activity III, Project 8

Date of Collection	Sample ID	Analysis	Quality Criteria	Flag	Comment
8/26/1998	MMW1 MMW2 MMW3 MMW4	pH	Duplicate every 10 samples or one per batch, whichever is more frequent	J	No duplicate pH samples recorded in field logbook or field notes. The associated samples should be flagged "J" as estimated.
10/16/1998	MMW1 MMW2 MMW3 MMW4	pH	Duplicate every 10 samples or one per batch, whichever is more frequent	J	No duplicate pH samples recorded in field logbook or field notes. The associated samples should be flagged "J" as estimated.
11/17/1998	MMW1 MMW2 MMW3 MMW4	pH	Duplicate every 10 samples or one per batch, whichever is more frequent	J	No duplicate pH samples recorded in field logbook or field notes. The associated samples should be flagged "J" as estimated.

Data Qualifier Definition:
J - The measurements are estimated.

8. Conclusions and Recommendations

The MWTP, Activity III, Project 8, Underground Mine Source Control Demonstration Project, reduced influx by injecting a water-activated polyurethane grout through the water-bearing fracture systems associated with the Miller Mine fault zone and secondary fractures associated with the contact of Belt sedimentary rocks and intrusive igneous rocks. During the technology emplacement, the limit of grout communication between core holes was controlled by the reduction in hydraulic conductivity of the fracture system and the viscosity of the polyurethane grout. Grouted fracture hydraulic conductivity is a function of not only the fractured medium but also of the grout density, viscosity, and rate of reactivity.

Two fracture-shear systems were identified and mapped from core data. This included the fracture-shear systems associated with the movement on the MMRF and fractures oriented at high angles to that structure. It is believed that the MMRF-related fracture-shear system provided the principal water path for AMD into the W1 drift. Moreover, the two fracture systems are interconnected in the immediate vicinity of the W1 drift and act together to conduct AMD charged groundwater into the W1 drift.

Grout injection into the rock fracture system and associated reduction of AMD was observed in the W1 drift and W4 (adit) following completion of the 1999 Stage I and 2001 Stage II grouting programs. The Stage I application was responsible for reducing the majority of the discharge from W4 and the W1 drift. The primary criteria for the success of the demonstration project was to reduce the cumulative volumetric flow at W1 by 95% using flow data from 10 months prior to and 10 months after both grout injections into the Miller Mine fracture system. However, average calculated flow at the W1 sample location, prior to the first grout emplacement in 1999, was 5.88 gpm, and the flow after both grout emplacements averaged 1.24 gpm, providing an approximately 77 +/- 5% reduction. Approximately 75% of the flow reduction at the Miller Mine was achieved within 1 month after the completion of the primary grouting program in 1999. Long-term flow and monitoring results indicate that the grouting program reduced the flow from the Miller Mine adit (W4), as well as the concentrations of dissolved metals. Flow data and the analyses of dissolved metals from samples taken at the W4 sample site were recorded for 4 years after completion of the primary grouting program in 1999.

Prior to the application of the source control technology, flow results and visual observation revealed that most of the flow into the workings occurred in the W1 drift (Figure 6-2). After grouting, flow in W1 was reduced; however, flow recorded from the W2 and W3 drifts indicate that they were not affected by the grout emplacement even with the close proximity of the workings. The last flow measurements taken inside the Miller Mine workings were on October 8, 2001. On that date, 53% of the flow was from the W1 drift and 47% was from the combined flows from the W2 and W3 drifts that were measured at the W2 weir. As the flow at W4 increased from 1.9 gpm to approximately 3.0 gpm in September 2003, it is unknown where the increase in flow originated because access to the underground mine workings was restricted.

The water quality of the water discharging at the Miller Mine portal (W4) improved as a result of the application of the source control technology. Water quality improvement was directly related to the reduction of influx into the W1 drift. The metal concentrations from W1 were diluted as the water from W1 merged with water from the W2 drift. The concentration of dissolved metal in the portal discharge significantly dropped after the application of each phase of the source control technology. Iron and Zn concentrations were reduced the most when comparing 1999 pregrout and 2003 postgrout analytical results. The dissolved concentrations for Pb did increase; however, due to the decrease in flow, Pb loads were on average reduced by 50%.

As calculated, the average percent reduction of dissolved metal loads was greater than 50% for all metals analyzed. Both Zn and Fe loading was reduced as much as 90%, and the Phase II/Stage II grouting each assisted with further reduction in the metals loading (Figure 6-8). The most significant reduction in metals loading at the Miller Mine portal, 600 ft from the W1 drift, included:

- Fe from 3.05 lb/d to 0.02 lb/d;
- Mn from 0.46 lb/d to 0.09 lb/d; and
- Zn from 0.06 lb/d to 0.004 lb/d.

Most of the dissolved metals concentrations were reduced by an order of magnitude or more. Not only is this a direct result of the reduction of the flow containing heavy metals, but it is also a result of the encapsulation of the ore/sulfide zones by the grout. The technology application eliminated the possibility of AMD formation by eliminating either the oxygen or water on the surface of the sulfide material. Due to the emplacement of grout, the water was unable to flow in the highly fractured, sulfide zones; thus, water flows in areas where the sulfide content is lower and does not contain heavy metals. This reduces the amount of metals loading in the Miller Mine discharge and as flow increases to the maximum of 3 gpm, the metals loading is low.

8.1 Lessons Learned

Future grout programs or any additional grouting of the Miller Mine should consider the suggestions listed below.

- Obtain core samples from the site to determine relative hydraulic conductivity of the fracture pattern prior to beginning coring and drilling operations. Occasionally core description and intact core is available. This data may be helpful in selecting an optimum grout for a given fracture system.

- Install an in-line mixer if expansive grouts are used to thoroughly mix grout and water prior to injection. This would ensure that all injected grout would be activated and expand within expected time constraints.

- Install an in-line heating system to reduce the viscosity of the grout during injection if a grout of lower viscosity is unavailable. This would reduce the viscosity of a viscous grout, provide for greater hydraulic conductivity of the fracture system, and increase the likelihood that the fracture system would be sealed.

- Consider using the Jackleg drill system more frequently. This would provide the opportunity to drill and grout many short holes in the vicinity of known seeps and conduits. In addition, this drill system is more mobile and can be used in tight and confined areas.

9. References

1. U.S. Environmental Protection Agency, *Site Characterization and Materials Testing Report: Underground Mine Source Control Demonstration Project, MWTP, Activity III, Project 8*, MWTP-145, April 1999.

2. U.S. Environmental Protection Agency, *Quality Assurance Project Plan Underground Mine Source Control Demonstration Project, MWTP, Activity III, Project 8*, MWTP-112R1, October 1999.

3. Johnson, E.A., *Geology and Gold Deposits of the Confederate Gulch-White Gulch Area, Broadwater County, Montana*, Montana College of Mineral Science and Technology, May 1973.

4. U.S. Department of the Interior, Water and Power Resources Service, *Ground Water Manual*, pg. 257-285, 1981.

5. Houlsby, A.C., *Construction and Design of Cement Grouting, A Guide to Grouting in Rock Foundations,* A.S.T.C., F.I.E. Aust., F. ASCE, John Wiley & Sons, Inc., New York, pg. 123-133, 1975.

.

6. Western Regional Climatic Center, Historical Climatic Information, Monthly Precipitation Listing, *Monthly Totals, 1998 – 2001*, http://www.wrcc.dri.edu/ CLIMATEDATA.html

7. U.S. Environmental Protection Agency, *USEPA Contract Laboratory Program National Functional Guidelines for Inorganics Data Review*, 1994.