REM IV

Remedial Planning Activities
at Selected Uncontrolled
Hazardous Waste Sites–Zone II

Environmental Protection Agency
Hazardous Site Control Division
Contract No. 68-01-7251

FINAL DRAFT

Groundwater/Surface Water
Operable Unit Feasibility Study
Supplement

Galena Subsite
Cherokee County, Kansas

July 1989

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FINAL DRAFT

Groundwater/Surface Water Operable Unit Feasibility Study Supplement

Galena Subsite
Cherokee County, Kansas

July 1989

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DEFINITIONS

CHAT	A coarse, sand or larger sized waste product from the recovery and concentration of minerals from ores, usually lead and/or zinc.
DEIONIZED WATER	Water that has passed through an ion exchange resin to remove selected cations and anions.
FROTH FLOTATION	A metallurgical process by which minerals are separated from each other or rock using mechanically produced froth. The minerals are treated to improve their ability to attach to froth bubbles.
LEACH TESTS	Laboratory or pilot-scale tests where the rate of transfer of one substance from another substance, usually a metal from an ore or waste material, is measured under a set of selected, measurable conditions.
MASS LOADING	The amount (mass) of dissolved metals, salts, or compounds per unit of time contained in the flow of groundwater or surface water.
MILL CIRCUIT	A combination of selected mineral processing unit operations used to treat and recover a mineral from ore (e.g., crushing, grinding, flotation).
MILLING	A particular unit operation where ore is ground, using a mill, to a size where the valuable mineral is liberated from the rock, and is then recovered in subsequent steps. Chat is a waste product of early milling processes.
MINE WASTES	Mine development rock, low grade ores, chat, and/or tailing with low economic value now or at the time of mining.
MINING	The process by which ores are extracted from the earth.
O&M COSTS	Operating and maintenance costs associated with implementation of a remedy that ensures continued and long-term effectiveness.

OXIDES	A compound composed of oxygen with another element or a metal (e.g. lead oxide).
POTABLE WATER	Water that is suitable for drinking.
RECONTOURING	Reshaping of the earth's surface, usually done to aid in control of surface water runoff.
ROUBIDOUX AQUIFER	A large, deep, and generally confined groundwater aquifer underlying southeastern Kansas.
SCREENING	The physical unit process in which granular material is separated into various-sized fractions.
SHALLOW AQUIFER SYSTEM	An aquifer system nearest to the ground surface with subsequent deeper aquifers below.
SIZING	See the definition for screening.
SUBSITE	A geographically smaller section of a Superfund site selected to be studied separately. For example, the Galena Subsite is a subsite of the Cherokee County Site.
SULFIDES	Typically, metal(s) combined with sulfur, associated with a lead-zinc ore body.
TAILING	A fine-sized waste product from the recovery and concentration of minerals from ores.
WASTE ROCK	Country rock, and/or low-grade ores left over from mining development with little or no economic value.
X-RAY FLUORESCENCE SPECTROMETER	An analytical instrument that uses X-rays to excite specific metallic elements (i.e., lead, zinc, etc.) in a sample, then measures the amount of energy they release. The energy released is directly proportional to the element's concentration in the sample.

ABBREVIATIONS

AIC	Acceptable Intake for Chronic Exposure
AKNL/AKAL	Alternate Kansas Notification/Action Levels
AMD	Acid Mine Drainage
ARAR	Applicable or Relevant and Appropriate Requirements
ATSDR	Agency for Toxic Substances and Disease Registry
AWQC	Ambient Water Quality Criteria
CCC	Criterion Continuous Concentration
CERCLA	Comprehensive Environmental Response, Compensation, and Liability Act of 1980
CLP	Contract Laboratory Program
CMC	Criterion Maximum Concentration
CWA	Clean Water Act
EPA	U.S. Environmental Protection Agency
FS	Feasibility Study
FSDWA	Federal Safe Drinking Water Act
GPR	Ground Penetrating Radar
GW/SW	Ground Water/Surface Water
IARC	International Agency for Research on Cancer
ICP	Inductively Coupled Plasma
IRIS	Integrated Risk Information System
KAR	Kansas Administrative Regulations
KDHE	Kansas Department of Health and Environment
KNL/KAL	Kansas Notification/Action Levels
MCL	Maximum Contaminant Level
MCLG	Maximum Contaminant Level Goal

NAS	National Academy of Sciences
NCP	National Contingency Plan
NPL	National Priorities List
O&M	Operating and Maintenance
OUFS	Operable Unit Feasibility Study
PRP	Potentially Responsible Party (pursuant to Section 107 of CERCLA, 42 USC 9607)
RfD	Reference Dose
RI	Remedial Investigation
RCRA	Resource Conservation and Recovery Act
ROD	Record of Decision
SARA	Superfund Amendments and Reauthorization Act
SDWA	Safe Drinking Water Act
SP	Spontaneous Potential
USGS	U.S. Geological Survey
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence

Section 1
INTRODUCTION

PURPOSE

A remedial investigation (RI) (EPA, 1986) and two operable unit feasibility studies (OUFS) (EPA 1987a and EPA, 1988a) have been conducted at the Galena Subsite of the Cherokee County Site. The first OUFS considered providing an alternative water supply to those parties currently relying on the shallow aquifer system for their potable water supply. A record of decision (ROD) was issued for this operable unit in December 1987. The new water system is expected to be completed by the winter of 1990.

The second OUFS (Groundwater/Surface Water Operable Unit) was prepared in 1987 and 1988 and addressed the means to remediate the metals-contaminated surface water and shallow groundwater that resulted from past mining activities. From the alternatives developed in the OUFS, EPA subsequently developed a preferred remedy in February 1988 and requested public comment (EPA, 1988c). The February 1988 preferred remedy consisted of:

- Collecting and milling the surface-deposited mine wastes
- Recovering the lead and zinc sulfides by froth flotation with tailing disposal in the open mine voids
- Recontouring and revegetating disturbed areas
- Diverting and channeling selected surface water drainages

- Rehabilitating or plugging defective wells that penetrate to the Roubidoux Aquifer

In the summer and fall of 1988, EPA obtained additional samples of surface-deposited mine wastes and performed bench-scale tests on them to substantiate the assumptions and analyses made during the OUFS. In addition, the potentially responsible parties (PRPs) conducted additional field studies on the mine wastes. EPA then developed a work plan for leach tests, that was conducted in the laboratory by the PRPs and monitored by the EPA during the fall of 1988.

The results of the additional field work and test work conducted by the EPA and the PRPs led to the decision to review the February 1988 preferred remedy as well as to develop additional alternatives for consideration. This supplement to the Groundwater/Surface Water (GW/SW) OUFS presents detailed analyses of those new alternatives that are consistent with the most recent guidance for Remedial Investigation/Feasibility Studies (RI/FS) (EPA, 1988b). In addition, a comparative analysis using criteria in effect when the 1988 OUFS was written is also presented for clarity (EPA, 1987b).

BACKGROUND

The Galena Subsite of the Cherokee County CERCLA site is located in southeastern Kansas and is one of six subsites designated by the EPA within the site (Figure 1-1). The Cherokee County Site is the portion of the Tri-State Mining District within Kansas on the Superfund National Priorities List where mining for lead and zinc was conducted for many years. Mining at the Galena Subsite began in the 1870s and

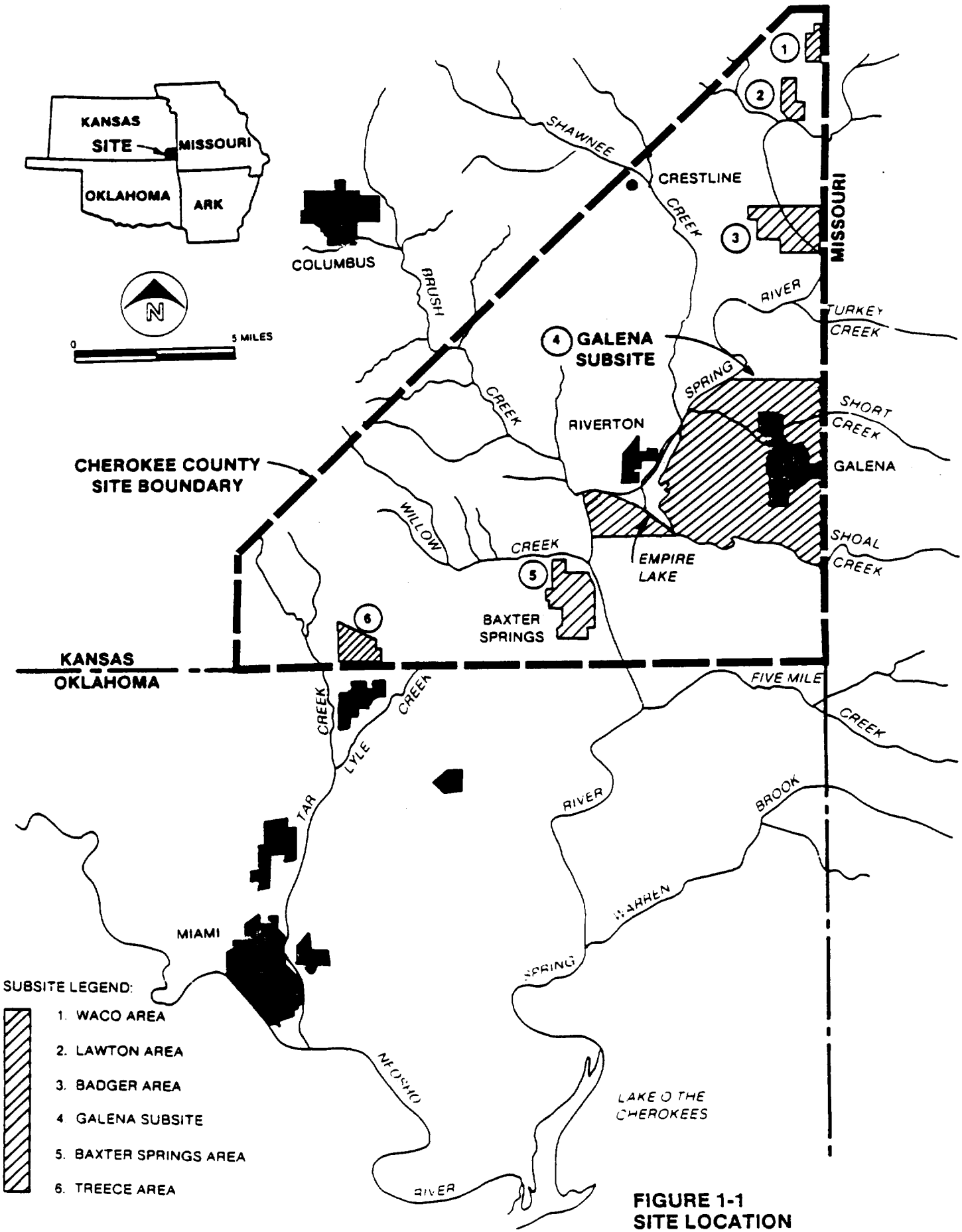


FIGURE 1-1
SITE LOCATION
CHEROKEE CO., KANSAS
GALENA SUBSITE—OUFS
GROUNDWATER/SURFACE WATER

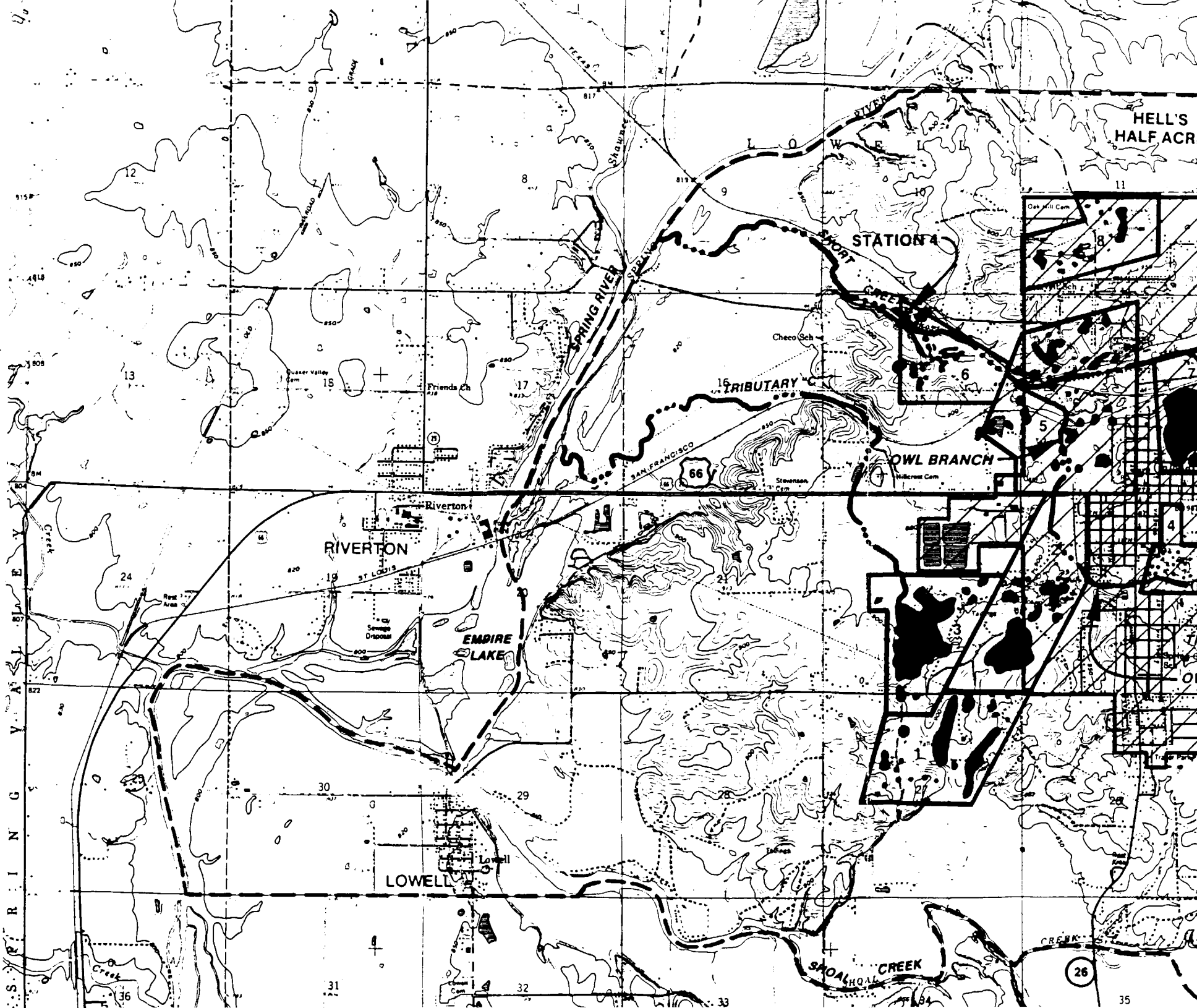
ended in the 1920s. The mining-related contaminants found at the Galena Subsite include lead, zinc, cadmium, and other metals in the surface and groundwater systems and in the surface-deposited mine wastes.

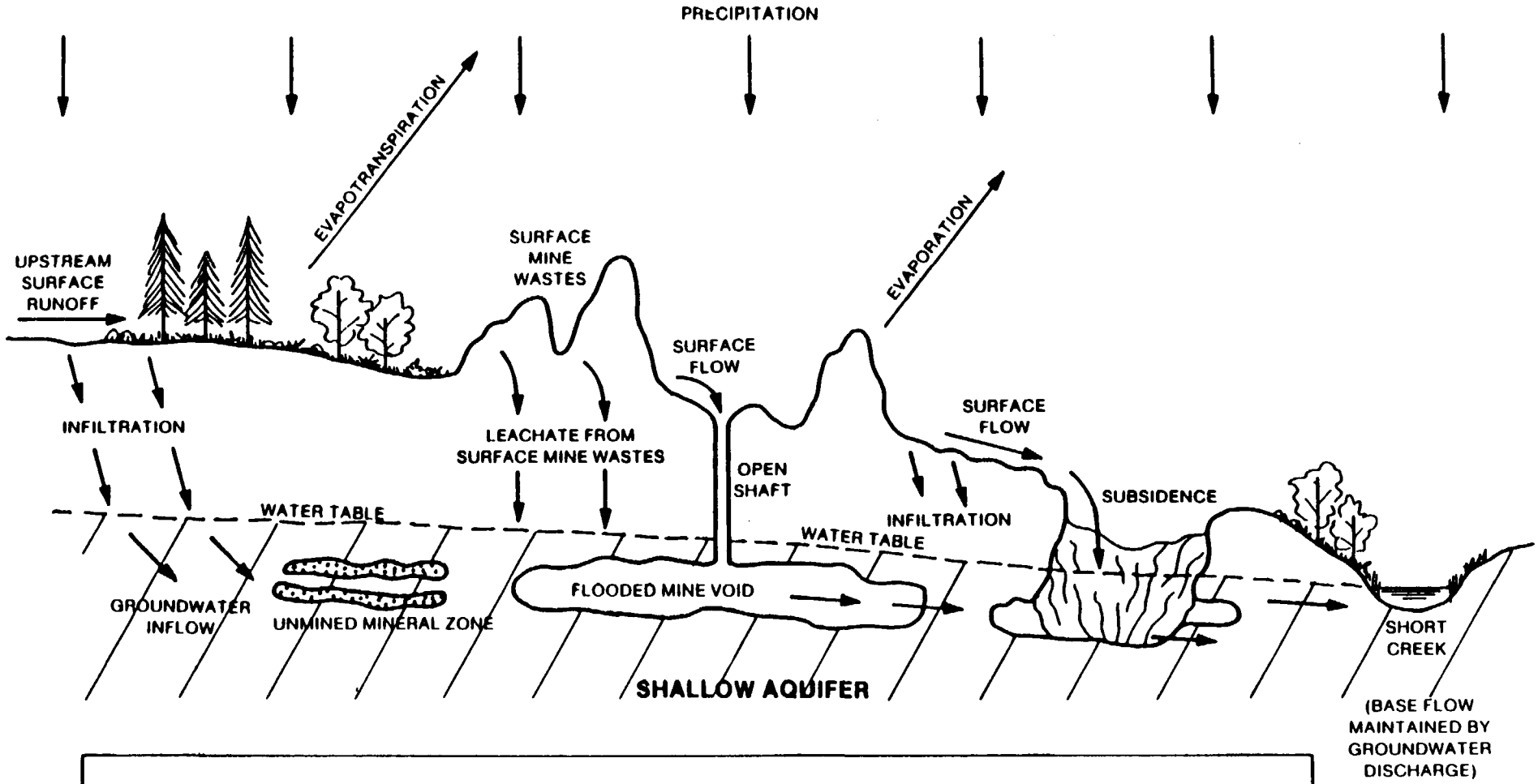
Two groundwater aquifers are in the Galena area. One is a shallow water table (unconfined) aquifer and the other is a deeper, confined aquifer. Wells placed in the shallow aquifer are usually less than 200 feet deep, while deep aquifer wells are generally greater than 1,000 feet deep. Because mining occurred in the formations containing the shallow aquifer, the wells in this formation are potentially subject to mining-related contamination.

The shallow aquifer is used by many residents within the Galena Subsite as their primary source of potable water (private shallow wells). Most, but not all, of these residents are expected to use the alternate water supply system. The regional deep aquifer, frequently referred to as the Roubidoux Aquifer, is used as a water source by many cities, rural water districts, and industries in Kansas and Oklahoma. Six deep aquifer wells have been located in the subsite. These wells could be potential conduits for the migration of contaminated shallow aquifer water into the deep aquifer if the wells were constructed incorrectly or are failing structurally.

The Galena Subsite drains to the Spring River, Short Creek, Shoal Creek, and their tributaries (Figure 1-2). The major drainage basin is the Short Creek watershed, which flows east to west (from Missouri) through the northern portion of the Galena Subsite.

Mining activities changed the hydrology of the subsite by creating underground voids, causing subsidences, and leaving mine wastes on the surface (Figure 1-3). These actions


HELL'S
HALF ACR
STATION 4
SPRING RIVER
TRIBUTARY
OWL BRANCH
RIVERTON
EMPIRE
LAKE
LOWELL
SHOAL CREEK
SPRING VALLEY
Checo Sch
Friends Ch
Riverton
Lowell
Oak Hill Cem
Quaker Valley Cem
Stevenson Cem
Hillcrest Cem
City Sewage Disposal
Rest Area
Tailings
66
26



WATER (WITH OXYGEN) + SULFIDE MINERALS $\xrightarrow[\text{(CATALYST)}]{\text{FERROOXIDANS}}$ ACID MINE DRAINAGE (AMD) (SULFURIC ACID AND DISSOLVED METALS)

FIGURE 1-3
SURFACE AND GROUND WATER INTERACTIONS
CHEROKEE COUNTY, KANSAS
GALENA SUBSITE — OUFS
GROUNDWATER/SURFACE WATER

disrupted the normal surface drainage and depleted the vegetation, thereby increasing infiltration of surface water into the shallow groundwater system. In some cases, the entire flow of a surface stream is captured by a subsidence or shaft, directing it into the shallow aquifer. Over much of the area covered by mine wastes, rain infiltrates the ground because of its highly permeable condition and the closed surface drainages. The shallow groundwater system, much of which is acidic and contains elevated levels of metals, in turn, flows into the major creeks such as Short Creek and maintains their year-round base flows. The interconnected shallow mine workings provide a conduit that can greatly enhance lateral groundwater movement.

Mine wastes resulting from shaft excavation, ore milling processes, and smelter operations have been disposed of on the ground near mine shafts and mill sites (Figure 1-2). These mine wastes contain residual sulfide minerals, plus the oxide, carbonate, and sulfate minerals resulting from sulfide mineral weathering. The term "mine wastes," as used in this report, refers to all mining-related wastes deposited on the surface of the Galena Subsite. The mine wastes include "waste rock," remaining from shaft and mine excavation, and "chat," the finer sized waste material from the early ore milling processes. "Slag," another mine waste from smelter operations, has been largely removed from the subsite.

Public Health Environmental Risks

The potential hazards to public health and the environment posed by metals concentrations in water and mine wastes within the Galena Subsite were assessed in the 1988 OUFS. The following paragraphs have been abstracted from that report.

For children and adults, the primary exposure pathways of concern are ingestion of contaminated groundwater and ingestion of metals-laden mine wastes. A review of the contaminants potentially present in the groundwater and surface materials indicated that no contaminants were found that would pose a quantifiable cancer risk (carcinogens). However, through the exposure pathways evaluated, concentrations of several metals were found that could be toxic. To evaluate the risks from exposure to these metals, the daily intakes of the metals were estimated and compared to established acceptable intakes or reference doses.

Based on the average metals concentrations in shallow-aquifer wells, cadmium and lead concentrations were determined to present health risks for children. None of the metals in the groundwater at current concentrations present a health risk for adults. Incidental ingestion of mine wastes presents a risk to children because of the amount of cadmium, chromium, and lead in the mine wastes. Incidental ingestion of mine wastes would also present a risk to adults because the estimated adult lead intake would exceed the acceptable intake value.

Water quality and biological data indicate that the aquatic biota in the upper reach of the Spring River in Kansas are generally unaffected by mining-related contaminants, although some tributaries above the Galena Subsite do contribute metals to the watershed. However, Short Creek, located within the subsite, and Willow Creek and Spring River downstream from the Galena Subsite (see Figure 1-2), are affected adversely by mining-related contaminants to varying degrees.

Both Empire Lake and Spring River upstream of the lake support a diverse and apparently stable fish population. Downstream of Empire lake, however, Kansas and federal ambient water quality criteria are commonly exceeded in the river.

Biological impairment has been reported by several studies. Although possible environmental effects from the Galena Subsite cannot be clearly distinguished from possible effects from other local sources, sampling during the Galena Subsite remedial investigation indicated that Short Creek (which drains much of the Galena Subsite) and Willow Creek near Baxter Springs are major contributors of metals to the Spring River.

The mining-related contaminants in Short Creek have kept the numbers and types of aquatic organisms lower than in non-impacted streams. Point source discharges from an upstream industrial plant and nonpoint discharges from upstream mined areas in Missouri also add to the overall contaminant load in Short Creek. Persistent high concentrations of zinc and cadmium preclude the occurrence of many aquatic organisms. Ambient water quality criteria for protecting aquatic life are routinely exceeded.

Ambient water quality criteria are exceeded occasionally in Shoal Creek, but the biota are not impacted nearly as much as in Short Creek. Shoal Creek, however, is of particular concern with regard to environmental impacts because several state-listed endangered species live within this watershed.

POST-OUFS STUDIES

The crushing, milling, and flotation operations proposed in the 1988 OUFS for remediation of the surface mine wastes were assumed to be a simple, conventional sulfide flotation process using a portable mill facility. The portable mill facility would be taken to selected locations throughout the subsite to process the locally available mine wastes, thereby reducing the required haul distances and expenses associated with a centrally located mill.

In May 1988, the EPA authorized additional technical assistance to better determine process treatment parameters to remove and mill the mine wastes. A more detailed understanding of specific process variables was needed before the final remedy for the subsite was selected. The primary objectives of the additional work tasks were to collect samples of high- and low-grade mine wastes and then conduct metallurgical tests on these materials to better define design and operating parameters for the treatment process proposed in the preferred remedy. The specific unit processes studied included crushing, grinding, and flotation.

During the field sampling in June 1988, a portable X-ray fluorescence (XRF) spectrometer was used to semiquantitatively identify lead and zinc concentrations of mine waste samples. The XRF measured metals concentrations of randomly selected mine waste rock and chat piles. The resulting data indicated that many chat piles contained substantial lead and zinc concentrations. As a result, chat samples, as well as waste rock samples, were collected for testing. A backhoe was used to dig into several mine waste rock piles to get a better understanding of material size distribution and degree of the piles' homogeneity. Since these piles contained waste rock from very fine sand sized material up to rocks 24 inches in diameter, some material was screened to better estimate the finer sized fraction of the piles. Samples of these materials were collected and securely stored onsite in 55-gallon drums for potential future testing.

This work indicated that there is a wide size distribution of materials in the waste rock piles with correspondingly highly variable metals concentrations. Rough estimates indicate the waste rock piles contain about 40 to 50 percent material that is less than 2 inches in diameter.

The waste rock and chat samples were subsequently chemically analyzed. The chemical results confirmed the XRF work in the field. Wet screening and further chemical analyses on the chat samples showed that most of the lead was in the very fine-sized (minus 400 mesh or 37 microns) fraction of the chat. This fine-sized fraction includes the materials most likely to be ingested. Bench-scale tests were conducted on the various waste rock and chat samples to better define process requirements and metal recoveries.

The results of the metallurgical tests revealed that the milling/flotation process required for suitable metal (primarily lead, zinc, and cadmium) recoveries from both the waste rock and the chat would be more complex than originally envisioned (see Appendix A). For example, the waste rock was harder than expected, so the crushing and grinding circuits would be larger and more expensive to build and operate. To liberate the minerals from the rock matrix, grinding to about 100 percent minus 200 mesh (75 microns) was necessary rather than the nominal 100 mesh (150 microns) size originally estimated.

In addition, lead and zinc sulfides were previously assumed to be the most prevalent form of metals concentration because of the nature of the ore body. The tests determined, however, that quantities of oxide forms were present in both waste rock and chat and that the oxides have to be recovered as well as the sulfides to produce a satisfactory metals removal and tailing. Therefore, the new mill circuit has to be able to process the fines from the chat (which contains metal oxides) as well as the mine waste rock. Process flowsheets and preliminary cost estimates were developed to reflect the results of the bench-scale test work for the revised milling processes (see Appendix A).

Preliminary analytical work using the limited number of available chat samples indicated that further chat data were needed. In November 1988, surface and depth composite samples were collected from four large chat piles on the Galena Subsite. Chat sample analyses confirmed previous findings that a substantial portion of the lead and zinc is contained in the very fine fraction (minus 400 mesh) of the chat. Appendix B presents the findings of the November 1988 chat sample analyses and other chat data from Appendix A.

During the summer of 1988, site information developed by the PRPs was shared with the EPA. The PRPs subsequently conducted some column leach tests (with EPA oversight) on waste rock, chat, and a simulated mill process tailing to better understand the geochemical behavior of these wastes. Particularly significant was the PRPs' work to more accurately estimate volumes of the various mine wastes within the subsite's eight defined (by EPA) waste zones (see Figure 1-2). In addition, estimates of below-grade surficial void-space volumes (with and without a shallow-water table influence) were also determined. Using the now expanded data base, the PRPs proposed a remedial action for remediating the surface-deposited mine wastes at the subsite and submitted it to the EPA for consideration.

Pertinent data developed by the PRPs are summarized as follows:

- An inventory of mine waste rock, chat, and available void space in the subsite within the eight EPA zones (Andes, 1988a). The PRPs' work indicated there are about 550,000 cubic yards (yd^3) of waste rock (including associated contaminated soils). In addition, there are approximately 750,000 yd^3 of chat. Andes also estimated that there is surficial void space of about 1.9 million yd^3. Included with the PRP void volumes were esti-

mates of voids having water in them (about 0.8 million yd^3) versus those that held no water (about 1.1 million yd^3) (Andes, 1988b).

- Samples of mine waste rock and chat with corresponding analyses of certain metals contents. The PRP interpretation of these data indicates that the materials vary widely in metal concentrations (AMAX, 1988). These results correspond with previous sampling results by the EPA.

- Leach tests conducted in columns on samples of crushed waste rock, chat, and simulated tailings from mill processed waste rock. Deionized water at approximately 5.5 pH was used to both simulate percolation from major rainfall events and subject the wastes to saturated conditions to simulate the saturated groundwater zone. A technical review of the PRPs' leach test work is presented in Appendix C.

To better understand the effects of placing mine wastes containing higher concentrations of metals in saturated conditions, particularly in acidic mine water that would continue to move through the backfilled voids, the EPA conducted a series of additional leach (jar) tests. Samples of fine-sized waste rock previously screened from the waste rock piles were selected for the tests. The fine-sized material (less than 2-inch size) reacts faster in acid solutions and solubilizes more metals than coarser material containing equivalent metals because of a greater available surface area to volume ratio. These samples were then leached in beakers containing distilled water adjusted with acid to lower the pH to 4.5 to replicate the acidic water in the mine voids. Hydrochloric acid was used so the additional

generation of sulfate from the sulfide minerals could be measured in addition to the metals leached into solution (see Appendix D).

The potential hazards to public health were also further evaluated subsequent to completion of the 1988 OUFS. The principal contaminant of concern at the Cherokee County Site is lead. To establish an action level for lead, EPA and ATSDR guidance for the health risks associated with ingestion of soils (direct contact does not present a health risk) located in a residential setting were considered. Subsequently, an action level for lead was established for mine wastes and waste-contaminated soils at 1,000 mg/kg, or ppm, for the Galena Subsite.

Concerns regarding zinc concentrations have also been raised subsequent to the 1988 OUFS. The geochemical activity of zinc at the site is such that approximately 5,000 ppm in the mine wastes may be a level of concern. Zinc above this level of concern will leach to groundwater and may pose a risk to aquatic life.

Both lead and zinc have been characterized at the Galena Subsite using a portable X-ray fluorescence (XRF) spectrometer, as previously described. The XRF analyses closely compared to conventional laboratory analysis of chat for lead and zinc. Use of the XRF and its results have facilitated chat metals content characterization and provided a means to assess the level of concern presented by individual piles or areas.

Though similar health concerns exist at the site for cadmium, it is not easily detected by the XRF at the concentrations present onsite. Available data using the XRF and laboratory analysis indicate that zinc is associated with cadmium at an average ratio of approximately 220:1 for

example, at 5,000 ppm zinc, the approximate cadmium concentration would be 23 ppm. This relationship between zinc and cadmium supports using the XRF to further characterize mine wastes in the future and also confirms the estimated level of concern for zinc as a conservative level of concern for cadmium.

REPORT ORGANIZATION

In light of the expanded data base now available for the subsite, the EPA has prepared this Groundwater/Surface Water OUFS Supplement. In Section 1, the purpose of the report is presented along with the background and chronology of activities since the OUFS public comment period.

In Section 2, "Technology Identification and Alternative Development," Chapters 5, 6, and 7 of the 1988 OUFS are reviewed in conjunction with the increased knowledge of the mine wastes. Revised remedial alternatives have been developed. These are subsequently described in detail.

In Section 3, "Detailed Analysis," the alternatives developed and presented in Section 2 are analyzed in detail. In addition to following the most current RI/FS guidance for alternative evaluation, this document presents a comparative evaluation using the criteria that were in effect when the 1988 OUFS was completed. This presentation allows the revised alternatives' evaluations to be directly compared to the evaluations presented in the 1988 OUFS.

Reference materials for the 1988 OUFS Supplement are organized into the following appendixes:

A. Technical Assistance for the Galena Subsite Mine Waste Characterization Program

B. Chat and Sediment Data Characterization

C. Technical Review of PRP Leach Test Report

D. Mine Waste Fines Leach Test Results

E. Mass Load Analysis--Proposed Remedial Alternatives

F. Alternative Cost Estimates

Section 2
TECHNOLOGY IDENTIFICATION AND ALTERNATIVE DEVELOPMENT

REVIEW OF TECHNOLOGY IDENTIFICATION AND SCREENING

The technology identification and screening process for the GW/SW Operable Unit at the Galena Subsite was documented in Section 5 of the 1988 OUFS. During this process, general remedial response actions that satisfied the subsite's remedial objectives were identified. For each response action, technologies and various process options were identified, evaluated, and screened. The selected technologies and process options were then used to develop remedial alternatives (Chapter 6 of the 1988 OUFS).

Although the 1988 OUFS identified ingestion of surface deposited mine waste rock as a human health risk, protection of human health from exposure to metal-contaminated mine wastes was not formally identified as a remedial objective (EPA, 1988a). Selection of the February 1988 preferred remedy did, however, take into account the recognized health threat from ingesting the mine wastes. Table 2-1 presents the remedial objectives for the Groundwater/Surface Water Operable Unit as defined by this supplement to the 1988 OUFS.

As described in Section 1, additional field studies and test work were conducted after publication of the 1988 OUFS. The additional knowledge of the mine wastes (waste rock and chat) characteristics, as well as the more detailed volume estimates for the surface-deposited mine wastes, were considered during a review of the technology identification and alternative development activities documented in the OUFS. The review of the technology identification and screening process

Table 2-1
GROUNDWATER/SURFACE WATER OPERABLE UNIT
REMEDIAL OBJECTIVES

1. Protect the Roubidoux Aquifer from deep well contaminant inflows within the subsite.

2. Protect human health of the population within the subsite from mining related contaminants in the groundwater and surface water systems.

3. Protect human health of the population within the subsite from mining-related contaminants in the surface-deposited mine wastes.

4. Provide suitable drinking water (meet primary MCLs at existing taps) for the population within the subsite[a].

5. Improve water quality and/or reduce the volume of surface water entering the shallow groundwater system.

6. Reduce metal loadings in Short Creek, Shoal Creek, and Spring River to support sitewide goals.

7. Improve water quality of the shallow aquifer within the Galena Subsite.

[a]A suitable alternative drinking water supply for the subsite has been addressed by the Alternative Water Supply Operable Unit ROD.

determined that the technology of subsurface disposal of surface-deposited mine wastes (which present a threat to human health and the environment) should be retained through the alternative development process. Subsurface disposal of mine wastes was not considered during the identification of technologies in the 1988 OUFS other than for the tailing from a mine waste treatment (milling) facility. Additional knowledge of the geochemical characteristics and metals contents of the various size mine waste fractions, gained since completion of the OUFS, supports retaining subsurface disposal as a potential technology for alternative development.

Subsurface disposal technology would utilize the nearest adjacent abandoned mine workings, shafts, and open subsidence pits for placement of mine waste below existing grade. Surface-deposited mine waste (waste rock and/or chat) would be mined (removed), characterized with respect to metals content, and then disposed of in the nearby shafts and pits. The disposal areas would then be covered and recontoured with materials that have lead and cadmium concentrations below the health-based action levels to protect public health. That portion of the chat that does not contain concentrations that exceed the action level for lead or cadmium, as well as other locally available soils, could be used as cover material.

Subsurface disposal can be effective in minimizing the ingestion exposure pathway and the potential for human exposure if the disposal area is covered by materials with contaminant concentrations below the action level. In addition, the compacted backfill would reduce rainwater infiltration through the previously exposed mine wastes and through the previously opened subsidences, pits, and shafts. Open subsidences, pits, and shafts, as they currently exist, directly capture precipitation and runoff from nearby areas. By backfilling these open subsidences, pits, and shafts, the direct inflow of precipitation and runoff would be eliminated. Infiltration through the backfill would be minimized by compacting and contouring.

This technology could be implemented using conventional earthmoving equipment to dispose of the material. The issue of ground instability would have to be addressed during the design and implementation phases because of the threat to operator safety posed by the use of heavy equipment over the old mine workings.

As noted in the 1988 OUFS, many of the mine workings and subsidence pits are flooded. Some of the waters are acidic because of the mineralized areas oxidizing in the shallow groundwater system. If backfilling of mine wastes is to be considered, it will be important to characterize whether or not these wastes are significantly geochemically active and, therefore, potentially a major contaminant source. Adding highly geochemically active mine wastes (such as waste rock fines) to the shallow groundwater could potentially exacerbate long-term mass metal loadings (see Appendixes C and D). Although this technology will be retained for further consideration during review of alternative development, recognition of the concern to mitigate potential leaching of the backfilled materials must be maintained.

REVIEW OF ALTERNATIVE DEVELOPMENT

In the 1988 OUFS, 12 alternatives were developed using technologies and process options that had been retained for alternative development. Each alternative was then described in sufficient detail to support a preliminary screening based on effectiveness, implementability, and cost criteria. The screening determined that five alternatives would be subjected to detailed analysis (including the no-action alternative).

From the alternatives that were analyzed in detail in the 1988 OUFS, EPA developed a preferred remedy. The primary remedial action in the preferred remedy published in the February 1988 proposed plan is the mining and milling/ flotation of the surface-deposited mine waste to remove the lead, zinc, and cadmium and produce a tailing containing acceptable metals concentrations that would then be disposed of in the mine voids. Other remedial actions in the proposed plan include diverting and channeling selected

surface streams to minimize recharge to the shallow groundwater system; recontouring and regrading some areas to reduce rainwater infiltration; and inspecting and remediating (rehabilitating), if necessary, wells that penetrate to the deep aquifer.

A review of the alternative development, screening, and detailed analysis documented in the OUFS, coupled with the additional site knowledge obtained since the OUFS, has demonstrated that modifications to the proposed plan and new alternatives should be developed and evaluated.

Five alternatives have been developed for this OUFS Supplement. The numbers assigned to these five alternatives do not correspond with numbers assigned to the alternatives presented in the 1988 OUFS.

Alternative 1 in this OUFS Supplement is the no-action alternative. As required by the National Contingency Plan, NCP, the no-action alternative was retained and re-evaluated for comparative purposes with the other alternatives evaluated herein.

Alternative 2, presented in this OUFS Supplement, will process mine waste rock and chat and represents the February 1988 EPA-preferred remedy now modified by considering the additional information regarding the physical and processing characteristics of the mine waste rock and chat. New data indicate that the chat contains lead concentrations that generally are in the very fine particle size ranges and that the lead cannot be recovered by conventional sulfide flotation. This alternative still provides a crushing, grinding, and sulfide flotation system for the mine waste rock, but also considers treating all of the chat through a wet screening process that separates the highly contaminated fraction of the chat (fines) from the coarse fractions. A process

circuit would be added to the flotation system for treating the fine size chat fraction and the oxide materials not recovered in the sulfide flotation circuits. Saleable metal concentrates would be produced and possibly be marketed from the flotation process.

Alternative 3 herein is a modification of Alternative 2. Alternative 3 recognizes that only a portion of the chat may be sufficiently contaminated to justify wet screening and processing. Before removal, the chat would be characterized to determine if it exceeds the action level. Only the chat exceeding the action level would be processed. The remaining chat could be used for recontouring to minimize infiltration and surface water capture.

Alternative 4 herein uses an equivalent RCRA-designed (double containment, double leachate collection system) containment unit for disposing of all surface-deposited mine wastes. This alternative represents the waste containment alternative (No. 5) developed in the 1988 OUFS.

Alternative 5 herein employs subsurface disposal of surface-deposited mine waste (waste rock and chat). Physical sizing of the waste rock (at a nominal 2 inches in diameter) would separate the geochemically active waste rock fines for disposal above the shallow groundwater table.

All alternatives (except the no-action alternative) include the deep aquifer (Roubidoux) well inspection/remediation and stream channelization and diversion actions that were originally included in the preferred remedy. Appropriate recontouring and revegetation of mining-related disturbed and subsidence areas are also included in each alternative. A detailed description of each alternative is presented below. These alternatives are analyzed with CERCLA guidance evaluation criteria (detailed analysis) in Section 3 of this report.

DESCRIPTION OF ALTERNATIVES

ALTERNATIVE 1--NO ACTION

The site would remain as it is with no remedial actions implemented. Since contaminants would remain onsite, a review of site conditions every 5 years must be conducted in accordance with the provisions of CERCLA. The no-action alternative provides a baseline for comparing other alternatives.

ALTERNATIVE 2--MINE AND MILL ALL MINE WASTES

The primary remedial process of this alternative is to remove and treat the surface-deposited mine waste rock and chat to essentially eliminate human exposure to metal contaminants in the mine wastes and reduce groundwater and surface water metals loadings. Figure 2-1 presents a block diagram showing the various components of this alternative.

Waste rock and chat would be removed and hauled to centrally located stockpiles adjacent to the processing facility. Waste rock from its stockpile would be processed using crushing, grinding, and sulfide/oxide flotation technologies to remove metals. Chat would be withdrawn from its stockpile and wet-screened to separate the highly contaminated fines from the relatively low metals content coarse fraction before grinding. The chat fines would then be processed in a portion of the milling/flotation facility with the waste rock. The flotation tailing material product would be disposed of in adjacent mine workings. A more detailed description of each remedial action is presented below.

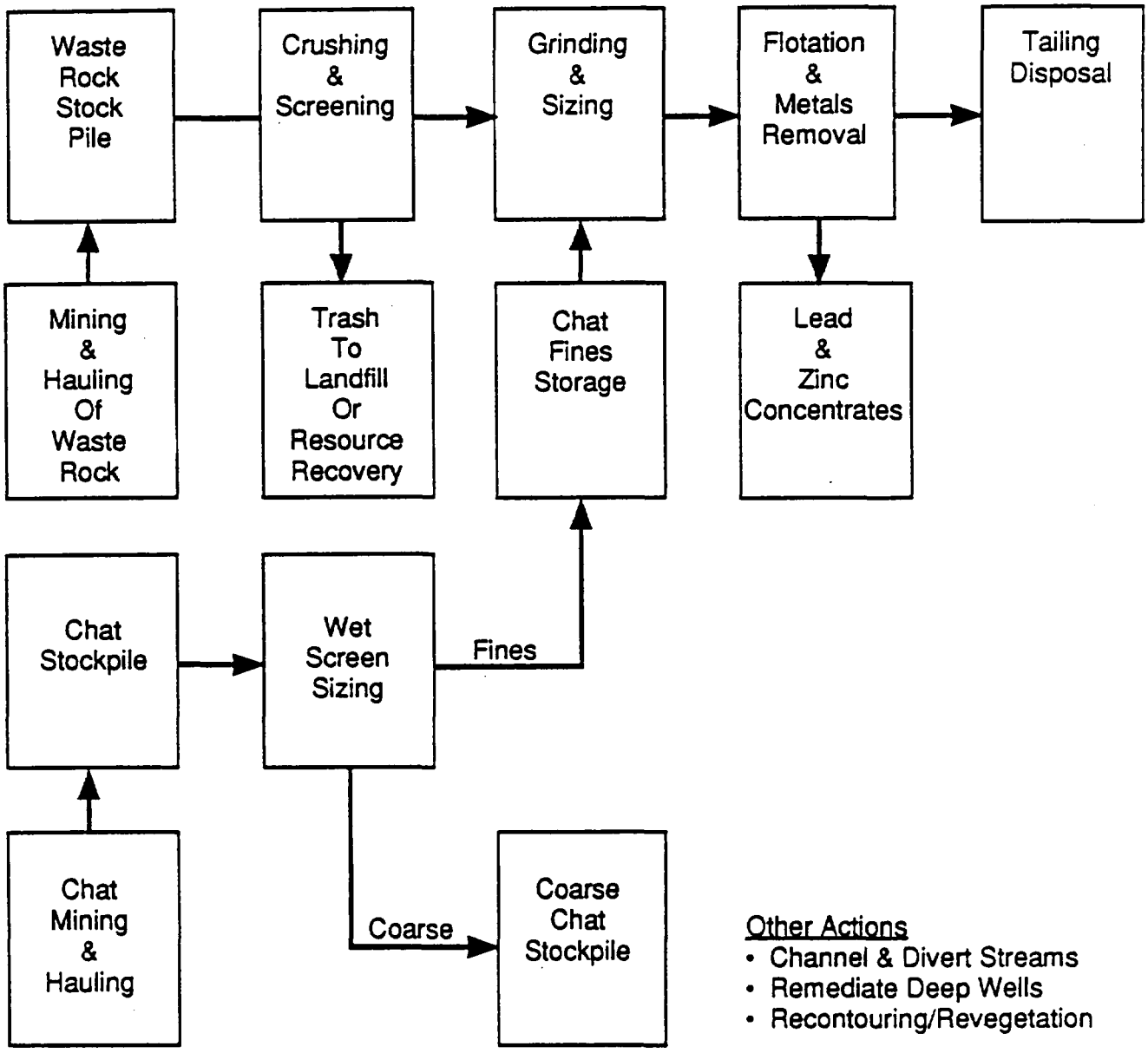


FIGURE 2-1
ALTERNATIVE 2:
MINE AND MILL
ALL MINE WASTES
CHEROKEE COUNTY, KANSAS
GALENA SUBSITE
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Mining and Milling Waste Rock

Mining technologies would be used to remove the surface-deposited waste rock. Mining sequences would be planned separately for each general area where mine waste rock is located. In most areas, waste rock could be transferred directly to haul trucks using backhoes or loaders. In areas like Hell's Half Acre (HHA), mining methods would have to be modified because of the unstable subsurface conditions and potential for subsidence or collapse from surface loading. The mining methods would, therefore, be less efficient for some areas.

Large pieces of trash mixed with the waste rock would be removed and set aside for sanitary landfill disposal or resource recovery at each location that was mined. The estimated nominal 662,600 tons of waste rock would be hauled over 2-1/2 years (with 10-yd^3 capacity highway trucks) to the mill to be located adjacent to the HHA area. This location was chosen because of the large amount of mine void space in HHA for tailing disposal and the relatively central location to a large amount of the mine wastes. The tailing product from the crushing, grinding, and flotation operations would be disposed of in the adjacent mine voids.

Mine waste rock that was stockpiled (adjacent to the processing facility) would be reclaimed and fed to the crushing and screening plant for size reduction. Equipment would consist of a jaw crusher, cone crusher, and dry screening plant. Some miscellaneous trash, such as plastic and metal parts, would be removed manually in the process circuit and disposed of before material enters the cone crusher. Sized material (minus 5/8 inch) from the crusher plant would go to fine ore (waste rock) storage bins via a conveyor system. Oversized material would be recycled through the crushing plant until it is reduced to minus 5/8 inch.

Fine ore from the storage bins would be fed to the wet grind ball mill for further size reduction. In addition, chat fines from the chat processing circuit would be added to the ball mill for further processing. The ball mill, in closed circuit with a hydrocyclone, produces the correct-sized feed material for the flotation circuits. The hydrocyclone is used to control the product grind size of the flotation circuits feed slurry. The final product size from the ball mill is approximately 85 to 90 percent minus 200 mesh (75 microns). This slurry would then be pumped to the flotation circuits for lead and zinc removal. The zinc removal steps also remove cadmium, which is associated mineralogically with the zinc sulfides.

The flotation system is separated into several different circuits: the lead and zinc sulfide circuit and a three-stage metal oxide circuit. The sized material from the hydrocyclone discharges to a conditioning tank where flotation reagents are added to both the individual lead and zinc sulfide flotation steps. The slurry then flows to the flotation cell banks where the lead and zinc sulfides are removed. The tailing material from the sulfide circuits flow to the three-stage oxide conditioning circuit where new reagents are added for oxide flotation. The slurry then flows to the oxide flotation circuit where the lead and zinc oxides are removed simultaneously. The lead sulfide concentrate, zinc sulfide concentrate, and the mixed lead and zinc oxide concentrates are collected, cleaned, filtered, and drummed.

The lead and zinc sulfide concentrates should be of saleable quality. Oxide concentrate marketablity is unknown.

The tailing material would be thickened from about 25 percent solids to about 60 percent solids. This thickened slurry

would then be pumped to and disposed of in the mine voids in the HHA area. Since hydraulic transport and deposition of the tailings would be used, whether or not the mine voids or pits contain water should not impact tailing disposal operations (i.e., water displacement would not impede tailing disposal).

Mining and Screening of Chat

All chat (estimated at about 750,000 yd^3, which is equivalent to about 1 million tons) would be mined and hauled over the 2-1/2-year operating period to a chat feed stockpile adjacent to the central mill plant for treatment. Chat from the stockpile would be fed directly to a wet screening process circuit where the minus 35 mesh fraction would be wet screened from the coarser fraction. Based on EPA chat sampling and analyses, the minus 35 mesh fraction (about 20 percent of the total) contains the bulk of the metal contamination (50 to 75 percent based on sample analyses). The coarse fraction from the screening circuit (with concentrations of lead at less than the action level) would be dewatered and conveyed to a stockpile for storage, sale, or subsequent use, such as a recontouring material. The minus 35 mesh fraction chat would be fed into the mine waste ball milling circuit to be processed with the crushed waste rock. The lead, zinc, and cadmium in the chat would then be removed in the flotation circuits as previously described. The concentrates and tailing product would be disposed of as mentioned earlier.

Recontouring/Revegetation

According to the PRP investigations (Andes, 1988a), there are approximately 710 acres of disturbed areas consisting of surface mine waste deposits and open mine voids within the eight EPA-defined waste zones. For Alternative 2, where all the mine wastes are removed for treatment, recontouring of

the appropriate surfaces in the mine waste zones was included as part of the "mining" or removal of the wastes.

For revegetation, the acreage subject to revegetation remediation can be up to 710 acres less than the amount of open mine voids still remaining after implementing the specific alternative. For Alternative 2, where only the mine voids in HHA will be filled with tailings, half of the 710 acres (360 acres) is estimated to be suitable for revegetation. This activity includes adding available topsoil, soil amendments, and reseeding at the EPA-estimated cost of $1,000 per acre.

Channelization

Surface drainages upgradient of two of the more heavily mined areas (HHA and Owl Branch) would be collected and diverted around these disturbed areas to prevent surface stream capture by mine shafts and subsidences (Figure 2-2). At the Blue Hole, local drainages would be diverted from the shafts and subsidences and the Owl Branch tributary would be placed in a concrete-lined channel. The streambed through HHA (named Tributary A in the 1988 OUFS) would be re-established with a concrete-lined channel to eliminate the present stream capture. The 1988 OUFS included an action to divert lower Owl Branch to Tributary C, which drains directly to Spring River rather than Short Creek. This diversion, it was assumed, would substantially reduce the surface water drainage in the lower Owl Branch area. Subsequent field observations indicate that such a diversion would not substantially reduce the lower Owl Branch drainage. The cost of the extensive excavation required to make the diversion would not support the minimal benefit of slightly reducing the overall water flow through lower Owl Branch. Therefore, the lower Owl Branch diversion is not part of this remedy.

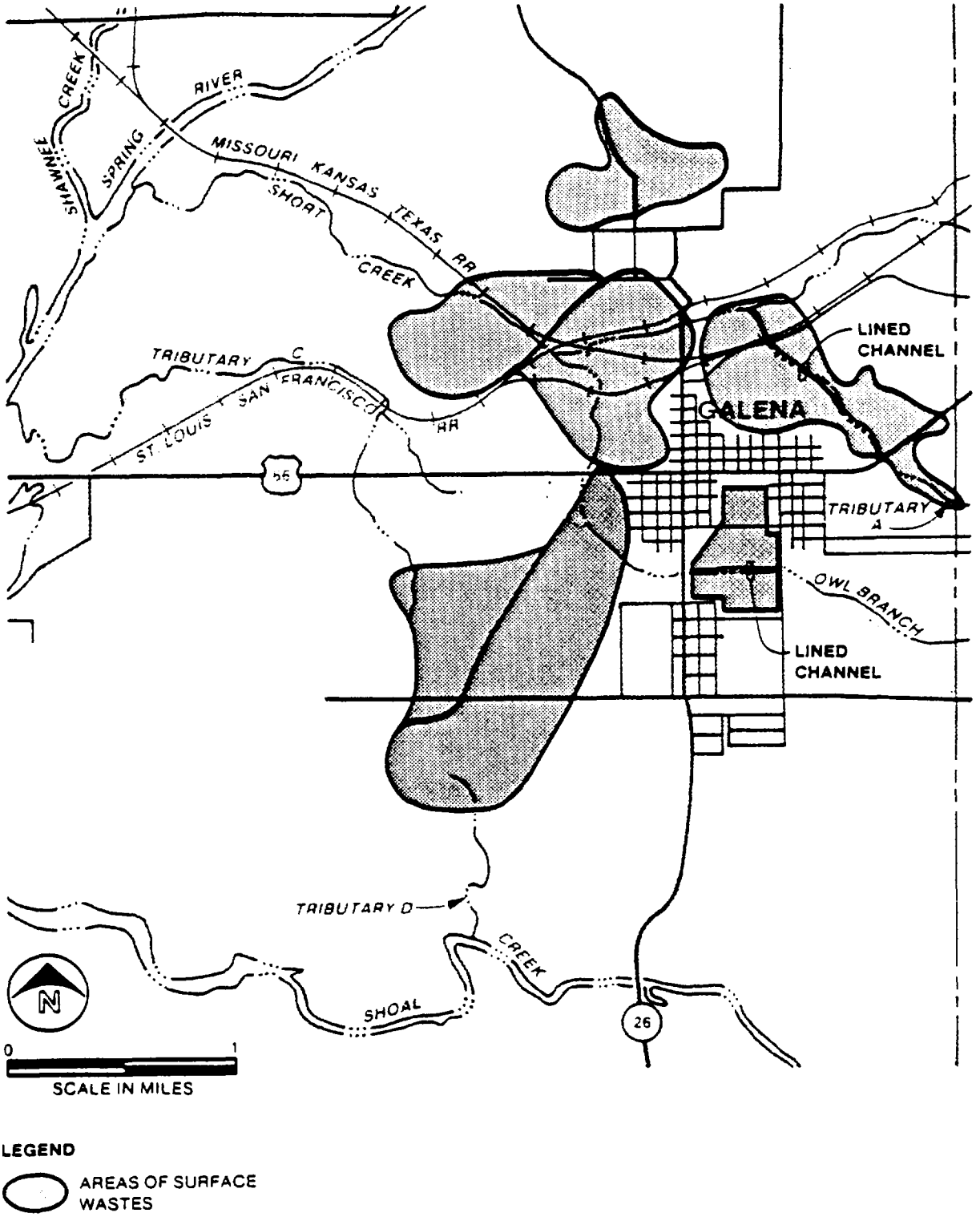

FIGURE 2-2
SURFACE WATER RECHANNELIZATION
CHEROKEE COUNTY, KANSAS
GALENA SUBSITE — OUFS
GROUNDWATER/SURFACE WATER

Deep Well Remediation

Deep aquifer well remedial actions would be taken to protect the Roubidoux formation groundwater from contaminant migration from the contaminated shallow aquifer. Remediation would consist of plugging all abandoned Roubidoux wells that can be located and inspecting the wells in use. Only those wells with structural defects found during inspection would be repaired. This remediation work would only be done on wells identified as extending to the Roubidoux formation. Based on the current knowledge of the Galena Subsite, three wells are estimated to require repairs and one well would be plugged.

Long-Term Operations and Maintenance

The surface water diversion channels would have to be maintained during their effective life. Maintenance activities will include annual inspections, cleaning, and repair of the concrete-lined channels. To monitor the effectiveness of the remedy and also provide data for the required site review every 5 years, groundwater and surface water would be sampled and monitored periodically.

ALTERNATIVE 3--MINE AND MILL ALL MINE WASTE ROCK AND HALF OF THE CHAT

The principal remedial processes for this alternative are the same as for Alternative 2: remove and treat mine wastes to essentially eliminate human exposure to metal contaminants in the mine wastes and reduce groundwater and surface water metal loadings. However, selective removal of only the chat with lead concentrations above the action level (as determined with XRF characterization techniques) would be performed to reduce the amount of chat requiring removal and

treatment. For cost estimating and evaluation purposes, this study assumes 50 percent of the chat (500,000 tons) would have to be removed and treated. Figure 2-3 presents a block diagram depicting the various components of this alternative.

Mining and Milling of Waste Rock

Waste rock would be mined, processed, and disposed as described in Alternative 2.

Characterization and Screening of Chat

Alternative 2 assumes that all of the chat contains contaminated fines exceeding the action level and must be moved to the processing facilities. In Alternative 3, a portable XRF spectrometer would be used to field characterize (in situ) the approximately 1 million tons of chat. The sampling, preparation, and characterization of these materials are important to determine if the materials should be removed, treated, and properly disposed of, or may remain in place without an unacceptable threat to public health or the environment.

These wastes can contain larger particles that are unlikely to be ingested, such as rocks and coarse grits. Therefore, a methodology to consider this factor was developed in conjunction with a quick analytical technique to field measure lead concentrations for the size fraction that could be ingested to make good decisions on whether these soils and mine wastes need to be removed. Representative bulk samples from the waste areas and adjacent soil areas would be collected, composited to represent the area, and properly split to about 5 kg. The saved composite sample splits will then be dried at 105°C for 2 hours until dry. The materials will then be dry sieved at 16 mesh (1.18 millimeters [mm]).

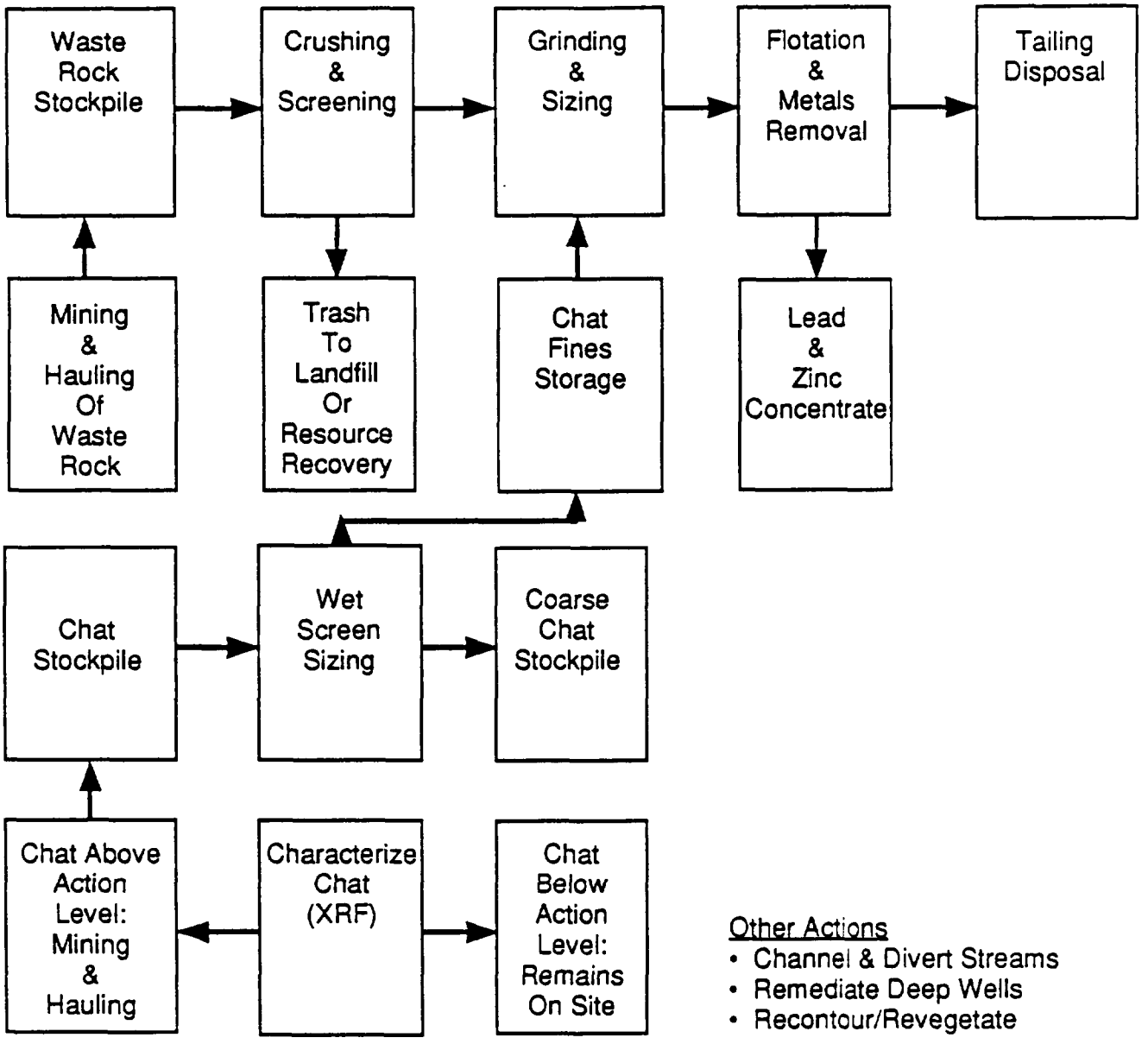

FIGURE 2-3
ALTERNATIVE 3:
MINE AND MILL ALL MINE WASTE ROCK AND HALF OF CHAT
CHEROKEE COUNTY, KANSAS
GALENA SUBSITE
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The undersized materials will be repeatedly split to two nominal 100-gram samples, one of which would be retained in a sample archive, the other used for analytical purposes. The analytical sample would be ground fine in a mortar and pestle, blended, and then analyzed with a field-portable, calibrated XRF spectrometer. Previous work has indicated that the device has a detection limit of 200 ppm lead with an accuracy of ±50 ppm. Measuring materials that are less than or greater than the action level should be straightforward.

Chat piles, or portions thereof characterized as having lead above the action level, would be mined, hauled, and screened for grinding and flotation as described in Alternative 2. Chat characterized as having lead below the action level would be left in the disturbed areas for use during the limited recontouring effort.

Recontouring/Revegetation

Disturbed areas would be recontoured during the mine waste removal actions to minimize closed basins and low spots as described for Alternative 2. In Alternatives 3, however, chat passing the lead action level would be used to aid in recontouring. Following recontouring, revegetation would be implemented. For this alternative, the estimated acreage suitable for revegetation would be about 400 acres (using the chat to fill some pits). The methodology of revegetation would be identical to that described for Alternative 2.

Channelization

Stream channelization would be implemented as described for Alternative 2. Surface drainages upgradient of two of the

more heavily mined areas would be collected and diverted around these disturbed areas to prevent surface stream capture by mine shafts and subsidences.

Deep Well Remediation

Deep-aquifer well remedial actions, as described for Alternative 2, would be taken to protect the Roubidoux formation groundwater from contaminant migration from the contaminated shallow aquifer.

Long-Term Operations and Maintenance

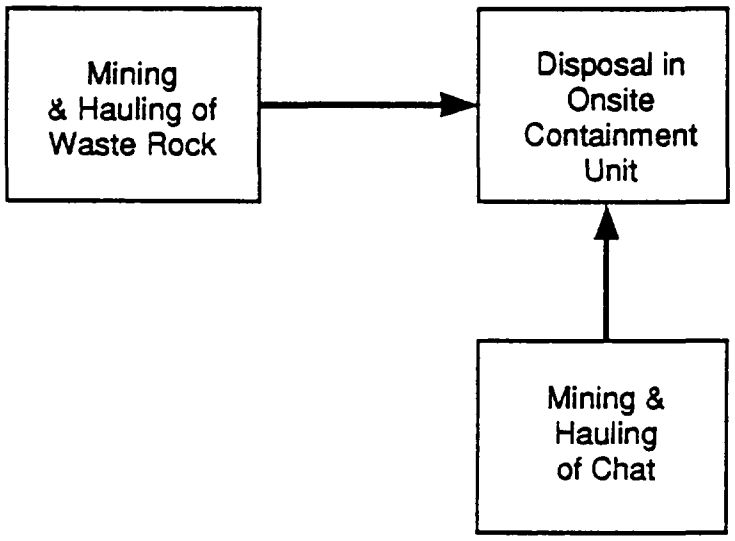
Long-term operations and maintenance activities would be essentially the same as described for Alternative 2.

ALTERNATIVE 4--MINE AND DISPOSE OF ALL MINE WASTES IN AN ONSITE CONTAINMENT FACILITY

The primary process of this alternative is to use a containment facility to essentially eliminate the human exposure (ingestion/inhalation) pathway from metal contaminants in the mine wastes and reduce groundwater and surface water metal loadings. All mine waste rock and chat would be mined, hauled to, and disposed of in an onsite containment unit. Figure 2-4 presents a block diagram depicting the various components of Alternative 4. This alternative is essentially the same as Alternative 5 presented in the 1988 OUFS. It is retained for analysis in order to present a range of alternatives for comparison.

Mining and Containing Waste Rock and Chat

All waste rock (about 662,600 tons) and all chat (about 1 million tons) would be mined and transported to a single



Other Actions
- Channel & Divert Streams
- Remediate Deep Wells
- Recontour/Revegetate

FIGURE 2-4
ALTERNATIVE 4:
MINE AND DISPOSE OF ALL WASTE ROCK & CHAT
CHEROKEE COUNTY, KANSAS
GALENA SUBSITE
DEN65541.AE

containment unit. For cost-estimating purposes, the containment unit was assumed to be sited near the center of the disturbed areas within the subsite to minimize mine waste haul distances (average haul distance about 1.2 miles). The containment unit must be sited on stable ground (not subject to mine subsidence). The site selected for the containment unit (for the purpose of alternative development) is located about 1 mile west of Galena in an area that has not been widely disturbed by mining.

The containment unit would be designed to meet RCRA design criteria for hazardous wastes. The unit would be constructed with a double-liner system and a double leachate collection system to contain the estimated 1,662,600 tons of waste. Assuming a containment unit working depth of 30 feet, approximately 20 acres would be required for containment unit construction.

Recontouring/Revegetation

Recontouring and revegetation methods and elements would be identical to Alternative 2. About 360 acres would be suitable for revegetation.

Channelization

Stream channelization would be implemented as described for Alternative 2.

Deep Well Remediation

Deep-aquifer well remedial actions would be implemented as described for Alternative 2.

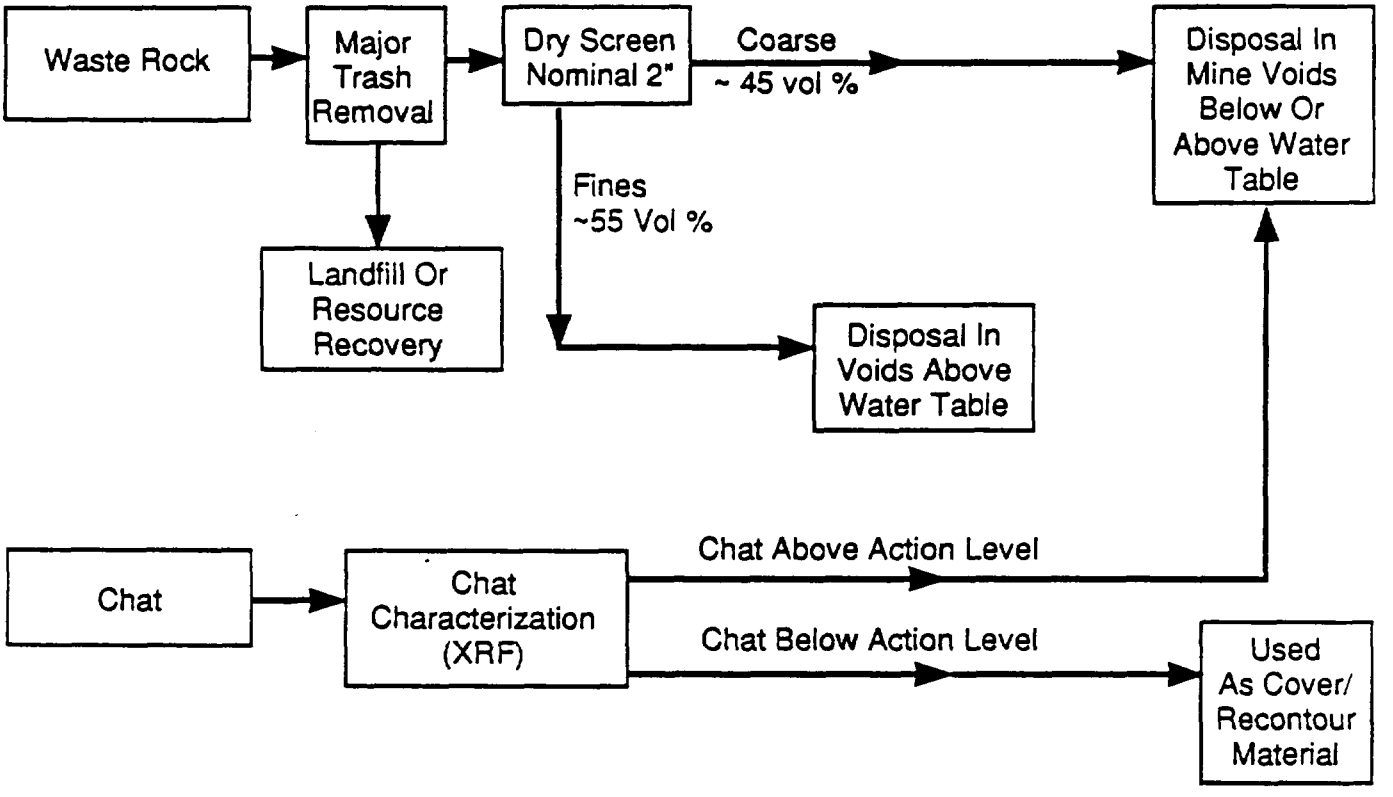
Long-Term Operations and Maintenance

In addition to the long-term operation and maintenance activities described for Alternative 2, this alternative would require long-term operation and maintenance activities for the containment unit during the effective life of the facility. These activities would include collection, monitoring, and disposal of the leachate; surface erosion protection maintenance; and environmental monitoring (perimeter wells). Regular inspections would be conducted to ensure erosion does not threaten the structural integrity of the unit.

ALTERNATIVE 5--GEOCHEMICALLY CHARACTERIZE WASTES, SEGREGATE BY PARTICLE SIZE, SELECTIVELY BACKFILL, AND RECONTOUR

The primary remedial process of Alternative 5 is to mine, characterize, and selectively dispose of surface-deposited mine wastes (waste rock and chat) in open subsidences, pits, and shafts. This action would essentially eliminate human exposure to contaminated mine wastes and reduce long-term groundwater and surface water metal loadings. Figure 2-5 presents a block diagram depicting the various components of this alternative.

Waste rock would be mined and then separated into two types of geochemically active size fractions using a portable screening circuit to physically separate coarser materials (plus 2 inches) from finer-sized materials. Based on testwork, the finer-sized materials are far more geochemically active than the coarser materials. This is due to the much greater surface area of the finer-sized fraction as well as the higher mineral content. The coarser-sized fraction would be disposed of belowgrade in nearby open pits, subsidences, or shafts and below the



Other Actions
- Channel & Divert Streams
- Remediate Deep Wells
- Recontour/Revegetation

FIGURE 2-5
ALTERNATIVE 5
GEOCHEMICALLY CHARACTERIZE WASTES, SEGREGATE BY PARTICLE SIZE, SELECTIVELY BACKFILL AND RECONTOUR
CHEROKEE COUNTY, KANSAS
GALENA SUBSITE
DEN65541.AE

water table. The finer-sized fraction of the waste rock would also be disposed of belowgrade, but above the area influenced by groundwater so as not to leach in the potentially acidic groundwater, thereby worsening its quality.

Chat would be characterized with the field-portable XRF spectrometer as mentioned previously for Alternative 3. Chat with lead concentrations above the action level and lower zinc concentrations (less than 5,000 ppm) would be disposed of belowgrade with the waste rock to fill coarse mine waste void space to minimize settling and to minimize zinc leaching to the groundwater. Chat with higher zinc and, hence, cadmium levels would be disposed of with waste rock fines (above the groundwater). Chat with lead concentrations below the action level and cadmium concentrations below any level of concern would be used as a cover material for the filled pits, subsidences, and shafts, or used to supplement recontouring. Figure 2-6 depicts a typical cross section showing the backfill disposal of the waste rock and chat. The placement of the various types of mine wastes is summarized in Figure 2-7.

Mining, Screening, and Disposal of Waste Rock

Within a given disturbed zone, waste rock would be mined, transferred to a nearby screening plant, and then dry screened at a nominal 2-inch size. This size was chosen since dry screening at this size is very fast and efficient, and some moisture is easily tolerated. Effective screening at smaller sizes becomes more difficult. Field observations indicate that about 45 volume percent (about 250,000 yd^3) of the waste rock is plus 2-inch. Tests indicate (see Appendix D) that the minus 2-inch (finer) fraction of waste rock will be highly reactive with acidic mine waters because of the greater available surface area associated with a volume of finer-sized waste rock and the fine-sized mineralization

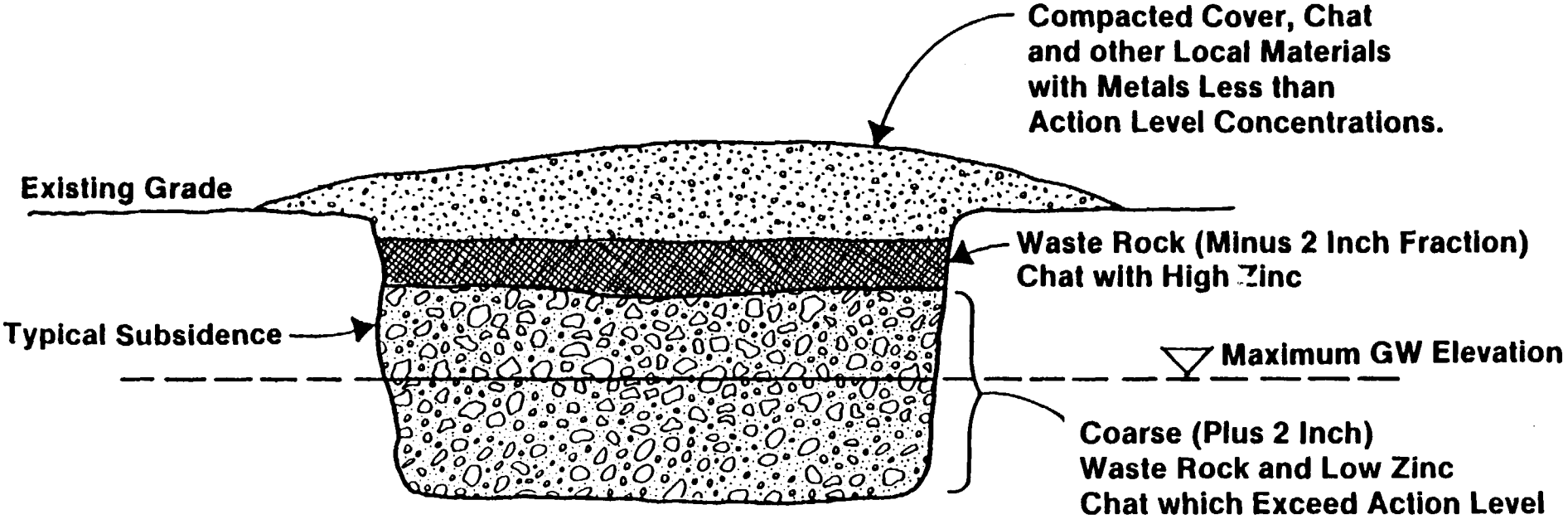


FIGURE 2-6
ALTERNATIVE 5 MINE WASTE BACKFILL
CHEROKEE COUNTY, KANSAS
GALENA SUBSITE — OUFS
GROUNDWATER/SURFACE WATER

WASTE TYPE / PREFERRED DISPOSAL LOCATION		WASTE ROCK		CHAT			
		+2"	-2"	HiPb HiZn	HiPb LoZn	LoPb HiZn	LoPb LoZn
ABOVE GRADE							X
BELOW-GRADE	ABOVE GW	X	X	X	X	X	
	BELOW GW	X			X [2]		

NOTES

[1] GW = Groundwater

[2] Preferred placement of + 2" waste rock and high lead/low zinc chat is below grade/above the groundwater table.

FIGURE 2-7
MINE WASTE DISPOSAL FOR ALTERNATIVE 5
CHEROKEE COUNTY, KANSAS
GALENA SUBSITE
GROUNDWATER/SURFACE WATER OUFS

present compared to the coarser material. The coarser material would be transported, backfilled, and compacted in pre-inventoried mine void/subsidences. Preferably, all wastes should be placed above the water table. However, if the coarse waste rock is placed in acidic waters within the void spaces, the leaching of metals would be many times less than if the fines were placed in that saturated environment.

The fine-sized mine waste rock would also be placed in subsidences, but above the seasonal-high (water table) saturated groundwater zone. Subsequently, chat characterized with the XRF as having lead concentrations below the action level would be used as a cover material to reduce infiltration. Field engineering studies to balance cut and fill quantities of characterized chat and waste rock and to determine wet and dry subsurface void space will be required to minimize handling and haulage of the mine waste types. The waste rock screening plant would be relocated from area to area as necessary to avoid excessive waste rock haul distances for processing and disposal.

Characterization and Disposal of Chat

Chat would be characterized with the XRF spectrometer to determine lead and zinc concentrations using the procedure specified earlier. The chat classified above the action level for lead and with lower zinc concentrations (below 5,000 ppm) would be used as supplemental fill with the coarse waste rock material disposed of below grade. Partial mixing of the chat with the coarse waste rock would fill the void space created by the removal of the finer-sized rock. This could minimize settling and, thus, reduce maintenance of the cover material. Additional engineering design would be required to finalize the fill composition. The chat lead content is not a controlling factor for placement belowgrade since the onsite chat has been highly processed rendering

the remaining lead content geochemically less active in the acidic groundwater environment. However, zinc at levels above 5,000 ppm will not be placed below the water table due to its geochemically active nature. Chat having these concentrations of lead and zinc are not heavily mineralized and, therefore, not highly geochemically active in an acidic groundwater environment compared to the mine waste rock fines having lead and zinc concentrations up to 5,000 ppm and 30,000 ppm, respectively (see Appendixes C and D). Chat with higher zinc concentrations would be disposed of belowgrade with the waste rock fines to remove the threat of cadium ingestion.

Chat below the action level for lead and with zinc less than 5,000 ppm would be used as a cover material. A compacted chat cover would reduce infiltration into the filled subsidences.

Recontouring/Revegetation

Disturbed areas would be recontoured during the mine waste removal actions to eliminate closed basins and low spots. As described previously, this would be done to minimize ponding and redirect surface runoff away from mine shafts and permeable areas. The chat characterization action of this remedy would also identify chat piles acceptable for recontouring purposes. Following recontouring, limited revegetation would be implemented over up to approximately 600 acres. It is estimated that with the backfilling, about 80 percent of the 710 disturbed acres (570 acres) would be suitable for revegetation.

Channelization

Steam channelization would be implemented as described for Alternative 2.

Deep Well Remediation

Deep-aquifer well remedial actions would be implemented as described for Alternative 2.

Long-term Operations and Maintenance

In addition to the long-term operation and maintenance for channelization described in Alternative 2, the compacted chat backfill, used to cover the mine wastes that have been disposed of in the mine subsidence, pits, and shafts, would have to be inspected routinely for erosion and subsidence problems. Additional backfill may have to be placed to maintain design grades.

ALTERNATIVE DEVELOPMENT SUMMARY

Table 2-2 presents a summary of the five alternatives developed for this OUFS. The appropriate technology and contaminated media is identified for each alternative. The table provides a means for comparing the alternatives to each other.

TABLE 2-2
DEVELOPMENT OF ALTERNATIVES

Action/Technology	Media	Alternative 1	2	3	4	5
No Action		X				
Removal of Surface Mine Wastes-Mining/Hauling	Waste Rock		X	X	X	X
	Chat (All)		X		X	
	Chat (Above Action Level)			X		X
Treatment-Sizing	Waste Rock					X
	Chat		X	X		
Treatment - Crush and Grind/ Flotation	Waste Rock		X	X		
	Chat (Sized)		X			
	Chat (Above Action Level)			X		
Disposal-Subsurface	Waste Rock					
	Waste Rock (Sized)					X
	Chat (All)					
	Chat (Above Action Level)					X
	Flotation Tailing Product		X	X		
Disposal-Containment Unit	Waste Rock & Chat				X	
Remediate Deep Wells	Groundwater	X	X	X	X	X
Divert Streams	Surface Water	X	X	X	X	X
Recontour/Revegetation	————	X	X	X	X	X

Section 3
DETAILED ANALYSIS OF ALTERNATIVES

APPROACH TO DETAILED ANALYSIS

The detailed analysis of the selected remedial alternatives assesses each alternative against specific criteria. The intent of this assessment is to provide sufficient relevant information for each alternative to allow selection of a preferred remedy for the Galena Subsite Groundwater/Surface Water Operable Unit.

The Guidance for Conducting Remedial Investigations and Feasibility Studies under CERCLA (EPA, 1988b) specifies nine individual assessment criteria. These criteria address specific statutory requirements of CERCLA, as modified by SARA, as well as other technical and policy considerations that have been proven essential for the remedy selection process. Each alternative that has been selected for detailed analysis has been defined to allow an adequate assessment against the criteria. Those elements of the alternatives that are common to all the action alternatives were evaluated against the criteria separately to avoid excessive repetition in the evaluation narrative and summary tables. Each alternative is then individually evaluated against the assessment criteria. An assessment of the no-action alternative is also presented to provide a comparative basis for the other alternatives.

Following the individual assessment, a comparative assessment of the alternatives is presented. This comparison among the alternatives is made to assess relative differences with respect to each criteria. The comparative assessment is presented in tabular form. An assessment for

each of the specific criteria and associated subcriteria is made for each alternative.

In addition to the table presented for comparative analysis, a second table is presented. In Chapter 8 of the 1988 OUFS, the detailed analysis used the criteria in effect at that time (EPA, 1987b). The most current EPA guidance (EPA, 1988b), however, provides the most current and a slightly different interpretation of the nine criteria. To be consistent with current guidance, yet provide a means for direct comparison to the detailed analysis presented in the 1988 OUFS, a second summary table has been developed. This format allows a direct comparison of the detailed analysis conducted for the alternatives in this document and the detailed analysis in the 1988 OUFS.

ALTERNATIVE EVALUATION CRITERIA

The nine assessment criteria and associated subcriteria, as presented in the most recent EPA guidance (EPA, 1988b), are shown in Table 3-1. The detailed analysis presented in this document addresses the first seven criteria in the table. Evaluation of the state and community acceptance (the last two criteria) will be conducted following the public comment period for this supplement to the OUFS. The compliance with ARAR's criteria evaluation refers to the potential ARARs discussed in Section 4 of the 1988 OUFS.

QUALIFICATIONS USED IN THE DEVELOPMENT OF DETAILED ANALYSIS

The following assumptions and qualifications were developed for the detailed analysis of alternatives. The preferred

Table 3-1
CRITERIA FOR DETAILED ANALYSIS OF ALTERNATIVES

OVERALL PROTECTION OF HUMAN HEALTH AND THE ENVIRONMENT

- How alternative provides human health and environmental protection

COMPLIANCE WITH ARAR'S

- Compliance with chemical-specific ARARs
- Compliance with action-specific ARARs
- Compliance with location-specific ARARs
- Compliance with other criteria, advisories, and guidances

LONG-TERM EFFECTIVENESS AND PERMANENCE

- Magnitude of residual risk
- Adequacy and reliability of controls

REDUCTION OF TOXICITY MOBILITY AND VOLUME THROUGH TREATMENT

- Treatment process used and materials treated
- Amount of hazardous materials destroyed or treated
- Degree of expected reductions in toxicity, mobility, and volume
- Degree to which treatment is irreversible
- Type and quantity of residuals remaining after treatment
- Statutory preference for treatment

SHORT-TERM EFFECTIVENESS

- Protection of community during remedial actions
- Protection of workers during remedial actions
- Environmental impacts
- Time until remedial action objectives are achieved

IMPLEMENTABILITY

- Ability to construct and operate the technology
- Reliability of the technology
- Ease of undertaking additional remedial actions, if necessary
- Ability to monitor effectiveness of remedy
- Ability to obtain approvals from other agencies
- Coordination with other agencies
- Availability of offsite treatment, storage, and disposal services and capacity
- Availability of necessary equipment and specialists
- Availability of prospective technologies

COST

- Capital costs
- Operating and maintenance costs
- Present worth cost

STATE ACCEPTANCE[a]

COMMUNITY ACCEPTANCE[a]

[a]These criteria are assessed following the public comment period.

remedy selection process must consider these qualifications/ assumptions and the potential effects they may have to the selected remedy's effectiveness, implementability, and cost.

- An action level for lead was established for mine wastes and waste-contaminated soils at 1,000 ppm for the Galena subsite. This action level is based on relevant EPA and ATSDR guidance for the health risk for ingestion of materials. For the backfill option, mine wastes containing zinc higher than 5,000 ppm are considered to have a potentially adverse impact if left on the surface due to related cadmium concerns or if backfilled below the water table due to possibly deteriorating water quality. The XRF will be used for estimating the concentration of lead and zinc in chat and mine waste contaminated soils. Cadmium concentrations can be estimated from the measured zinc concentrations.

- Sizing treatment process systems and estimating associated costs depend on waste types, quantities, and locations. All mine waste volume estimates were based on information provided by the PRPs since their mine waste volume data base is more extensive than EPA's. The PRPs conducted relatively extensive field investigations in identifying mine waste types and estimating mine waste volumes at the Galena Subsite. If actual waste types, tonnages, and locations are substantially different than currently estimated, major changes in costs are possible.

- For all alternatives, it has been assumed that access to mine wastes (both waste rock and chat)

for removal, transport, and/or disposal can be obtained by EPA. No costs were allocated to purchase mine waste rock or chat.

- Possible revegetation of up to 710 acres of disturbed area was included as part of each alternative. Based on the individual alternative, the acreage for revegetation was estimated and, therefore, differs for various alternatives. EPA Region VII estimated that the revegetation can be conducted for about $1,000 per acre including topsoil addition, soil amendments, and reseeding.

- Mine waste tonnages and the remedy areas consist of those within the eight defined EPA zones. Other areas of the subsite were excluded since these areas have minor mining-related disturbances (Andes, 1988a) relative to the eight defined zones.

- All remedial construction has been presumed to be conducted at Health and Safety Level D.

- All costs were calculated in current 1989 dollars. Federal pricing guidelines for contracts were considered.

- Because of the Bevill Amendment provisions, Subtitle C of RCRA considerations have been assumed to be inapplicable to the detailed analysis of the remedy alternatives.

- The potential ARARs for this operable unit are presented and discussed in the 1988 OUFS, Chapters 4 and 6. However, the preferred alternative may not attain contaminant-specific applicable or relevant and appropriate requirements, such as the

maximum cotaminant levels (MCLs) established by the Safe Drinking Water Act for cleanup of the groundwater and the ambient water quality criteria (AWQC) established by the Clean Water Act for cleanup of the surface water. Compliance with such requirements at this subsite appears tecnically impractical for this operable unit remedial action (see Section 121 (d)(4)(C) and (F) of CERCLA 42 USC ¶9621 (d)(4)(C) and (F). Some are specific to alternatives developed in the 1988 OUFS, whereas others may be specific to alternatives developed for this OUFS Supplement. Tables 4-1, 4-2, 4-3, and 6-4, from the 1988 OUFS, have been duplicated for convenience and are presented as Tables 3-2, 3-3, 3-4, and 3-5 in this document.

- The cost estimates have been prepared for guidance in project evaluation and implementation from the information available at the time of the estimate. The final costs of the project would depend on actual labor and material costs, actual site conditions, productivity, competitive market conditions, final project scope and schedule, the firm selected for final engineering design, and other variable factors. As a result, the final project costs would vary from the estimate presented here. Because of these factors, funding needs must be carefully reviewed prior to making specific financial decisions or establishing final budgets.

INDIVIDUAL ANALYSIS OF ALTERNATIVES

The remedial alternatives selected for detailed analysis were developed and described in the previous section. The

Table 3-2
POTENTIAL CONTAMINANT-SPECIFIC EVALUATION CRITERIA
CHEROKEE COUNTY SITE
GALENA SUBSITE

Contaminant	CWA Human Health	Federal (SDWA) (µg/l) MCL Primary	Federal (SDWA) (µg/l) MCL Secondary	Federal (SDWA) (µg/l) MCLG	Kansas (KDHE) (µg/l) Domestic Water Supply	Kansas (KDHE) (µg/l) Kansas Action Level[a]
Arsenic	--	50	--	50	50	50
Barium	--	1,000	--	1,500	1,000	1,000
Cadmium	10	10	--	5	10	5
Chromium (VI)	50	50 (total)	--	120 (total)	50	50
Copper	1,000	--	1,000	1,300	--	1,000
Iron	--	--	300	--	--	300
Lead	50	50	--	20	50	50
Manganese	--	--	50	--	--	50
Mercury	10	2	--	3	2	2
Nickel	15.4	--	--	--	--	1,000
Selenium	10	10	--	45	10	10
Silver	50	50	--	--	50	50
Zinc	5,000	--	5,000	--	--	5,000

[a]KAL--Groundwater Contaminant Cleanup Target Concentrations for fresh, usable aquifer.

AQUATIC LIFE

Contaminant	Federal (CWA) (µg/l) Aquatic Life Chronic	Federal (CWA) (µg/l) Aquatic Life Acute	Kansas (µg/l) GW Targets[a] AKNL[b] (Chronic)	Kansas (µg/l) GW Targets[a] AKNL[b] (Acute)	Kansas (µg/l) Kansas WQC Chronic	Kansas (µg/l) Aquatic Life Acute
Arsenic	190	360	--	--	190	360
Barium	--	--	--	--	--	--
Cadmium	1.1	3.9	--	--	0.66[c]	1.8[c]
Chromium (VI)	11	16	--	--	11	16
Copper	12	18	26	42[d]	6.5[c]	9.2[c]
Iron	1,000	--	--	--	1,000	--
Lead	3.2[e]	82[e]	--	--	1.3[c]	34[c]
Manganese	--	--	--	--		
Mercury	0.012	2.4	0.012	2.4	0.012	2.4
Nickel	160	1,400	--	--	56[c]	1,089[c]
Selenium	35	260	--	--	35	260
Silver	0.12	4.1	0.12	198[d]	0.12	1.2[c]
Zinc	110	120	47	690[d]	47	180[c]

[a]Groundwater Contaminant Cleanup Target Concentration (aquifer discharge via springs or seeps to surface). Nonpromulgated.

[b]Alternative Kansas Notification/Action Levels applies to aquifers that surface through springs or seeps.

[c]Hardness dependent (value based on $CaCO_3$ greater than 150 mg/l).

[d]Hardness dependent (value based on 251-400 mg/l $CaCO_3$).

[e]Hardness dependent (value based on 100 mg/l $CaCO_3$).

From 1988 OUFS, Table 4-1.

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Table 3-3
POTENTIAL FEDERAL LOCATION-SPECIFIC EVALUATION CRITERIA

Location	Requirement	Prerequisite(s)	Citation
Within 100-year flood plain	Facility must be designed, constructed, operated, and maintained to avoid washout. Facility must not cause encroachment into legally defined floodway.	All facilities including RCRA treatment, storage, or disposal	40 CFR 264.18(b)
Within flood plain	Action to avoid adverse effects, minimize potential harm, restore and preserve natural and beneficial values	Action that will occur in a flood plain, i.e., lowlands, and relatively flat areas adjoining inland and coastal waters and other flood-prone areas	Executive Order 11988, Protection of Flood Plains (40 CFR6, Appendix A)
Within areas where action may cause irreparable harm, loss, or destruction of significant artifacts	Action to recover and preserve artifacts	Alteration of terrain that threatens significant scientific, prehistorical, historical, or archeological data	National Archeological and Historical Preservation Act (16 U.S.C. Section 469); 36 CFR Part 65
Historic project owned or controlled by federal agency	Action to preserve historic properties; planning of action to minimize harm to National Historic Landmarks	Property included in or eligible for the National Register of Historic Places	National Historic Preservation Act, Section 106 (16 U.S.C. 470 et seq.); 36 CFR Part 800
Critical habitat upon which endangered species or threatened species depend	Action to conserve endangered species or threatened species, including consultation with the Department of Interior	Determination of endangered species or threatened species	Endangered Species Act of 1973 (16 U.S.C. 1531 et seq.); 50 CFR Part 200, 30 CFR Part 402
Wetland	Action to minimize the destruction, loss, or degradation of wetlands	Wetland as defined by Executive Order 11990, Section 7	Executive Order 11990, Protection of Wetlands (40 CFR, 6, Appendix A)
Wetland	Action to prohibit discharge of dredged or fill material into wetland without permit		Clean Water Act, Section 404; 40 CFR, Parts 230 and 231
Area affecting stream or river	Action to protect fish or wildlife	Diversion, channeling, or other activity that modifies a stream or river and affects fish or wildlife	Fish and Wildlife Coordination Act (16 U.S.C. 661 et seq.); 40 CFR 6.302
Waters of the United States	Action to dispose of dredge and fill material into waters is prohibited without a permit	Waters of the United States	Clean Water Act, Section 404, 40 CFR 125, Subpart M

From 1988 OUFS, Table 4-2

Table 3-4
POTENTIAL KANSAS LOCATION-SPECIFIC EVALUATION CRITERIA

Location	Prerequisite/Requirements	Citation
Within 100-year flood plain	Public water supply facility, except for intake structures, will not be constructed at a site within the 100-year flood plain	KDHE Regulations 28-15-17, Siting Requirements
Adjacent to potential pollution source	Public water supply facility will not be located adjacent to a major source of pollution which may potentially adversely affect the water supply	KDHE Regulations 28-15-17, Siting Requirements
Risks from floods, earthquakes, fires, or other disasters (including potential instabilities from mine voids and subsidences)	Public water supply facility will not be located at a site where it may be subject to risk from earthquakes, floods, fires, or other disasters which could cause a breakdown of the water supply	KDHE Regulations 28-15-17, Siting Requirements
Critical habitat upon which endangered or threatened species depend	Action to conserve threatened or endangered species, in consultation with Kansas Fish & Game Commission	Kansas Nongame and Endangered Species Conservation Act of 1975

From 1988 CUFS, Table 4-3

Table 3-5
POTENTIAL ACTION-SPECIFIC ARAR'S--FEDERAL AND STATE

Remedial Measure	ARARs	Comments
Removal of Sulfide Minerals	30 U.S.C. 801--Federal Mine Safety and Health Act	Pertains to worker safety at mining operations
	40 CFR 122, 125--National Pollutant Discharge Elimination System	Regulates the discharge of pollutants from any point source into waters of the United States or Kansas
	40 CFR 440--Effluent Limitations	Sets technology-based effluent limitations for point source discharges in the Ore Mining and Dressing Point Source Category
Treatment of Sulfide Minerals	40 CFR 61--National Emission Standards for Hazardous Air Pollutants	Sets emission standards for designated hazardous pollutants, including mercury, beryllium, and inorganic arsenic
	40 CFR 264--Standards for Owners and Operators of Hazardous Waste Treatment, Storage, and Disposal Facilities K.S.A. 34, 29 Standards for acceptable management of solid and hazardous wastes	Portions of these regulations may be relevant and appropriate.
	40 CFR 122, 125--National Pollutant Discharge Elimination System	Regulates the discharge of pollutants from any point source into waters of the United States
Containment of Sulfide Minerals	40 CFR 257--Criteria for Classification of Solid Waste Disposal Facilities and Practices	Establishes criteria for solid waste disposal facilities
	40 CFR 264--Standards for Owners and Operators of Hazardous Waste Treatment, Storage, and Disposal Facilities K.S.A. 34, 29 Standards for acceptable management of solid and hazardous wastes	Portions of these regulations may be relevant and appropriate to disposal of mined or processed sulfide rock
Shaft and Mine Backfilling	30 U.S. C. 801-962--Federal Mine Safety and Health Act	Pertains to worker safety at mining operations
Treatment of Groundwater/Surface Water and Disposal of Byproducts	40 CFR 122, 125--National Pollutant Discharge Elimination System	Regulates the discharge of pollutants from any point source into waters of the United States or Kansas
	40 CFR 257--Criteria for Classification of Solid Waste Disposal Facilities and Practices	Establishes criteria for solid waste disposal facilities, if the treatment plant sludge qualifies as a solid waste. Portions of 40 CFR Part 264 may also be relevant and appropriate.

Table 3-5
(continued)

Remedial Measure	ARARs	Comments
Treatment of Groundwater/Surface Water and Disposal of Byproducts (continued)	40 CFR 264--Standards for Owners and Operators of Hazardous Waste Treatment, Storage, and Disposal Facilities	If the treatment plant sludge or biomass waste qualify as a hazardous waste, these requirements establish standards for managing hazardous waste
	Kansas Groundwater Management District Act; Kansas Water Appropriation Act	Controls well location, spacing, water use, aquifer depletion, and appropriations
	Kansas Water Appropriation Act	Controls water rights and appropriations.
	Kansas Water Transfer Act	Requires state approval for water transfers
	Kansas Statutes Annotated, Articles Nos. 34 and 29	Standards define the acceptable management of solid and hazardous wastes
	Kansas Administrative Regulation 28-30-1	Regulates construction, reconstruction, treatment, and plugging of water wells
Surface Water Diversion	40 CFR 230-231--Clean Water Act--Dredge or Fill Requirements	Establishes requirements for discharge of dredged or fill materials, or work in or affecting, navigable waters
	33 CFR 320-330--Rivers and Harbors Act--Section 10 Permit	
Transportation	49 CFR 107, 171 et seq.--Hazardous Materials Transportation Regulations	For the transportation of mines or processed material to a permanent, offsite disposal facility
	40 CFR 262--Standards Applicable to Generators of Hazardous Waste	Establishes regulations for offsite transportation of hazardous waste
	40 CFR 263--Standards Applicable to Transporters of Hazardous Waste	Establishes regulations for offsite disposal of hazardous waste
General	29 U.S.C. 667--Occupational Safety and Health Act	Regulates worker health and safety. Applies, except where the Mine Safety and Health Act

From 1988 OUFS, Table 6-4

specific components that make up the alternatives are summarized in Table 2-2. For each of the alternatives, the block flowsheets shown in Section 2 can be referenced. A discussion of each alternative in reference to the assessment criteria is then presented.

ALTERNATIVE 1: NO ACTION

The no-action alternative is analyzed to provide a baseline for the comparative analysis. Because this alternative does not implement remedial actions, the long-term human health and the environmental risks would remain the same as those identified in the OUFS risk assessment (EPA, 1988a).

Overall Protection of Human Health and the Environment

This alternative does not protect human health and the environment more than the current situation. The existing health threat from the metals-contaminated mine wastes exposed on the surface would continue. Shallow groundwater would continue to exceed primary drinking water standards. The groundwater contamination and contaminated surface water runoff would continue to have a detrimental impact to surface water quality on the subsite and the Spring River.

Compliance with ARARs

Implementation of the no-action alternative would not meet any of the potential ARARs identified in the 1988 OUFS for the subsite. The 1988 OUFS (see Chapter 8) presented a model that evaluated the contaminant source contributions to the surface water and groundwater systems. The results of the model indicated about 26 percent of the sulfate, zinc, and cadmium contributing to Short Creek metals loadings is from the surface-deposited mine wastes on the Galena Subsite.

However, over 70 percent of the contributing load is from the subsurface mine workings and the mine wastes within the workings. Therefore, regardless of the action taken with surface wastes, metals concentrations in the groundwater and surface water will not achieve ARARs for these media.

Long-Term Effectiveness and Permanence

This alternative provides no controls for the contaminants onsite. Current and potential future risks are not affected by this alternative.

Reduction of Toxicity, Mobility, and Volume

This alternative does not reduce the toxicity, mobility, or volume of the contaminants onsite. The weathering process and other transport mechanisms would continue to allow dissolution and migration of metal contaminants.

Short-Term Effectiveness

This alternative poses no additional risks to the community, workers, or the environment relative to short-term effectiveness.

Implementability

Implementability concerns do not apply because no action would be implemented.

Costs

This alternative would have no capital costs. However, since contaminants are left onsite, a review of the site conditions

would have to be conducted at 5-year intervals. The cost associated with each review is estimated at $125,000.

ALTERNATIVES 2 THROUGH 5: COMMON COMPONENTS

Deep well remediation and surface water rechannelization are remedial actions common to all the action alternatives (Nos. 2 through 5). Assessments of these actions against the seven evaluation criteria are presented below. The assessments of the remainder of the alternatives do not repeat the assessments of these specific actions.

A simplified mass load model (presented in Appendix E) was developed to measure the relative effectiveness of individual remedial actions. The estimated changes in mass metal loads for the specific alternatives are order of magnitude and suitable only for a relative comparison between alternatives. The simplified model assumes a finite water budget: the total annual flow remains a constant, and flow is conserved (changes in shallow groundwater flow become surface water flow). Even though these considerations are quite simplistic, the relative impact between each alternative will permit a reasonable estimation of the alternatives' effectiveness in improving the water quality. Values used in estimating geochemical activity of various mine wastes are from work conducted by both EPA and PRPs during the course of the Galena site work.

The mass load model considered sulfate, dissolved cadmium, and dissolved zinc concentrations. These metals are conservative in behavior and, therefore, provide a conservative estimate for the behavior of other metal contaminants in the surface and groundwater system. As determined in the 1988 OUFS, the combined mass loads from the Owl Branch and Tributary A (Hell's Half Acre) catchments (as measured in Short

Creek) are 4,190 lb/day, 600 lb/day, and 3.6 lb/day for sulfate, zinc, and cadmium, respectively (assuming no-action). The mass load model predicts the changes in mass load for each alternative being evaluated as a percent change for each of the three parameters.

It is assumed that the deep well remediation action has no impact on the mass metals loads in the shallow groundwater and surface water systems. The estimated reduction in mass loads due to rechanneling surface streams is 13.7 percent, 24.3 percent, and 23.6 percent, for sulfate, zinc, and cadmium, respectively. The impact of implementing all components of each alternative will be presented in the section that discusses each alternative.

Deep Well Remediation

Overall Protection of Human Health and the Environment. Deep well remediation would be the only aspect of the alternatives that directly addresses the potential for contaminant migration from the metals-contaminated shallow aquifer to the regional deep aquifer (the Roubidoux). The Roubidoux aquifer is used widely in the area as a source of drinking water. The proper remediation of wells that penetrate to the Roubidoux and plugging abandoned wells can eliminate a potential contaminant migration pathway.

Compliance with ARARs. Current Roubidoux water quality in the Galena area meets primary drinking water standards. Completion of this action would help maintain the quality of water in the Roubidoux relative to potential negative influences from the Galena Subsite shallow groundwater.

Long-Term Effectiveness and Permanence. This action is limited in long-term effectiveness to the degree that deep wells can be located, inspected, and remediated if necessary. An

investigation should be conducted prior to design to locate all deep wells. In addition, continued corrosion potential for the future would require periodic inspections of the wells to verify integrity. This action protects human health because the deep aquifer is a required source of drinking water.

Reduction of Toxicity, Mobility, and Volume. This action does not employ a treatment process and, therefore, would not result in a reduction of toxicity or volume of the contaminants. However, the mobility of the contaminants that potentially may migrate into the Roubidoux formation through structural failure of a deep well would be reduced or eliminated from those wells that are remediated.

Short-Term Effectiveness. This action poses minimal risks to the community, workers, or the environment during implementation. This action can be completed within a year.

Implementability. This remedial action would use conventional technologies for well inspection, rehabilitation, and plugging. If deep wells other than those initially identified are discovered at a later time, additional actions can then be taken. Coordination with local agencies will be necessary.

Costs. The capital cost associated with this action, about $193,000, is included in the capital costs for the specific alternatives. The scope of the action included in the cost estimate is to plug one well and to rehabilitate (repair casings) three wells. No long-term O&M costs have been included for this remedial action.

Future remedial costs may be incurred if further remediation of the wells is required to prevent contamination migration from future well failure or if other abandoned wells located

in the future require remediation. These costs would be similar to the unit costs presented for well rehabilitation in Appendix F. The assumed life of a remediated well is 30 years.

Channelization

Overall Protection of Human Health and the Environment. Stream diversion and rechannelization actions are aimed at eliminating major stream capture by mine shafts and open subsidences. These actions would, therefore, reduce the recharge of the shallow groundwater system with surface water draining from upgradient areas. By reducing the groundwater recharge, the groundwater discharge to Short Creek is reduced. This, in turn, would reduce the metals loading in the Short Creek's base flow. This would result in a reduction of the metals loading from Short Creek to the Spring River and, thereby, improve water quality in Spring River. Based on the mass load model, rechannelization can reduce the net mass loads for sulfate by 13.7 percent, zinc by 24.3 percent, and cadmium by 23.6 percent. It is a very effective method to reduce mass loads and, hence, improve water quality.

Compliance with ARARs. Water quality would still not meet potential chemical-specific ARARs following implementation. The diversions and lined channel would be designed to comply with action- and location-specific ARARs.

Long-Term Effectiveness and Permanence. The long-term effectiveness of the diversion and lined channels depends on routine maintenance. The effective life of the lined channels and diversions is expected to be 30 years with routine maintenance. Routine maintenance of the structures would ensure adequate and reliable performance.

Reduction of Toxicity, Mobility, and Volume. This remedial action does not employ a treatment process and, therefore, would not result in a reduction of toxicity or volume of the contaminants. However, the mobility of the contaminants that are contained within the subsurface mine workings and mineralized areas would be reduced because of the reduced amount of recharge to the shallow groundwater system.

Short-Term Effectiveness. This action poses minimal risks to the community, workers, or the environment and can be completed within a year. When combined with other remedial actions that involve the removal of surface-deposited mine wastes, this action should be implemented following mine waste removal. Construction activities may cause some disruption to the community and streams.

Channelization and diversion activities would be scheduled for the low-flow season to minimize the potential adverse impacts of construction within the stream channel. Within the Hell's Half Acre basin (Tributary A), the flows are currently captured in a large subsidence toward the upper end of the basin. This capture could be maintained until all downstream work is finished to enable all downstream improvements to be constructed without adverse impacts on Short Creek. For the Owl Branch basin, localized stream diversions would be used to lower the potential adverse impacts of construction-related activities for the channelization and diversions.

Implementability. This remedial action uses conventional technologies, designs, and materials and should be straightforward to implement. The design of the diversions and channels must consider the stability of the local ground. Future subsidence in the area of the diversions may require construction of additional stream channels or reconstruction.

Coordination with local, state, and federal agencies will be required.

Costs. The capital cost associated with this action, about $766,000, is included in the capital costs for the specific alternative. The cost estimate includes a total of 5,600 linear feet of channel (divided among two streams). A contingency for rebuilding some of the existing, deteriorated channel in the Hell's Half Acre area is included. Annual long-term O&M associated with this remedial action is estimated to be about $3,800. O&M activities include inspection, cleaning, and repair of the lined channels.

Future costs may be incurred if the channels have to be replaced. The estimated life of the channel is 30 years. Replacement costs are assumed to be the same as initial installation costs.

ALTERNATIVE 2: MINE AND MILL ALL MINE WASTES

The primary remedial process of this alternative is to remove and treat the mine waste rock and chat to eliminate the possibility of human exposure to metals-contaminated surface-deposited mine wastes and reduce the metals loadings in the groundwater and surface water systems. This alternative proposes to remove all mine waste rock and all chat. As shown previously, Figure 2-1 presents a block diagram depicting the various components of this alternative.

Overall Protection of Human Health and the Environment

This alternative protects human health by removing the lead-contaminated exposed surface mine wastes that exceed health-based action levels for lead from the subsite. Removing the

surface mine wastes would result in reduced metal loadings in the groundwater and surface water systems, but water quality would still exceed ARARs. The Alternative Water Supply Operable Unit for the Galena Subsite would make suitable drinking water available to those users who depend on the shallow groundwater.

A simplified, relative metals mass loading model has been developed to evaluate relative effectiveness between the alternatives. Details of this model are given in Appendix E. By removing the surface mine waste and implementing the surface water rechannelization, metals loadings in the groundwater and surface water would be reduced. The model, evaluated for sulfate, zinc, and cadmium, predicts that contaminant mass loadings would be reduced by about 27.7 percent, 36.2 percent, and 31.7 percent, respectively, compared to no action.

Compliance with ARARs

This alternative would reduce metals loadings in the groundwater and surface water systems, but water quality would still not meet potential chemical-specific ARARs. Location-specific ARARs would be met through proper engineering design and implementation. The surface mine waste removal and treatment actions would achieve action-specific ARARs through proper design and operation. The designated action level for lead in soil would be attained through the removal and treatment of all the surface-deposited mine wastes to produce a tailing with metal concentrations below the action levels.

Long-Term Effectiveness and Permanence

This alternative permanently and effectively reduces the ingestion risk to human health from metals-contaminated surface mine waste rock and chat. However, subsurface metal

contaminants remain onsite. Surface water and groundwater monitoring would be required to monitor the effectiveness of the remedy.

Reduction of Toxicity, Mobility, and Volume

Reductions of mobility and volume of the contaminants in the waste rock and chat would be achieved by this alternative. Milling and flotation of the mine wastes would irreversibly remove most of the lead, zinc, and cadmium from mine wastes and produce a concentrate that has a much smaller volume than the original mine waste volume. The mobility of these metals in the environmental system is reduced through their concentration and either disposal in an appropriate containment facility or their recycling into feed stocks for finished products. Contaminants in the subsurface mine workings would remain onsite though mobility would be slightly reduced by those actions that reduce infiltration to the groundwater system.

Toxicity of the mine waste materials is reduced since all materials (waste rock and chat) would be removed and treated in the milling process. The remaining mine wastes would be less toxic since their lead levels would be less than the prescribed action level.

Short-Term Effectiveness

During implementation of this alternative, a temporary increase of subsite dust production and local heavy equipment traffic would be expected. Dust suppression would be required to protect the workers from inhalation and dermal contact. Some dust suppression activities may also be required to prevent exposure by the community. Any impact on the environment would be minimal and temporary. Workers would have to be cautious regarding work in unstable ground

areas. The implementation time required to complete mine waste removal and processing is about 3-1/2 to 4 years.

Implementability

This alternative uses available recovery, transport, and beneficiation technologies that are easily implementable. However, to achieve the metal recoveries required to produce a tailing that exhibits concentrations within established action levels, zinc and lead oxide removal require a multi-stage conditioner and flotation circuit. Additional pilot testing of the oxide flotation circuit would be required because of the expected variability in mineralogical content of the mine wastes. Mine waste feed and tailing would have to be monitored to ensure tailing quality.

Although onsite services (such as water and sewer) and capacities are expected to be adequate, access approvals to the mine waste areas from the land owners would have to be obtained. During and after implementation, groundwater and surface water quality would have to be monitored to evaluate effectiveness.

Costs

Feasibility-level (plus 50 percent to minus 30 percent) cost estimates are presented in Appendix F. Specific assumptions regarding development of the costs are also presented in the appendix. The capital cost estimate for Alternative 2 is $20,303,000. This alternative would also require long-term operating and maintenance costs for the surface stream diversion channels, estimated to total $3,830 annually. Over an analysis period of 30 years, the present worth of this alternative, calculated at a 5 percent discount rate, is $20,737,000.

The primary factors that may significantly affect the final cost of the alternative are the following:

- Actual volume/tonnage of mine waste rock and chat and required haul distances
- Proximity and volume of subsurface voids for tailing disposal
- Actual plant operating and disposal costs ($/ton)

Future remediations may be required if groundwater and surface water metal mass loads are not acceptable following implementation. These actions may include source control (such as removal of additional mineralization) or management of migration (such as pump and treat, additional void backfill, and so forth).

ALTERNATIVE 3: MINE AND MILL ALL WASTE ROCK AND HALF OF THE CHAT

The primary remedial process of this alternative is to remove and treat mine waste rock and a portion of the chat that exceeds lead action levels to eliminate human exposure to metals-contaminated surface mine wastes and reduce the metals loadings in the groundwater and surface water system. Chat characterized as having lead concentrations above the action level is removed for treatment. In-place characterization of the chat would reduce the amount of chat to be mined and transported to the mill for treatment. Contaminated (greater than the action level) chat would be mined, hauled to the mill, and then undergo wet screening to separate the fine fraction for milling and flotation to remove the metals. As shown earlier, Figure 2-3 presents a block diagram depicting the various components of this alternative.

Overall Protection of Human Health and the Environment

This alternative provides the same level of protection of human health as Alternative 2 by removing the lead-contaminated exposed surface mine wastes that exceed health-based EPA action levels from the subsite. The mass loading model (Appendix E) estimates that for this alternative, the sulfate load is reduced by 24.5 percent, the zinc load by 33.2 percent, and the cadmium load by 29.4 percent.

Compliance with ARARs

This alternative would reduce metals loadings in the groundwater and surface water systems, but water quality would still not meet potential chemical-specific ARARs. Location-specific ARARs would be met through proper engineering design and implementation. The surface mine waste removal and treatment actions would achieve action-specific ARARs through proper design and operation. The action level for lead would be attained through characterization and removal of the contaminated portion of the surface mine wastes.

Long-Term Effectiveness and Permanence

This alternative, like Alternate 2, permanently and effectively reduces the ingestion risk to human health from metals-contaminated surface mine waste rock and chat. However, subsurface metal contaminants remain onsite. Surface water and groundwater monitoring would be required to monitor the effectiveness of the remedy.

Reduction of Toxicity, Mobility, and Volume

Reductions of mobility and volume of the contaminants in the mine waste rock and chat would be achieved by this alternative. Milling and flotation of the mine wastes would irreversibly remove most of the lead, zinc, and cadmium from the mine wastes and concentrate the contaminants in a much smaller volume. Contaminants in the subsurface mine workings would remain onsite though mobility would be slightly reduced by those actions that reduce infiltration to the groundwater system. The toxicity of the metal contaminants is not reduced due to the persistence of metals regardless of treatment.

Short-Term Effectiveness

The analysis of short-term effectiveness is the same as Alternative 2. The implementation time required to complete mine waste characterization, removal, and processing would be about 3 to 3-1/2 years.

Implementability

The evaluation of implementability for this alternative is similar to that for Alternative 2. However, chat characterization would use field portable XRF technology. Procedures to standardize and ensure adequate characterization must be developed.

Although onsite services and capacities are expected to be adequate, access approvals to the mine waste areas would have to be obtained. During and following implementation, groundwater and surface water quality would be monitored to evaluate effectiveness.

Costs

Feasibility-level (plus 50 percent to minus 30 percent) cost estimates are presented in Appendix F. Specific assumptions regarding development of the costs are also presented in the appendix. The capital cost estimate for Alternative 3 is $18,400,000. This alternative would also require long-term operating and maintenance costs for the surface stream diversion channels, estimated to total $3,830 annually. Over an analysis period of 30 years, the present worth of this alternative, calculated at a 5 percent discount rate, is $18,834,000.

The primary factors that may significantly affect the final cost of the alternative are the following:

- Actual volume/tonnage of chat characterized as requiring treatment
- Actual volume/tonnage of chat/waste rock and required haul distances
- Proximity and volume of voids for tailing disposal
- Actual plant operating and disposal costs ($/ton)

Future remediations may be required if groundwater and surface water metal mass loads are not acceptable following implementation. These actions may include source control (such as removal of additional mineralization) or management of migration (such as pump and treat, additional void backfill, and so forth).

ALTERNATIVE 4: MINE AND DISPOSE OF ALL MINE WASTES IN AN ONSITE CONTAINMENT FACILITY

The primary remedial process of this alternative is to remove and dispose of mine waste rock and chat in a containment unit to eliminate human exposure to surface mine wastes exceeding action limits for lead and reduce the metals loadings in the groundwater and surface water system. As shown previously, Figure 2-4 presents a block diagram depicting the various components of this alternative.

Overall Protection of Human Health and the Environment

This alternative would provide the same level of protection of human health as Alternative 2 by removing and containing the lead-contaminated surface mine wastes. The modeling results are the same as Alternative 2. Containment of the mine wastes in a RCRA-designed facility that has essentially zero discharge would result in reduced metals loadings in the groundwater and surface water systems. The mass load model estimates the same load reduction that was calculated for Alternative 2.

Compliance with ARARs

This alternative would reduce metals loadings in the groundwater and surface water systems, but water quality would still not meet potential chemical-specific ARARs. Location-specific ARARs would be met through proper engineering design and implementation. The surface mine waste removal and containment actions would achieve action-specific ARARs.

Long-Term Effectiveness and Permanence

This alternative would permanently and effectively reduce the ingestion risk to human health from metals-contaminated surface mine waste rock and chat through containment. However, subsurface metal contaminants would remain onsite. Long-term maintenance of the containment unit would be required to maintain its integrity. Surface water and groundwater monitoring would be required to monitor the effectiveness of the remedy.

Reduction of Toxicity, Mobility, and Volume

Reduction of mobility of the contaminants in the mine waste rock and chat would be achieved by this alternative although volume would not be reduced. Toxicity of the metal contaminants would not be reduced.

Short-Term Effectiveness

The short-term effectiveness analysis is similar to that for Alternatives 2 and 3. The implementation time required to complete the mine waste characterization, removal, and processing would be about 2-1/2 years.

Implementability

As in the previous alternatives, this alternative uses conventional recovery and transport technologies that are easily implementable. The containment unit would be designed and constructed using equivalent RCRA design criteria for double containment/double leachate collection systems. Siting of the unit would need to include an evaluation of ground stability.

Access approvals to the mine waste areas would have to be obtained. During and following implementation, groundwater and surface water quality would be monitored to evaluate effectiveness.

Costs

Feasibility level (plus 50 percent to minus 30 percent) cost estimates are presented in Appendix F. Specific assumptions regarding development of the costs are also presented in the appendix. The capital cost estimate for Alternative 4 is $29,379,000.

This alternative would also require long-term operating and maintenance costs for the surface stream diversion channels and the containment unit estimated to total $222,350 annually. Most of the O&M expense would be used to maintain the containment unit (i.e., inspections; leachate collection, treatment, and disposal; erosion maintenance; and environmental monitoring). Over an analysis period of 30 years, the present worth of this alternative, calculated at a 5 percent discount rate, would be $33,173,000.

The primary factors that may affect the final cost of the alternative are the following:

- Actual volume/tonnage of waste rock/chat and haul distances
- Local availability of containment unit construction materials (clay, cover material, and topsoil)
- Location of a satisfactory site for the containment unit

Future remediations may be required if groundwater and surface water metal mass loads are not acceptable following implementation. These actions may include source control (such as removal of additional mineralization) or management of migration (such as pump and treat, additional void backfill, and so forth).

ALTERNATIVE 5: GEOCHEMICALLY CHARACTERIZE WASTE, SEGREGATE BY PARTICLE SIZE, SELECTIVELY BACKFILL, AND RECONTOUR

The primary remedial process of this alternative is to mine and selectively dispose of surface-exposed mine wastes in mine shafts and open subsidences. Recognizing that the geochemically active portion of the waste rock is the fine-sized fraction, all waste rock would be screened to separate the fines (nominally less than 2 inches) from the coarse waste rock. The waste rock fines would then be placed below grade above the groundwater zone while coarser waste rock and chat (that exceeds the lead action level) would be placed in subsidences either below or above the water table (see Appendixes C and D). Chat below lead action levels of 1,000 ppm and zinc less than 5,000 ppm would be used as cover material to reduce infiltration and prevent human contact with contaminated waste. As shown previously, Figure 2-5 presents a block diagram depicting the various components of this alternative.

Overall Protection of Human Health and the Environment

This alternative provides protection of human health by removing from potential human contact the exposed surface mine wastes that have lead concentrations in excess of the EPA action level. Selective subsurface disposal of the surface mine wastes in conjunction with surface water

rechannelization and recontouring would result in reduced metals loadings in the groundwater and surface water systems but water quality would still exceed ARARs. The mass load model (Appendix E) estimates reduction in mass loads for sulfate, zinc, and cadmium to be 18.4 percent, 29.5 percent, and 25.6 percent, respectively. The alternative water supply operable unit for the Galena Subsite would make suitable drinking water available to users who depend on the contaminated shallow groundwater system.

Compliance with ARARs

This alternative would reduce metals loadings in the groundwater and surface water systems, but water quality would still not meet potential chemical-specific ARARs. Location-specific ARARs would be met through proper engineering design and implementation.

Long-Term Effectiveness and Permanence

This alternative, like Alternative 2, effectively reduces the ingestion risk to human health from metals contaminated surface mine waste rock and chat. Selective subsurface placement of the mine wastes results in a controlled release (controlled by waste surface area available for mineral dissolution) of metals to the groundwater system. Compaction of the wastes during backfill will reduce the likelihood of subsidence resulting in placement of the waste rock fines (highly mineralized) in the saturated zone. The metals loadings in the shallow groundwater and surface water systems would be reduced due to reduced groundwater flow. As in the other alternatives, surface water and groundwater monitoring would be required to monitor the impact of the remedy.

Reduction of Toxicity, Mobility, and Volume

Volume of the onsite contaminants would not be reduced. Reduction of metal mobility of the contaminants in the mine waste rock and chat would be achieved because of the reduced shallow groundwater flow through the workings that would result from the selective placement of mine wastes. Toxicity of the metal contaminants would not be reduced.

Short-Term Effectiveness

As in the previous alternatives, a temporary increase of subsite dust production and local heavy equipment traffic would be expected. During backfill of mine wastes into water-filled voids, displacement of the water would occur. However, due to the length of time (1 year) over which the backfilling would be implemented, displacement of water would be gradual and, therefore, have minimal impact to the quality of groundwater and surface water. The implementation time required to complete the mine waste removal and disposal remedial actions would be about 1 year.

Implementability

Mining and disposal of the surface mine wastes would use conventional technologies. Waste rock screening would use conventional dry screening technology that can be easily relocated from area to area as needed. Characterization of the chat would use XRF technology and sample preparation practices described previously.

Access approvals to the mine waste areas would have to be obtained. During and following implementation, groundwater and surface water quality would be monitored to evaluate effectiveness.

Costs

Feasibility level (plus 50 percent to minus 30 percent) cost estimates are presented in Appendix F. Specific assumptions regarding development of the costs are also presented in the appendix. The capital cost estimate for Alternative 5 is $8,295,000. This alternative also requires operating and maintenance costs for the surface stream diversion channels and the mine waste backfill covers estimated to total $14,960 annually. Over an analysis period of 30 years, the present worth of this alternative, calculated at a 5 percent discount rate, is $8,900,800.

The primary factors that may significantly affect the final cost of the alternative would be the following:

- Actual volume/tonnage of chat/waste rock and required haul distance

- The relative fractions of the waste rock and chat that fall into each classification

Future remediations may be required if groundwater and surface water metal mass loads are not acceptable following implementation. These actions may include source control (such as removal of mineralization) or management of migration (such as pump and treat and additional void backfill).

COMPARATIVE ANALYSIS OF ALTERNATIVES

A comparative analysis of the alternatives provides a relative evaluation of the alternatives for each of the specific criteria. This evaluation is presented in matrix format in Table 3-6. This table uses the most recent guidance interpretation of the assessment criteria (EPA, 1988b). These criteria were presented in Table 3-1. State and community

acceptance criteria will be addressed following the public comment period.

The detailed analysis of alternatives in the 1988 OUFS was developed under previous EPA RI/FS guidance (EPA, 1987b). The details of some of the assessment criteria as presented in the most recent guidance (EPA, 1988b) have been modified from the former guidance. Table 3-7 summarizes the detailed analysis presented in this supplement to the OUFS using the requirements in the former guidance that were used in the OUFS. This format allows a direct comparison of the detailed analysis for the alternatives developed in both the OUFS and this supplement to the OUFS.

Criteria	Alternative 1 No Action	Alternative 2 Mine and Mill All Mine Wastes	Alternative 3 Mine and Mill All Mine Waste Rock and Half of the Chat
OVERALL PROTECTIVENESS			
Human Health Protection			
- Direct Contact/Mine Wastes Ingestion	Existing health threat from surface areas contaminated with lead greater than action level.	All contaminated surface solids above action level in disturbed areas removed.	All contaminated surface solids above action level in disturbed areas removed.
- Groundwater Ingestion	Primary drinking water standards exceeded. Alternate water supply for existing users being provided through other remedy.	Primary drinking water standards exceeded. Alternate water supply for existing users being provided through other remedy.	Primary drinking water standards exceeded. Alternate water supply for existing users being provided through other remedy.
Environmental Protection	Contaminated waters exceed AWQC in Short Creek and other surface waters.	Mass metal loads reduced but water quality still does not meet AWQC.	Mass metal loads reduced but water quality still does not meet AWQC.
COMPLIANCE WITH ARARS			
Chemical-Specific ARARs	No action does not meet chemical-specific ARARs.	Does not meet chemical-specific ARARs but will reduce mass metal loading.	Does not meet chemical-specific ARARs but will reduce mass metal loading.
Location-Specific ARARs	Not relevant for no action.	Facilities designed to meet location-specific ARARs.	Facilities designed to meet location-specific ARARs.
Action-Specific ARARs	Action-specific ARARs are not relevant.	Mining and milling actions would be designed to comply with 30 USC 801-962. Milling plant would be designed and operated to meet appropriate NPDES discharge requirements including 40 CFR 440. The air emissions from the milling operation would be designed to meet the criteria of 40 CFR 61.	This alternative would have the same action-specific ARARs to be considered as for Alternative 2.
Other Criteria and Guidance	Would not protect human exposure to lead levels greater than action level in waste rock and chat.	All contaminated surface solids within disturbed areas exceeding lead action level removed.	All contaminated surface solids within disturbed areas exceeding lead action level removed.
LONG-TERM EFFECTIVENESS AND PERMANENCE			
Magnitude of Residual Risk	Mass metal loads not reduced.	Sulfate, zinc, and cadmium net mass loads reduced 27.6, 36.2, and 31.7%.	Sulfate, zinc, and cadmium net mass loads reduced 24.5, 33.2, and 29.4%.
- Mine Waste Ingestion	No long-term changes to current risk.	Alternative permanently removes ingestion risk from those areas where mine waste rock and chat are removed.	Alternative permanently removes ingestion risk from those areas where mine waste rock and chat are removed. Residual risk substantially reduced due to removal of metals in contaminated surface solids.

Criteria	Alternative 1 No Action	Alternative 2 Mine and Mill All Mine Wastes	Alternative 3 Mine and Mill All Mine Waste Rock and Half of the Chat
- Groundwater Ingestion	Primary drinking water standards continue to be exceeded in shallow groundwater. Alternate water supply (AWS) available for shallow groundwater users.	Metals mass loadings reduced; however, primary drinking water standards continue to be exceeded. AWS available for shallow groundwater users.	Metals mass loadings reduced; however, primary drinking water standards continue to be exceeded. AWS available for shallow groundwater users.
- Environmental Protection	AWQC exceeded in surface streams.	Metal mass loading reduced but AWQC still exceeded in surface streams.	Metal mass loading reduced but AWQC still exceeded in surface streams.
Adequacy and Reliability of Controls	No controls over contamination. No reliability.	Milling provides permanent removal; therefore, adequate and reliable. Stream diversions and revegetation actions provide some control of migration for remaining contaminants, though long-term O&M will be required. Long-term ground and surface water monitoring required to monitor effectiveness of controls.	Milling provides permanent removal; therefore, adequate and reliable. Stream diversions and revegetation actions provide some control of migration for remaining contaminants, though long-term O&M will be required. Long-term ground and surface water monitoring required to monitor effectiveness of controls.
Need for 5-year Review	A review at 5-year intervals will be needed to assess site conditions.	A review at 5-year intervals will be needed to assess site conditions.	A review at 5-year intervals will be needed to assess site conditions.
REDUCTION OF TOXICITY, MOBILITY, OR VOLUME THROUGH TREATMENT			
Treatment Process Used	None.	Chat wet screening and milling and flotation of waste rock and chat fines.	Field XRF characterization and screening of chat exceeding action levels, milling and flotation of waste rock and chat fines.
Amount Destroyed or Treated	None.	Would recover for potential resale about 85 percent of metals from processed waste rock and chat.	Would recover for potential resale 85 percent of metals from processed waste rock and chat.
Degree of Expected Reduction of Toxicity, Mobility, or Volume	None.	Reduction of mobility and volume of site contaminants will be achieved through the milling process for waste rock and chat. There is no reduction of toxicity.	Reduction of mobility and volume of site contaminants will be achieved through the milling process for waste rock and half of the chat. There is no reduction of toxicity.
Statutory preference for treatment	Does not meet this criterion because there is no treatment.	Utilizes milling process to remove the metals from the mine wastes and chat. This process is permanent and is considered to meet the statutory preference for innovative treatment as expressed in SARA.	Alternative 3 uses the basic processes as Alternative 2 for mine waste rock and the chat exceeding the action level.

Criteria	Alternative 1 No Action	Alternative 2 Mine and Mill All Mine Wastes	Alternative 3 Mine and Mill All Mine Waste Rock and Half of the Chat
Irreversible Treatment	Not applicable.	Milling and flotation are irreversible treatments.	Milling and flotation are irreversible treatments.
Type and Quantity of Residuals Remaining After Treatment	Mine waste rock and chat will continue to weather, releasing metals to ground and surface water.	Low levels of metals with concentrations less than action levels, remain in mill tailing.	No contaminated surface mine waste remains onsite.
SHORT-TERM EFFECTIVENESS			
Community Protection	Risk to community not increased by remedy implementation.	Temporary increase of dust production and truck traffic (haulage of waste rock and chat to mill).	Temporary increase of dust production and truck traffic (haul age of waste rock and chat to mill).
Worker Protection	No risk to workers.	Protection from dust exposure and dermal contact will be required. Workers must be cautious concerning unstable ground conditions.	Protection from dust exposure and dermal contact will be required. Workers must be cautious concerning unstable ground conditions.
Environmental Impacts	Continued impact from existing conditions.	As mine waste rock is removed, contaminant mass load will decrease.	As mine waste rock is removed, contaminant mass load will decrease.
Time until Action is Complete	Not applicable.	3-1/2 to 4 years.	3 to 3-1/2 years.
IMPLEMENTABILITY			
Ability to Construct/Operate Technology	Not applicable.	Conventional recovery, transport, and beneficiation technologies are easily implementable. However, enhanced recovery to meet action level requires innovative technologies. Rechanneling, recontouring, revegetation, and well remediation are all conventional technologies that should be easy to implement.	Conventional recovery, transpor and beneficiation technologies are easily implementable. However, enhanced recovery to meet action level requires innovative technologies. Innovative use of XRF technology will provide characterization o chat. Rechanneling, recontouring, revegetation, and well remediation are all conventiona technologies that should be eas to implement.
Reliability of the Technology	Not applicable.	Reliability of treatment process requires monitoring of feed and tailing product quality. The reliability of the recontouring, channelization, revegetation, and well remediation will depend on routine maintenance.	Reliability of treatment proces requires monitoring of feed and tailing product quality. Reliability of the chat XRF characterization operations will be controlled through detailed procedures. The reliability of the recontouring, channelization, revegetation, and well remediation will depend on routine maintenance.
Ease of Doing More Action if Needed	Not applicable.	Alternative removes and treats all surface mine wastes.	Additional chat could be removed.

Criteria	Alternative 1 No Action	Alternative 2 Mine and Mill All Mine Wastes	Alternative 3 Mine and Mill All Mine Waste Rock and Half of the Chat
Ability to Monitor Effectiveness	Preremediation baseline water quality well documented.	Monitoring of shallow groundwater and surface water will provide measure of remedy effectiveness.	Monitoring of shallow groundwater and surface water will provide measure of remedy effectiveness.
Ability to Obtain Approvals/ Coordination with Agencies	Not applicable.	Property access must be obtained. Regulatory approval should be easily obtained.	Property access must be obtaine Regulatory approval should be easily obtained.
Availability of Services and Capacities	No services or capacities required.	Services and capacities easily obtainable.	Services and capacities easily obtainable.
Availability of Equipment, Specialists, and Materials	None required.	Readily available.	Readily available.
Technology Availability	None required.	Oxide mineral flotation stage will require pilot testing for expected wide range of mine waste mineralogical variability and content. Rechannelization, revegetation, and well remediation are all readily available technologies.	Oxide mineral flotation stage will require pilot testing for expected wide range of mine waste mineralogical variability and content. Waste characterization with XRF technology will require development of standard procedures. Rechannelization, revegetation, and well remediation are all readily available technologies.
COST			
Capital Cost	$0	$20,302,000	$18,400,000
First Year Annual O&M Cost	$0	$3,830	$3,830
Present Worth Cost	$0	$20,737,000	$18,834,300

Criteria	Alternative 1 No Action	Alternative 2 Mine and Mill All Mine Wastes	Alternative 3 Mine and Mill All Mine Waste Rock and Half of the Chat
IMPLEMENTABILITY			
Ability to Construct/Operate Technology	Not applicable.	Contional recovery, transport, and beneficiation technologies are easily implementable. However, enhanced recovery to meet action level requires innovative technologies. Rechanneling, recontouring, revegetation, and well remediation are all conventional technologies that should be easy to implement.	Conventional recovery, transport and beneficiation technologies are easily implementable. However, enhanced recovery to meet action level requires innovative technologies. Innovative use of XRF technology will provide characterization of chat. Rechanneling, recontouring, revegetation, and well remediation are all conventional technologies that should be easy to implement.
Reliability of the Technology	Not applicable.	Reliability of treatment process requires monitoring of feed and tailing product quality. The reliability of the recontouring, channelization, revegetation, and well remediation will depend on routine maintenance.	Reliability of treatment process requires monitoring of feed and tailing product quality. Reliability of the chat XRF characterization operations will be controlled through detailed procedures. The reliability of the recontouring, channelization, revegetation, and well remediation will depend on routine maintenance.
Ease of Doing More Action if Needed	Not applicable.	Alternative removes and treats all surface mine wastes.	Additional chat could be removed.
Ability to Monitor Effectiveness	Preremediation baseline water quality well documented.	Monitoring of shallow groundwater and surface water will provide measure of remedy effectiveness.	Monitoring of shallow groundwater and surface water will provide measure of remedy effectiveness.
Ability to Obtain Approvals/ Coordination with Agencies	Not applicable.	Property access must be obtained. Regulatory approval should be easily obtained.	Property access must be obtained. Regulatory approval should be easily obtained.
Availability of Services and Capacities	No services or capacities required.	Services and capacities easily obtainable.	Services and capacities easily obtainable.
Availability of Equipment, Specialists, and Materials	None required.	Readily available.	Readily available.
Technology Availability	None required.	Oxide mineral flotation stage will require pilot testing for expected wide range of mine waste mineralogical variability and content. Rechannelization, revegetation, and well remediation are all readily available technologies.	Oxide mineral flotation stage will require pilot testing for expected wide range of mine waste mineralogical variability and content. Waste characterization with XRF technology will require development of standard procedures. Rechannelization, revegetation, and well remediation are all readily available technologies.

Criteria	Alternative 1 No Action	Alternative 2 Mine and Mill All Mine Wastes	Alternative 3 Mine and Mill All Mine Waste Rock and Half of the Chat
REDUCTION OF TOXICITY, MOBILITY, OR VOLUME THROUGH TREATMENT			
Treatment Process Used	None.	Chat wet screening and milling and flotation of waste rock and chat fines.	Field XRF characterization and screening of chat exceeding action levels, milling and flotation of waste rock and chat fines.
Amount Destroyed or Treated	None.	Would recover for potential resale about 85 percent of metals from processed waste rock and chat.	Would recover for potential resale 85 percent of metals fro processed waste rock and chat.
Degree of Expected Reduction of Toxicity, Mobility, or Volume	None.	Reduction of mobility and volume of site contaminants will be achieved through the milling process for waste rock and chat. There is no reduction of toxicity.	Reduction of mobility and volum of site contaminants will be achieved through the milling process for waste rock and half of the chat. There is no reduc tion of toxicity.
Statutory preference for treatment	Does not meet this criterion because there is no treatment.	Utilizes milling process to remove the metals from the mine wastes and chat. This process is permanent and is considered to meet the statutory preference for innovative treatment as expressed in SARA.	Alternative 3 uses the basic processes as Alternative 2 for mine waste rock and the chat exceeding the action level.
Irreversible Treatment	Not applicable.	Milling and flotation are irreversible treatments.	Milling and flotation are irreversible treatments.
Type and Quantity of Residuals Remaining After Treatment	Mine waste rock and chat will continue to weather, releasing metals to ground and surface water.	Low levels of metals with concentrations less than action levels, remain in mill tailing.	No contaminated surface mine waste remains onsite.
SHORT-TERM EFFECTIVENESS			
Community Protection	Risk to community not increased by remedy implementation.	Temporary increase of dust production and truck traffic (haulage of waste rock and chat to mill).	Temporary increase of dust production and truck traffic (haul age of waste rock and chat to mill).
Worker Protection	No risk to workers.	Protection from dust exposure and dermal contact will be required. Workers must be cautious concerning unstable ground conditions.	Protection from dust exposure and dermal contact will be required. Workers must be cautious concerning unstable ground conditions.

Criteria	Alternative 1 No Action	Alternative 2 Mine and Mill All Mine Wastes	Alternative 3 Mine and Mill All Mine Waste Rock and Half of the Chat
Environmental Impacts	Continued impact from existing conditions.	As mine waste rock is removed, contaminant mass load will decrease.	As mine waste rock is removed, contaminant mass load will decrease.
Time until Action is Complete	Not applicable.	3-1/2 to 4 years.	3 to 3-1/2 years.
LONG-TERM EFFECTIVENESS AND PERMANENCE			
Magnitude of Residual Risk	Mass metal loads not reduced.	Sulfate, zinc, and cadmium net mass loads reduced 27.6, 36.2, and 31.7%.	Sulfate, zinc, and cadmium net mass loads reduced 24.5, 33.2, and 29.4%.
- Mine Waste Ingestion	No long-term changes to current risk.	Alternative permanently removes ingestion risk from those areas where mine waste rock and chat are removed.	Alternative permanently remove: ingestion risk from those area: where mine waste rock and chat are removed. Residual risk sub stantially reduced due to removal of metals in contaminated surface solids.
- Groundwater Ingestion	Primary drinking water standards exceeded. Alternate water supply for existing users being provided through other remedy.	Primary drinking water standards exceeded. Alternate water supply for existing users being provided through other remedy.	Primary drinking water standar exceeded. Alternate water supply for existing users bein provided through other remedy.
Environmental Protection	Contaminated waters exceed AWQC in Short Creek and other surface waters.	Mass metal loads reduced but water quality still does not meet AWQC.	Mass metal loads reduced but water quality still does not meet AWQC.
- Groundwater Ingestion	Primary drinking water standards continue to be exceeded in shallow groundwater. Alternate water supply (AWS) available for shallow groundwater users.	Metals mass loadings reduced; however, primary drinking water standards continue to be exceeded. AWS available for shallow groundwater users.	Metals mass loadings reduced; however, primary drinking wate standards continue to be exceeded. AWS available for shallow groundwater users.
- Environmental Protection	AWQC exceeded in surface streams.	Metal mass loading reduced but AWQC still exceeded in surface streams.	Metal mass loading reduced but AWQC still exceeded in surface streams.
Adequacy and Reliability of Controls	No controls over contamination. No reliability.	Milling provides permanent removal; therefore, adequate and reliable. Stream diversions and revegetation actions provide some control of migration for remaining contaminants, though long-term O&M will be required. Long-term ground and surface water monitoring required to monitor effectiveness of controls.	Milling provides permanent removal; therefore, adequate a reliable. Stream diversions a revegetation actions provide some control of migration for remaining contaminants, though long-term O&M will be required. Long-term ground and surface water monitoring required to monitor effectiveness of controls.
Need for 5-year Review	A periodic 5-year review will be needed to assess site conditions.	A periodic 5-year review will be needed to assess site conditions.	A periodic 5-year review will b needed to assess site conditions.

Criteria	Alternative 1 No Action	Alternative 2 Mine and Mill All Mine Wastes	Alternative 3 Mine and Mill All Mine Waste Rock and Half of the Chat
COST			
Capital Cost	$0	$20,302,000	$18,400,000
First Year Annual O&M Cost	$0	$3,830	$3,830
Present Worth Cost	$0	$20,737,000	$18,834,300
COMPLIANCE WITH ARARS			
Chemical-Specific ARARs	No action does not meet chemical-specific ARARs.	Does not meet chemical-specific ARARs but will reduce mass metal loading.	Does not meet chemical-specific ARARs but will reduce mass meta loading.
Location-Specific ARARs	Not relevant for no action.	Facilities designed to meet location-specific ARARs.	Facilities designed to meet location-specific ARARs.
Action-Specific ARARs	Action-specific ARARs are not relevant.	Mining and milling actions would be designed to comply with 30 USC 801-962. Milling plant would be designed and operated to meet appropriate NPDES discharge requirements including 40 CFR 440. The air emissions from the milling operation would be designed to meet the criteria of 40 CFR 61.	This alternative would have the same action-specific ARARs to b considered as for Alternative 2
Other Criteria and Guidance	Would not protect human exposure to lead levels greater than action level in waste rock and chat.	All contaminated surface solids within disturbed areas exceeding lead action level removed.	All contaminated surface solids within disturbed areas exceedin lead action level removed.
OVERALL PROTECTIVENESS			
Human Health Protection			
- Direct Contact/Mine Wastes Ingestion	Existing health threat from surface areas contaminated with lead greater than action level.	All contaminated surface solids above action level in disturbed areas removed.	All contaminated surface solids above action level in disturbed areas removed.
- Groundwater Ingestion	Primary drinking water standards exceeded. Alternate water supply for existing users being provided through other remedy.	Primary drinking water standards exceeded. Alternate water supply for existing users being provided through other remedy.	Primary drinking water standard exceeded. Alternate water supply for existing users being provided through other remedy.
Environmental Protection	Contaminated waters exceed AWQC in Short Creek and other surface waters.	Mass metal loads reduced but water quality still does not meet AWQC.	Mass metal loads reduced but water quality still does not meet AWQC.

REFERENCES

Andes, Gary. 1988a. Surface Mine Waste at the Galena Subsite, Cherokee County, Kansas. April 4, 1988.

Andes, Gary. 1988b. Field Estimate of Available Disposal Space for Surface Mine Waste at Galena, Kansas. November 1988.

AMAX Mineral Resources Company. 1988. Results and Analysis of Leaching Test, Galena Subsite, Cherokee County Site, Kansas. Prepared by Adrian Brown Consultants. December 5, 1988.

AMAX Mineral Resources Company. 1988a. Summary of PRP Additional Alternative and Comparison to the EPA Criteria. Prepared by Adrian Brown Consultants. December 5, 1988.

U.S. Environmental Protection Agency. 1988a. Groundwater and Surface Water Operable Unit Feasibility Study, Final Draft. Galena Subsite, Cherokee County Site, Kansas. February 25, 1988.

U.S. Environmental Protection Agency. 1988b. Guidance for Conducting Remedial Investigations and Feasibility Studies under CERCLA, Interim Final. OSWER Directive 9355.3-01. October 1988.

U.S. Environmental Protection Agency. 1988c. Preferred Remedy Fact Sheet, Groundwater/Surface Water Operable Unit, Galena Subsite, Cherokee County, Kansas. February 1988.

U.S. Environmental Protection Agency. 1987a. Alternate Water Supply Operable Unit Feasibility Study, Final Draft. Galena Subsite, Cherokee County Site, Kansas. November 1987.

U.S. Environmental Protection Agency. 1987b. Additional Interim Guidance for FY 87 Records of Decision. Office of Solid Waste and Emergency Response. Directive 9355.0-21. July 1987.

U.S. Environmental Protection Agency. 1986. Phase 1 Remedial Investigation Report, Final Draft, Cherokee County Site, Galena Subsite. April 1986.

Appendix A
Technical Assistance for the Galena Subsite Mine Waste Characterization Program

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Appendix A.1
MINE WASTE MILLING/FLOTATION TREATABILITY STUDY

Section 1
INTRODUCTION

The Galena Subsite of the Cherokee County CERCLA Site is located in southeastern Kansas. The Galena Subsite is one of six subsites designated by the U.S. Environmental Protection Agency (EPA) within the Cherokee County Site. The Cherokee County Site is the Kansas portion of a lead-zinc mining area known as the Tri-State Mining District (including parts of Kansas, Missouri, and Oklahoma). Mining for lead and zinc in the Galena Subsite area began in the 1870s and ended in the 1920s.

Following remedial investigation (RI) activities conducted at the Galena Subsite in 1985 and 1986, an operable unit feasibility study (OUFS) conducted in 1987 and early 1988 (EPA, 1988a) identified a wide range of alternatives for remediating the groundwater and surface water systems. Subsequently, EPA developed a preferred remedy that consisted of the following:

1. Conventional milling/flotation of mine wastes deposited on the surface to remove lead, zinc, and cadmium and produce a "clean" tailing.

2. Divert surface waters to minimize recharge to the shallow groundwater in subsurface mineralized zone.

3. Recontour and regrade certain areas to minimize infiltration.

4. Inspect and remediate if necessary, wells and boreholes penetrating to the regional deep (Roubidoux) aquifer.

EPA felt that these actions would improve shallow groundwater and surface water quality. The milling/flotation process as outlined in the OUFS was assumed to be a simple, conventional bulk flotation process using a portable plant that could be taken to those areas in the subsite containing the bulk of the mine wastes.

In May 1988, EPA Region VII authorized additional technical assistance to better determine treatment parameters for removing and processing the surface mine wastes at the Galena Subsite (EPA, 1988b). The objectives of this work were to conduct additional sampling of mine wastes, and to conduct metallurgical test work to establish design and operating parameters for the removal of lead, zinc, and cadmium minerals from the surface mine wastes to be processed as outlined in the preferred remedy. Specific unit processes to be studied included crushing, grinding, and mineral flotation operations.

The field sampling program used an x-ray fluorescence (XRF) instrument to provide semiquantitative analyses for lead and zinc to focus the sampling on areas of more significant contamination. In addition to spot sampling of the mine rock wastes, some chat piles were sampled and measured with the XRF to better define metals content in the chat. XRF and subsequent laboratory wet chemistry analyses indicated that some chat material contains lead concentrations in excess of 1,000 milligrams per kilogram (mg/kg). As a result, the metallurgical test work program was altered to include chat as well as the other surface-deposited mine waste materials.

Upon completion of the metallurgical test work, it was determined that the processes required to remove metal contaminants from the mine wastes would be more complex than

originally envisioned in the OUFS. Preliminary flowsheets have been developed to reflect the additional information and data resulting from the test work. The preliminary capital and operating costs have also been developed (assuming the worst-case scenario) along with a list of recommendations for future work as submitted in the Hazen Report (Appendix A-3). This report presents the results of the sampling, metallurgical test work, and preliminary process flowsheet and cost estimate. This information is intended to aid Region VII in developing the final subsite remedy for the Record of Decision (ROD).

Section 2
BACKGROUND

Section 2
BACKGROUND

The Phase I RI (EPA, 1986) was conducted at the Galena Subsite during 1985 and 1986. The mining-related contaminants found at the subsite included lead, zinc, cadmium, and other metals. The mineralized mine wastes that were left on the surface and underground by mining activities were determined to be a major source of the metallic contaminants found in the surface water and groundwater systems. The mine wastes on the surface consist of a relatively fine material referred to locally as chat and coarser material referred to as bullrock and/or dump material. Chat, a product from early milling processes, typically contains lower metal concentrations than other mine wastes. The bullrock mine waste is primarily rock remaining after mine development (shaft and mine working excavation) that contains mineralization to varying degrees. Dump material is a mixture of materials including process wastes, bullrock, and uneconomic low-grade ores that were not processed at that time.

Subsequent evaluation of the health risks determined that human exposure to metallic contaminants may occur through direct ingestion of finer-sized mineralized mine wastes. Concentrations of dissolved metals found in the shallow groundwater and surface water system have been exacerbated as a result of mine drainage. Acid mine drainage results when water and oxygen react with iron sulfide minerals to form an acidic water. The acidic water, in turn, reacts with other sulfide minerals and results in the dissolution of metals. The dissolved metals can then be transported via the groundwater and surface water systems.

Two OUFSs have been conducted at the Galena Subsite. The first OUFS was the Alternative Water Supply OUFS (EPA, 1987),

completed in late 1987. This OUFS evaluated remedial alternatives developed specifically to protect the health of subsite residents that rely on metal-contaminated shallow groundwater for potable water. A remedy (providing another source of water) has been selected for this OU.

The second OUFS (EPA, 1988b), completed in February 1988, evaluated a range of alternatives that were developed to decrease the risks to public health and the environment posed by contaminants in the shallow aquifer and surface waters. The contaminated media identified in the OUFS includes surface mine wastes, subsurface mineralization in the mine workings, and the shallow groundwater and surface water systems within the subsite.

The alternatives evaluated in the OUFS were comprised of a wide range of contaminant source control and migration management remedial actions. The range of alternatives developed were then compared and evaluated against several criteria, including implementability; short- and long-term effectiveness; overall protection of human health and the environment; compliance with applicable or relevant and appropriate requirements (ARARs); reduction of toxicity, mobility, or volume; and cost.

Following completion of the second OUFS, EPA identified a preferred remedy. The preferred remedy (a modification of those evaluated in the OUFS) would remove and process surface mine wastes (to remove primarily lead, zinc, and cadmium) and produce a tailing of sufficient quality (low metal concentrations) to permit disposal in the mine working voids.

The preferred remedy also included surface water diversions, surface recontouring/revegetation, and deep aquifer well remediation measures. Surface water diversions would be implemented at specific areas to reduce the volume of recharge

to the shallow groundwater system, and, thereby, reduce the formation of acidic mine drainage and improve the quality of the shallow groundwater. Surface recontouring and revegetation are remedial actions designed to somewhat improve groundwater quality by reducing infiltration of precipitation to the groundwater system. Some wells that penetrate to the deep aquifer were identified in the OUFS as potential pathways for contamination to migrate from the contaminated shallow aquifer to the deep aquifer (a major regional source of drinking water). Well remediation measures may include inspection, plugging, or rehabilitation.

The removal and processing of the surface mine wastes, as identified in the preferred remedy, would use mining, trucking, crushing, grinding, and flotation to remove metals (primarily lead, zinc, and cadmium) and produce an acceptable tailing. For the purposes of developing this remedial action, the OUFS assumed that a portable plant using conventional bulk lead-zinc sulfide flotation circuits would produce tailing containing lead, zinc, and cadmium at sufficiently low concentration levels that would permit disposal in the mine workings.

Other key assumptions made during the feasibility study to develop and evaluate this remedial action included the following:

- An estimated 327,000 tons of exposed mine wastes, based on visual field observations made during the OUFS, would have to be mined, trucked, and processed to eliminate the primary source of metals from the surface mine wastes.

- The processing facility could be constructed as a portable facility, assuming a simple bulk sulfide flotation circuit would provide adequate

(85 percent) lead-zinc removal. It was determined that a portable plant with a capacity of 630 tons per day (tpd) would allow completion of the remedial action in about 2 years. The primary advantage of the portable facility is that relocation would minimize mine waste trucking costs. Because of the relatively short life requirement of the portable plant, it was assumed that used equipment would be adequate and available for plant construction.

EPA recognized that these assumptions made during the OUFS would have to be investigated further before selecting the remedy. The primary factors that impact the overall feasibility and cost of the preferred remedy are the volume and tonnage of material to be mined, trucked, and processed and the material's metallurgical characteristics. Therefore, additional sampling and metallurgical test work have been conducted to allow further development of the preferred remedy before the ROD.

The objective of the additional mine waste sampling was to collect reasonably representative samples of high- and low-grade mine wastes for metallurgical testing. The metallurgical test work would better determine treatment parameters for processing the mine waste. This would allow development of a preliminary process flowsheet and cost estimate.

During the fieldwork, a portable XRF, calibrated using known standards, was used to semiquantitatively identify lead and zinc concentrations in various types of mine wastes in the field. Lead and zinc concentrations in chat piles were measured with the XRF to verify assumptions that chat fines may pose a risk to public health through ingestion and possibly inhalation. XRF measurements on many chat samples determined

lead and zinc levels were considerably higher than expected, but were substantially lower than mine waste rock. Therefore, it was determined that the chat would be a good source of low-grade material for testing. A sample of chat from a pile that was analyzed as having a high lead content was collected to represent the low-grade sample required for metallurgical testing.

The chat sample was subsequently submitted for wet chemistry analyses. Analyses performed on the bulk sample and the plus and minus 80-mesh (180μ) screened fractions supported the semiquantitative measurements observed in the field with the portable XRF instrument. The minus 80-mesh fraction represents that portion most likely to be windblown and ingested. Table 2-1 gives the conversions between screen (mesh) size and metric particle size.

Because of the above observations, the metallurgical test work proceeded using the hand-picked mine waste sample to represent high-grade mine waste and the chat sample to represent the low-grade mine waste. Bench-scale tests were conducted on the various mine waste and chat samples to better define process requirements and metal recoveries.

Table 2-1
SCREEN MESH SIZE AND METRIC EQUIVALENTS

Unit		Unit	
Mesh Size	Metric Size	Mesh Size	Metric Size
5 in.	125 mm	No. 8	2.36 mm
4.24 in.	106 mm	No. 10	2.00 mm
4 in.	100 mm	No. 12	1.70 mm
3-1/2 in.	90 mm	No. 14	1.40 mm
3 in.	75 mm	No. 16	1.18 mm
2-1/2 in.	63 mm	No. 18	1.00 mm
2.12 in.	53 mm	No. 20	850 µm
2 in.	50 mm	No. 25	710 µm
1-3/4 in.	45 mm	No. 30	600 µm
1-1/2 in.	38.1 mm	No. 35	500 µm
1-1/4 in.	31.5 mm	No. 40	425 µm
1.06 in.	26.5 mm	No. 45	355 µm
1 in.	25.0 mm	No. 50	300 µm
7/8 in.	22.4 mm	No. 60	250 µm
3/4 in.	19.0 mm	No. 70	212 µm
5/8 in.	16.0 mm	No. 80	180 µm
0.530 in.	13.2 mm	No. 100	150 µm
1/2 in.	12.5 mm	No. 120	125 µm
7/14 in.	11.2 mm	No. 140	106 µm
3/8 in.	9.5 mm	No. 170	90 µm
5/16 in.	8.0 mm	No. 200	75 µm
0.265 in.	6.7 mm	No. 230	63 µm
1/4 in.	6.3 mm	No. 270	53 µm
No. 3-1/2	5.6 mm	No. 325	45 µm
No. 4	4.75 mm	No. 400	38 µm
No. 5	4.00 mm		
No. 6	3.35 mm		
No. 7	2.80 mm		

Source: ASTM Standard Test Sieves Series.

Section 3
TESTING PROGRAM

Section 3
TESTING PROGRAM

The preferred plan for the Galena Subsite was to metallurgically treat all the surface mine waste and produce a clean tailing that could be placed back into the mine voids. The metallurgical treatment that was assumed used conventional crushing, milling, and flotation processes to recover the lead and zinc minerals with an emphasis on producing a clean tailing material. Because this would be a short-term project lasting approximately 2 years, conventional process technologies were assumed. It was also assumed there would be a supply of used equipment available to construct the milling circuit to reduce capital costs.

To provide technical assistance to EPA for the preliminary design of the milling circuit, CH2M HILL developed a work plan that would provide additional information on critical design parameters. Parameters that would have to be determined include fineness of grind in the milling circuit, retention times, number of flotation stages, flotation reagents and amounts of additions, and metal recoveries and expected tailing quality.

CH2M HILL selected a subcontractor, following EPA guidelines, to perform the metallurgical test work on the mine waste material. The subcontractor chosen and approved by EPA for this work was Hazen Research of Denver, Colorado. Their report of the test work is attached as an appendix for reference.

SAMPLING

To provide sample material for the metallurgical testing, a field sampling event was organized and performed. The goal of this sampling program was to collect sufficient high- and low-grade material to perform a variety of metallurgical tests to determine the design parameters. To help in-field screening of the material, a portable XRF was used for field analysis. Because of the coarseness of the mineral grains in the host rock, the XRF did not work well in analyzing the high-grade material correctly and visual methods and hand picking had to be employed to collect this sample.

Standards used in calibrating the XRF included mine waste samples previously analyzed by the Contract Laboratory Program (CLP), a prepared ore sample with a known amount of lead and zinc, and specially prepared standards using silica sand and known quantities of lead and zinc added as chemical compounds. The prepared ore sample contained 5,830 mg/kg lead and 425 mg/kg zinc. The specially prepared standards contained 100 mg/kg lead and 1,000 mg/kg zinc.

EPA requested CH2M HILL to collect several samples of chat while collecting other mine wastes and to analyze the bulk and minus 80-mesh fraction for lead and zinc content. The minus 80-mesh fraction represents the potential size fraction that may be windblown. Using the XRF, a large chat pile on the south side of Hell's Half Acre, No. 07-01 (Zone 7, Pile 1), was chosen for the low-grade sample material. This chat pile was field analyzed for lead near the surface and at depths of 1 to 2 feet. The analysis results for the surface and deeper samples were 600 to 700 mg/kg and between 1,600 and 2,500 mg/kg, respectively. In addition to the low- and high-grade samples, several samples of chat were collected. Both the bulk chat samples and the minus

80-mesh (180 μ) fraction of the chat samples were analyzed for lead and zinc content.

PRELIMINARY CHAT TESTING

Two chat samples (Nos. 07-01 and 01-02) were sent to a laboratory for further analysis. Sample 01-02 was collected southwest of Galena near Shoal Creek. The samples were split into four equal parts; one part was dry screened at 80 mesh, and one part was used for bulk sample analysis. The other two parts were saved for future work. Dry screening was done to prevent any loss of soluble lead from the sample. During wet screening, soluble lead compounds would be washed from the particles. Although this water was saved and analyzed for lead, it was impossible to determine which size fraction it came from. This initial screening test showed the minus 80-mesh fraction (less than 180 microns) of the samples contained lead concentrations between 200 and 23,000 mg/kg and zinc concentrations between 800 and 63,000 mg/kg.

To determine if the lead and zinc may be more concentrated in other size fractions, individual wet screen analyses were conducted on Samples 01-02 and 07-01, which were split from the bulk sample collected for metallurgical test work (see Tables 3-1 and 3-2). The samples were wet screened, using 17 individual screens, between plus 3 mesh and 400 mesh. Each size fraction was saved, weighed, and analyzed for lead, zinc, cadmium, and arsenic. The water used in the screening was also analyzed for lead and showed no dissolved lead. As can be seen from Tables 3-1 and 3-2, the tests showed that as much as 72 percent of the lead was contained in only 12 percent of the material. This work indicated that most of the lead may be concentrated by wet screening the chat to isolate the fine-size fractions.

Table 3-1
WET SCREEN ANALYSES RESULTS--SAMPLE 01-02

	Product	Weight		Analyses, mg/kg			Distribution, Percent		
		G	Percent	Pb	Zn	Cd	Pb	Zn	Cd
1.	+3 mesh	5.78	1.63	110	590	0	0.4	1.9	0.0
2.	-3 + 4 mesh	63.01	17.75	100	460	3	3.9	16.3	25.5
3.	-4 + 6 mesh	67.88	19.12	100	450	2	4.2	17.1	18.3
4.	-6 + 6 mesh	10.47	2.95	100	240	0	0.6	1.4	0.0
5.	-9 + 10 mesh	94.54	26.63	200	190	0	11.7	10.1	0.0
6.	10 + 14 mesh	17.26	4.86	150	230	0	1.6	2.2	0.0
7.	-14 + 20 mesh	24.62	6.93	190	320	2	2.9	4.4	6.6
8.	-20 + 28 mesh	3.59	1.01	200	730	3	0.4	1.5	1.5
9.	-28 + 35 mesh	21.84	6.15	170	820	3	2.3	10.1	8.8
10.	-35 + 48 mesh	6.07	1.71	240	1,100	7	0.9	3.7	5.7
11.	-48 + 65 mesh	6.47	1.82	230	1,100	7	0.9	4.0	6.1
12.	-65 + 100 mesh	4.23	1.19	250	1,300	8	0.7	3.1	4.6
13.	-100 + 150 mesh	2.6	0.73	460	1,400	9	0.7	2.0	3.2
14.	-150 + 200 mesh	1.24	0.35	300	1,400	9	0.2	1.0	1.5
15.	-200 + 270 mesh	1.51	0.43	320	1,300	8	0.3	1.1	1.6
16.	-270 + 400 mesh	0.61	0.17	360	1,200	9	0.1	0.4	0.7
17.	-400 mesh	23.34	6.57	4,700	1,500	5	68.2	19.7	15.9
18.	Water	1.800 L	--	<0.1	0	0.0	0.0	0.0	0.0
Head, Calc'd		355.06	100.00	454.4	501.8	2.1	100.0	100.0	100.0
Head, Direct				N/A	N/A	N/A			

Notes: Calculated head as percent 0.045%; 0.050%; and <001%.

Table 3-2
WET SCREEN ANALYSES RESULTS--SAMPLE 07-01

	Product	Weight		Analyses, mg/kg			Distribution, Percent		
		G	Percent	Pb	Zn	Cd	Pb	Zn	Cd
1.	+3 mesh	0	0.00				0.0	0.0	0.0
2.	-3 + 4 mesh	9.76	2.47	210	930	3	1.2	1.4	1.3
3.	-4 + 6 mesh	164.13	41.53	240	1,200	4	23.6	30.5	29.4
4.	-6 + 6 mesh	131.61	33.30	250	780	3	19.7	15.9	17.7
5.	-9 + 10 mesh	40.51	10.25	230	460	2	5.6	2.9	3.6
6.	10 + 14 mesh	10.27	2.60	240	780	5	1.5	1.2	2.3
7.	-14 + 20 mesh	8.88	2.25	280	2,000	12	1.5	2.8	4.8
8.	-20 + 28 mesh	2.64	0.67	330	7,400	35	0.5	3.0	4.1
9.	-28 + 35 mesh	3.63	0.92	350	14,000	48	0.8	7.9	7.8
10.	-35 + 48 mesh	1.50	0.38	350	20,000	65	0.3	4.6	4.4
11.	-48 + 65 mesh	1.31	0.33	400	22,000	70	0.3	4.5	4.1
12.	-65 + 100 mesh	0.72	0.18	400	21,000	60	0.2	2.3	1.9
13.	-100 + 150 mesh	0.3	0.08	600	34,000	70	0.1	1.6	0.9
14.	-150 + 200 mesh	0.14	0.04	700	46,000	20	0.1	1.0	0.1
15.	-200 + 270 mesh	0.18	0.05	0	100,000	0	0.0	2.8	0.0
16.	-270 + 400 mesh	0.05	0.01	0	0	0	0.0	0.0	0.0
17.	-400 mesh	19.55	4.95	3,800	5,800	20	44.6	17.6	17.6
18.	Water	1.356 L	--	<0.1	0	0	0.0	0.0	0.0
Head, Calc'd		395.18	100.00	421.7	1,633.2	5.7	100.0	100.0	100.0
Head, Direct				N/A	N/A	N/A			

Notes: Calculated head as percent 0.042%; 0.163%; and <001%.

Bulk sample analysis during metallurgical testing showed lead concentrations lower than encountered during the field sample collection program where XRF was used to characterize the mine wastes. These higher XRF analyses are attributed to a particle coating that is not fully understood at this time. The chemical sample analyses of the two samples showed lead concentrations of 454 mg/kg for 01-02 and 422 mg/kg for 07-01. The minus 400-mesh fraction of these two samples contained lead concentrations of 3,800 and 4,700 mg/kg.

To determine the most probable chemical analysis of the chat samples being tested, two chat samples were prepared for special analysis. The samples, 01-02-01 and 07-01-02, were crushed to minus 10 mesh, blended, and split into four equal parts using a riffle-type splitter. One part from each sample was sent to three independent laboratories for total lead and zinc analysis. The average analysis is reported in Table 3-3. These analyses indicate concentrations for lead and zinc much lower than the initial field analyses provided.

METALLURGY

Two 5-gallon buckets of mine waste samples, one of high-grade and one of low-grade material, were delivered to Hazen Research on July 11, 1988. Hazen personnel received the samples and prepared them for analysis and test work by crushing to minus 6 mesh, blending, and splitting into 2-kg samples. The prepared samples were bagged in plastic and stored in a freezer to minimize the possibility of surface oxidation. Bulk chemical analyses of the samples used in metallurgical testing are listed in Table 3-3.

To provide information on sizing of crushing and grinding equipment, Bond Work Indices were derived for each of the feed samples. The Bond Work Indices of the high- and low-

Table 3-3
BULK MATERIAL ANALYSIS RESULTS

Description	Identification No.	Pb (mg/kg)	Zn (mg/kg)
High Grade Mine Waste	--	6,340	61,800
Low Grade Mine Waste (chat)	07-01-01	730	27,000
Chat Analysis (second sampling period)	07-01-02	520	2,010
Chat Analysis (used in screening tests)	01-02-01	550	400

Notes: Pb = lead.
Zn = zinc.

grade samples were 20.2 and 18.8 kwh/t, respectively, to produce a product size of 80 percent minus 118 microns, approximately 150 mesh. This means the material is very hard and requires considerable energy to grind it to the required size for flotation and metals removal.

FLOTATION

Both types of samples were readily amenable to metals removal by flotation methods. Good quality lead and zinc sulfide concentrates were produced easily using conventional reagents and a simple rougher-single cleaner stage circuit. The lead sulfide concentrate averaged 51 percent lead, and the zinc sulfide concentrate averaged 58 percent zinc. The results of the flotation test work are summarized in Table 3-4.

Oxidized lead and zinc minerals were recovered in additional staged flotation circuits subsequently using chelating agents, polyethylene oxide, and dodecyl mercaptan to

Table 3-4
FLOTATION TESTING SUMMARY

Test No.	Conditions	Pb Circuit: Oxide Ro Conc., Percent Wt.	Pb Circuit: Oxide Ro Conc., Percent Pb	Pb Circuit: Oxide Ro Conc., Percent Rec.	Pb Circuit: Ro. Conc., Percent Wt.	Pb Circuit: Ro. Conc., Percent Pb	Pb Circuit: Ro. Conc., Percent Rec.	Zn Circuit: Oxide Ro Conc., Percent Wt.	Zn Circuit: Oxide Ro Conc., Percent Zn	Zn Circuit: Oxide Ro Conc., Percent Rec.	Zn Circuit: Ro. Conc., Percent Wt.	Zn Circuit: Ro. Conc., Percent Zn	Zn Circuit: Ro. Conc., Percent Rec.	Ro. Tail., Percent Pb	Ro. Tail., Percent Zn	Total Rec., Percent Pb	Total Rec., Percent Zn
High Grade Sample																	
1.	Conventional	0.5	51.0	39.6	1.9	18.0	52.2	11.9	57.0	97.0	14.1	48.1	97.3	0.310	0.070	60.7	99.2
4.	Oxide scav: sulfidized	3.2	3.5	16.8	2.5	14.5	54.9	3.2	0.48	0.2	15.4	43.5	96.8	0.160	0.059	80.9	99.3
7.	Oxide: 81469C, PED, sulfdized	22.5	0.8	27.6	3.0	12.0	56.0	22.5	0.22	0.7	14.4	48.0	95.8	0.110	0.045	89.8	99.6
10.	T7, finest PG	22.1	1.0	34.0	2.7	12.0	51.0	22.1	0.35	1.1	15.9	41.5	96.3	0.055	0.020	94.8	99.8
11.	T7, coarsest PG	13.1	1.0	19.1	1.9	20.5	55.6	13.1	0.32	0.5	16.9	43.5	96.7	0.185	0.055	82.0	99.5
19.	Sulfidized at start of oxide	47.1	0.4	28.9	2.7	14.0	58.9	47.1	0.14	0.9	12.9	50.0	92.5	0.115	0.025	93.3	99.9
20.	Sulfidized + chelate together	34.0	0.5	24.8	3.4	12.5	62.7	34.0	0.21	1.0	13.3	50.5	91.0	0.115	0.029	91.6	99.8
21.	Sulfidized + R-6410 together	5.7	1.1	9.4	2.8	13.5	57.6	5.7	0.34	0.3	16.0	42.5	91.7	0.215	0.040	75.6	99.6
22.	Sulfidized + PED together	13.7	0.5	10.4	4.9	7.5	62.3	13.7	0.23	0.5	13.7	46.0	89.5	0.185	0.029	78.7	99.7
Low Grade Samples																	
2.	Conventional	0.1	3.6	5.9	1.3	0.7	11.1	1.6	59.5	35.5	4.4	24.3	39.5	0.065	1.65	18.8	42.5
3.	Oxide scav: sulfidized	5.3	0.31	23.4	1.1	0.6	8.6	5.3	5.0	9.2	5.7	20.5	41.6	0.044	1.50	44.8	53.6
9.	T8, finest PG	46.1	0.07	47.0	9.7	0.2	27.0	46.1	1.9	33.1	6.6	16.0	39.4	0.025	0.88	85.9	87.7
13.	T9, coarsest PG	11.5	0.22	33.5	1.7	0.6	12.9	11.5	4.0	15.8	4.4	26.5	40.3	0.039	1.40	57.8	60.5
14.	-20m classif. chat 07-01	15.7	0.4	28.4	9.2	0.3	12.9	15.7	0.3	4.3	12.6	5.9	80.4	0.095	0.05	69.9	96.8
15.	71.5% high grade and 28.5% -28m chat 01-02	9.6	0.5	8.8	3.4	7.8	48.8	9.6	0.4	0.7	11.3	42.5	93.7	0.250	0.07	65.3	99.0

condition the pulps and recover the oxide minerals. These additional stages are necessary to remove elevated levels of lead for the tarting to meet EP Toxicity test criteria.

The high-grade sample treated with conventional flotation methods removed 60.7 percent of the lead and 99.2 percent of the zinc. Using the special chelating agents to enhance lead oxide flotation, lead removal was increased to 93 to 94 percent, and zinc removal was increased to 99.8 percent. To produce this level of removals, a product grind size of approximately 90 percent minus 200 mesh would be needed.

In testing the low-grade chat sample, conventional flotation methods produced poor removals of 18.8 percent for lead and 42.5 percent for zinc. By using the special chelating agents to enhance oxide lead flotation and a finer grind of 100 percent minus 200 mesh, a lead recovery of 85.9 percent and a zinc recovery of 87.7 percent were achieved.

The majority of the lead and zinc content in the chat is contained in the minus 20-mesh fraction. This size fraction represents about 20 percent of the available material. To test the oxide lead flotation procedure further, a sample of minus 20-mesh chat material was separated by screening and subjected to grinding and flotation tests. The procedure removed 69.9 percent of the lead and 96.8 percent of the zinc. The fact that the sample was only ground to 80 percent minus 200 mesh likely accounts for the reduced removal of lead. Additional test work is needed in this area if the procedure is to be used in the remediations.

To determine if metal recovery problems experienced in chat testing occurred when processed with high-grade mine waste

and mixed with chat fines produced from screening, a separate flotation test was performed. A sample containing 71.5 percent high-grade material and 28.5 percent low-grade material fines (01-02-01) was mixed and floated using the parameters in Test No. 10. An overall lead removal of 65 percent was achieved, somewhat lower than the other tests. The zinc removal was 99 percent. Additional test work is needed in this area if the procedure is to be used in the remediations. It may be possible to introduce the chat fines to the processing circuit at a different location to enhance performance.

Several additional liberation studies were done to determine the correct grind size to separate the lead and zinc minerals from the gangue material. These tests were accomplished at grind sizes ranging from 70 to 100 percent minus 200 mesh. The results clearly show that metal removal is increased as the material is ground finer; however, materials must not be overground to cause sliming, since sliming can decrease metal recovery. Grinding also has an economic consideration; costs increase dramatically as the grind size decreases. Table 3-5 shows the correlation of grind size and lead removal.

Table 3-5
CORRELATION OF GRIND SIZE AND LEAD REMOVAL

Test No.	Percent Passing 200 Mesh	Percent Lead Recovered
7 (high grade)	83.5	89.8
10 (high grade)	96.3	94.8
11 (high grade)	67.3	82.0
8 (low grade)	85.0	77.3
9 (low grade)	100.0	85.9
13 (low grade)	70.0	57.8

TAILING QUALITY

To provide a measure of the quality of the tailing being produced in the test work, EP toxicity testing was done on selected tailing samples. The results of these EP Toxicity tests are provided in Table 3-6. The analyses of the tailing material producing these results are also reported in Table 3-6.

Because of the low zinc removals in Test Nos. 2, 12, 13, and 9, and since cadmium is typically present with zinc, cadmium concentrations were greater than the EP toxicity standards in the tailing EP Toxicity tests. Lead contents in Test Nos. 1, 7r, 11, and 7 were greater than the EP toxicity standards because of low lead removal. Tests having a tailing lead content of less than 700 mg/kg (0.07 percent) will likely produce results less than the EP Toxicity tests. Zinc is not one of the criteria in the EP Toxicity tests.

Table 3-6
EP TOXICITY TESTS

EP Toxicity Standards (mg/kg)	Elements	Flotation Test Number								
		1	2	7r	10	11	12	13	9	7
5	As	ND	ND	ND	ND	ND	ND	ND	ND	0.001
100	Ba	0.39	0.2	0.27	0.48	0.28	1	0.1	0.18	0.37
1	Cd	0.06	1.7	0.33	0.02	0.03	1.6	1	0.98	0.04
5	Cr	0.06	ND	ND	ND	ND	ND	ND	0.04	0.02
5	Pb	25	1	8.1	2.9	46	3	1.3	0.4	0.02
0.2	Hg	ND	ND	ND	ND	ND	ND	ND	ND	ND
1	Se	ND	ND	ND	ND	ND	ND	ND	ND	ND
5	Ag	ND	ND	ND	ND	ND	ND	ND	ND	ND
Tailing Analyses										
mg/kg	Pb	3,100	600	1,100	500	1,800		400	200	1,100

Notes: ND = not detected.
Test 7r is a repeat of Test 7 using a new sample split.

PROCESS WATER ANALYSIS

The water used in grinding and flotation in four tests, which represented all the testing parameters, was saved and analyzed for lead, zinc, copper, and arsenic. This was done to determine if water treatment would be necessary and if a buildup of these metals would occur during the milling process since water recycle would be required. Table 3-7 shows the results of these analyses.

The analysis results indicate that lead may be a problem in water that is discharged to the environment. A treatment plant using lime softening to produce heavy metal hydroxides may be necessary to reduce lead concentrations. Lime softening and filtration should lower the metal concentrations. This type of treatment would produce a sludge containing lime and metal hydroxides.

Table 3-7
PROCESS WATER ANALYSIS

Flotation Test No.	Mg/l			
	Cu	Pb	Zn	As
10	<0.1	0.1	<0.1	<0.2
11	0.3	7.4	1.5	<0.2
12	1.7	0.3	2.4	<0.2
13	6.0	0.1	1.7	<0.2

Notes: Cu = copper.
Pb = lead.
Zn = zinc.
As = arsenic.

Disposal of this sludge, though a small quantity, may possibly have to be handled as a hazardous substance.

Section 4
PROCESS DESCRIPTIONS AND FLOWSHEET

Section 4
PROCESS DESCRIPTIONS AND FLOWSHEET

The selected alternative was initially planned for a movable milling plant. As envisioned, this plant would be relocated periodically to locations near large volumes of mine waste. With the increased complexity of the milling process caused by the addition of the oxide flotation circuit, and increased capacity and size, it has been necessary to reconsider this plan. A cost analysis provided additional information on the costs, which are discussed in the next section.

The cost of relocating the expanded plant once during the plant life would exceed the cost of transporting the mine waste material to a central plant. The location determined to be the most cost-effective plant location is near Hell's Half Acre. This location was chosen for the following reasons:

1. It contains enough mine voids to hold all the tailing material produced during the milling operation.

2. Groundwater is likely available that can be used as process water in the milling operation.

3. It is reasonably centrally located in the subsite and itself contains a major portion of the mine waste that is to be treated, reducing that portion of the mine waste haulage costs.

Areas north of Galena and adjacent to Hell's Half Acre could possibly be used as site locations for placement of the chat screening and washing plant and chat stockpiles.

MINING

The mining sequences would be planned separately for each general area of mine waste location. In many locations, the waste rock could be transferred directly to haul trucks using loaders and backhoes. However, in areas like Hell's Half Acre, mining would be very difficult because of the potentially unstable ground conditions. These conditions include large subsidence areas or areas that are bridged over mine voids. Large equipment would not be able to work safely in these conditions; therefore, smaller equipment and conveyor transfer may be necessary.

A mine waste reconnaissance team may also be necessary to survey the waste rock locations and determine the approximate grade of the materials to assist in supplying a constant head grade to the mill. This is important to maintain suitable metal recoveries. Certain areas contain large amounts of trash mixed with mine wastes. This trash will have to be removed before mining activities begin. Haul roads to certain areas would have to be constructed to properly access the mine wastes. Ten-cubic-yard capacity trucks are assumed to be used for all hauling.

For cost estimating purposes, different mining scenarios were considered for several mining sites. These scenarios determined approximate equipment efficiencies, truck and loader cycle times, and haul distances. The difficulty of mining at each site and which equipment would be most appropriate at each location was considered. Block flow diagrams were developed showing the flow of material through each phase of the mining operation. Mining production tonnages were estimated to supply the mill with an adequate supply of feed materials. Average costs estimated for mining and transporting mine waste to the centrally located

mill are assumed to be $4.00 per ton and $1.50 per ton, respectively, totalling $5.50 per ton. These costs include road construction, maintenance, safety control, and dust control.

During mining activities, the mine waste rock would be hauled to a stockpile area next to the plant. At this location, the rock would be blended to supply the plant with a more constant head grade material.

Chat material would also be hauled to a chat stockpile area near the chat treatment area. No difficulties are foreseen in mining or hauling the chat. The cost to mine and load the chat is estimated at $1.75 per ton, and to haul to a centrally located chat screening plant would be $1.50 per ton, totalling $3.25 per ton. Mining and hauling would most likely be subcontracted to companies equipped to perform these activities.

CRUSHING AND SCREENING

Rock from the mine waste stockpile (estimated to be a maximum of 24 inches) would be fed to a primary jaw crusher, which is capable of crushing to approximately minus 2 inches. Between the feed point and the jaw crusher, an opportunity would be provided to remove metal objects and trash that could cause trouble in the screening circuit. The product from the jaw crusher would go to a screening plant, in closed circuit with a cone crusher, to further reduce the size of the rock to minus 3/8 inch. The screening plant would separate a product size material of minus 3/8 inch and send the larger material back to the cone crusher for further size reduction. The product size material would go to a fine ore storage bin using a conveyor system.

The crushing circuit would be designed to operate for one shift a day to supply enough fine ore to the milling circuit for a 24-hour operation.

GRINDING AND CLASSIFICATION

The grinding and classification section would be operated on a 7-day, 3-shift-per-day operation based on a 700-tpd feed rate. This would mill the 300,000 to 500,000 tons of mine waste in about a 2-year period. This capacity would nearly double that of the mill identified in the OUFS.

Fine ore from the fine ore storage bin would be fed to the ball mill along with some fine chat, using a conveyor and weightometer system, at an average feed rate of 31 tons per hour (tph). Water would also be fed to the mill at the point to provide the proper slurry consistency for grinding. The ball mill circuit would operate in closed circuit with a hydrocyclone for product size control and classification. The overflow at the opposite end of the ball mill would discharge into a sump and be pumped to the hydrocyclone. The product material from the hydrocyclone (overflow), approximately 85 to 90 percent minus 200 mesh, would pass to the flotation circuit. The coarser size (underflow) would pass back into the ball mill for additional grinding.

FLOTATION

The flotation circuit is separated into the following different parts:

- A lead sulfide circuit
- A zinc sulfide circuit
- A three-stage metal oxide circuit

The cyclone overflow discharges to a conditioning tank where flotation reagents, specific for sulfide flotation, are added. From the conditioning tank, the slurry flows through the flotation cell band where the lead is removed. The underflow is conditioned a second time and is then sent to the zinc circuit where zinc sulfides are removed.

These first-stage concentrates report each to cleaning stages where cleaner lead and zinc concentrates are produced. The sulfide circuits remove approximately 60 to 65 percent of the lead and 98 percent of the zinc from the feed material. The tailing from the sulfide circuits pass into the oxide conditioning tank where the oxide reagents are added. From the oxide conditioning tank, the slurry flows through three to five stages of oxide flotation to remove the lead and zinc oxides from the feed.

Because of the nature of the oxide minerals, it would be impossible to collect separate lead and zinc concentrates; therefore, a mixed bulk concentrate would be collected. The feed rate to the flotation circuit may vary depending on the amount of chat fines being added to the circuit.

The circuit would be designed into two separate banks, each being able to treat one-half of the feed stream. If the feed to the mill is reduced for any reason, one bank can be shut down. This will permit a more efficient operation of the flotation circuit and allow for tonnage variations. The oxide circuit removes an additional 28 to 34 percent of the lead and 1 percent of the zinc from the feed. Each of the sulfide concentrates would be filtered and drummed for sale to a lead smelter. The oxide concentrate would be filtered and drummed for sale or disposal.

The tailing leaving the flotation circuits would be thickened from approximately 25 to 30 percent solids to near 60 percent solids for disposal into the mine voids. Water would be reclaimed from the thickener and used as process water in the milling operation. If the water contained in the tailing ready for disposal contains significant levels of lead, the tailing could be washed with process make-up water in a second thickener before disposal to reduce the lead levels. Any bleed water to be disposed of that would not meet disposal criteria would be lime softened to remove heavy metals. The tailing could be checked in-line with an XRF to determine metals concentrations before disposal. If not, a recycle from the tailing thickener back to the oxide circuit could be included to further remove more lead.

CHAT TREATMENT

Several points were considered and evaluated to produce the most cost-effective treatment process for the chat material. Access to individual chat piles could be difficult. Some chat may have low metals levels and not meet the treatment criteria. Therefore, these piles would not need to be processed. A method to sample and screen each pile on a go/no-go basis for treatment may be difficult. Either method must truck haul chat to a central location or haul a portable treatment plant to each chat site location for processing. A portable plant would allow the cleaned chat to be left behind in the same location while hauling the dirty fines to the mill. A central plant location would allow the dirty fines to be directly fed to a mill facility while the clean chat could be stocked in a central stockpile for sale or direct use.

To determine the most cost-effective method of treating the chat, a cost analysis was completed. The following variables were considered:

- Mining costs
- Transportation costs of the chat versus that of the plant
- Operating efficiency caused by downtime
- Equipment efficiency of fixed versus portable equipment

Block flow diagrams were developed showing the flow of material through each of the unit processes. The conclusion is that a central plant is considerably more cost-effective than a portable plant.

Chat from the dirty chat stockpile would be fed to a vibratory screening circuit where the minus 35-mesh fines would be separated, using two-stage wet screening, from the coarse fraction. The minus 35-mesh fraction represents approximately 20 percent of the bulk and 60 to 70 percent of the contained lead and zinc. The coarse fraction would be transported to a clean chat stockpile for storage or subsequent use, while the fine fraction would be kept in slurry form for feeding into the ball mill circuit. Lead and zinc in the chat would be removed in the flotation circuit. A block flowsheet of the chat and mine waste processes are shown in Figure 4-1.

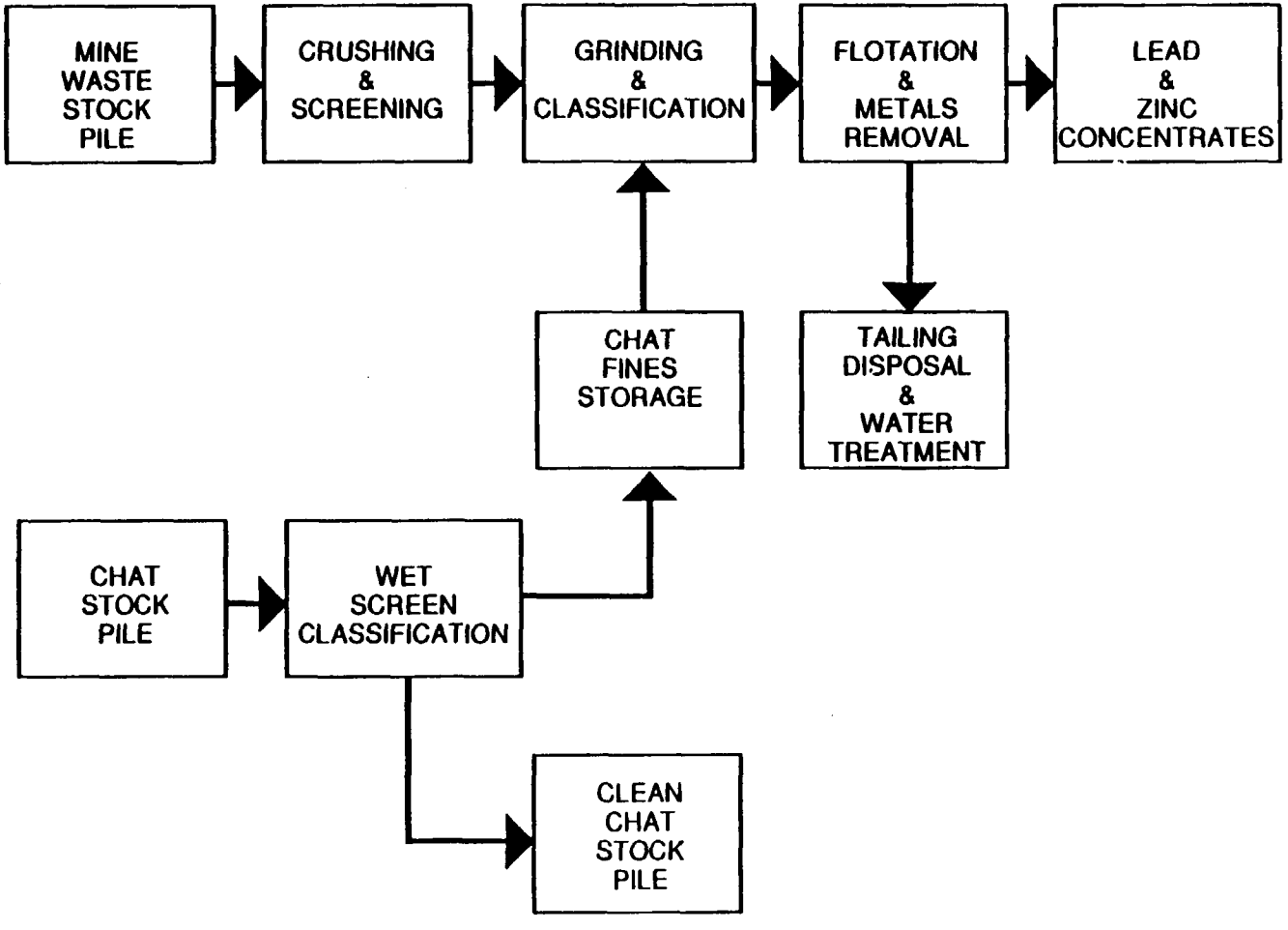

FIGURE 4-1
CHAT AND MINE WASTE UNIT PROCESSES
GALENA SUBSITE
CHEROKEE COUNTY SITE
DEN67900

Section 5
COST ESTIMATES

Section 5
COST ESTIMATES

To review the OUFS processes, only 327,000 tons of mine wastes were to be milled using a small, portable plant having a simple bulk sulfide flotation circuit. The capital costs were estimated at $810,000, haulage at $49,000, operating costs for the 2-year period at $1,296,000, and no mining costs were included, thus totalling $2,155,000.

Field and laboratory work performed for this report have demonstrated the following:

- Mine waste rock to be treated may increase to 500,000 tons because of a supplemental volume estimate by the potentially responsible parties (PRPs).

- The Bond work index is very hard, thus requiring significantly more power and consumables to grind to size to liberate the metals of concern.

- The metallurgy is much more complex, requiring a 3- to 5-stage flotation circuit rather than the simple bulk flotation system assumed earlier.

- The topsize of mine waste was determined to be considerably larger, thus increasing the size of the crushing circuit.

- Several samples of chat were determined to contain lead and zinc concentrations much higher than anticipated. Therefore, as much as one-half of the chat will be processed, which is more than initially planned.

- The lead and zinc contained in the chat are concentrated in the fine fraction that can be separated by wet screening.

- Test work indicated that it may be possible to process the chat fines in conjunction with the mine wastes to remove lead and zinc. Additional work is needed in this area.

- Up to 1 million tons of chat may be contaminated. More fieldwork must be conducted to finalize this determination.

Based on these differences identified during this study, new capital and operating cost estimates were prepared to assist EPA in making a decision on the final remedy for the Galena Subsite. To compare with the FS, the first estimate is the new flowsheet required to mill 327,000 tons of mine waste based on larger size, higher Bond work index, and far more complex flotation metallurgy. Mining and haulage costs have been updated to reflect haul distances to a centrally located mill. Comparison is shown with the cost estimate in the FS in the following Table 5-1.

Table 5-1
HAULING COSTS

	FS Estimate	Revised Estimate
Capital Costs	$ 810,000	$1,500,000
Haul Costs	49,000 @ 15/ton	450,000 @ $1.50/ton
Mining Costs	Not included	1,200,000 @ $4.00/ton
Operating Costs	1,296,000	3,470,000
TOTALS	$2,155,000	$6,620,000

If the mine waste tonnage increased to 500,000 tons to be milled and chat quantity to 1,000,000 tons to be screened and milled, the costs would increase to the following levels:

Capital Cost	$ 2,209,000
Haul Costs	2,250,000
Mining Costs	3,750,000
Operating Costs	8,100,000
TOTAL	$16,309,000

Because of the large volume increases into the treatment process, the impact is clearly seen. More work must be conducted on the action levels of metals content for the chat; furthermore, spot samples must be taken to see how much chat must be processed to meet the action levels required. Additionally, the costs of ancillary operations, such as fencing, surface recontouring, and surface water diversion that could increase costs even further were not updated. If less chat were to be treated, capital costs and operating costs would decrease.

The cost estimates have been prepared for guidance in project evaluation and implementation from the information available at the time of the estimate. The final costs of the project would depend on actual labor and material costs, actual site conditions, productivity, competitive market conditions, final project scope and schedule, the firm selected for final engineering design, and other variable factors. As a result, the final project costs would vary from the estimate presented here. Because of these factors, funding needs must be carefully reviewed before making specific financial decisions or establishing final budgets.

Details of the cost estimates are included in the appendix.

REFERENCES

REFERENCES

U.S. EPA. Phase I Remedial Investigation Report, Final Draft, Cherokee County Site, Galena Subsite. April 1986.

U.S. EPA. Groundwater and Surface Water Operable Unit Feasibility Study, Final Draft, Cherokee County Site, Galena Subsite. February 26, 1988a.

U.S. EPA. Work Plan, Technical Assistance for the Galena Subsite Mine Waste Characterization, Cherokee County Site. May 1988b.

U.S. EPA. Alternative Water Supply Operable Unit Feasibility Study, Galena Subsite, Cherokee County Site. November 4, 1987.

Appendix A.2
COST ESTIMATE

Appendix A.2
COST ESTIMATE

Haulage Cost (chat or mine waste) including road construction, maintenance, safety dust control average 5-mile haul dist. Subcontractor basis--including O&P.	$ 1.50/ton
Mining Costs	
Mine Waste--Move, stockpile, load	$ 4.00/ton
Chat	$ 1.75/ton
Operating Costs	
Personnel--total 36	$ 5.10/ton of mill feed
Some 5 day/week--1 shift/day	
Balance 7 day/week--3 shifts/day	
Process Supplies	$ 4.75/ton
Utilities	$ 1.38/ton
Maintenance	$ 0.33/ton
TOTAL	$11.56/ton of mill feed
Capital Costs	
Mine Waste Store/Crush Plant[a]	$ 658,000
Chat Handling and Screen Plant	$ 551,000
Grind/Flotation and Waste Disposal	$1,000,000
	$2,209,000

[a]Assumes use of some used equipment.

Appendix A.3
METALLURGICAL REPORTS

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CHEROKEE COUNTY GALENA SUBSITE

Metallurgical Studies

Volume 1

March 1, 1989

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CHEROKEE COUNTY
GALENA SUBSITE

Metallurgical Studies

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INTRODUCTION

On June 7, 1988, CH_2M Hill authorized Hazen Research, Inc. (Hazen) to conduct a metallurgical test program on surface waste materials from the Tri-State Mining District, located in the bordering areas of Kansas, Oklahoma, and Missouri. Historically, this area has been a major site for lead and zinc production. The Cherokee County site is specifically in the Kansas portion of the area, and the Galena Subsite is part of the Cherokee County EPA Superfund cleanup effort.

Extensive mineral wastes have been deposited throughout the area from mining and milling processes, and smelter operations. The wastes are composed predominantly of the sulfide minerals galena (PbS) and sphalerite (ZnS).

Cadmium in association with the zinc is also identified as a problem. There are also oxide forms of lead and zinc produced from natural weathering of the materials. These mixed sulfide/oxide wastes are continuing to weather, resulting in toxic leachates of lead and zinc which are contaminating surface runoffs and groundwater.

The Superfund Cleanup process alternative preferred by EPA for these mine wastes is to remill the materials by a flotation process to produce non-toxic tailings which will be disposed of in the mine excavations. The recovered lead and zinc concentrates may be sold to appropriate metals processors for commercial use and, hopefully, to pay for a portion of the reprocessing expenses.

The primary objective of the test work was to develop a flotation process which would produce a final tailing which would contain metals in concentrations which are consistent with current EP Toxicity standards.

This report summarizes, interprets, and presents all data developed for the project. Details of test procedures, metallurgical balances, etc., are shown in the appendix to this report.

SUMMARY

A list of the test samples provided by CH_2M Hill and pertinent head analyses is shown below. A detailed description of the samples is provided in the following "Samples" section, and qualified head analyses are described in Table 1.

HRI No.	Description	Client Identification	Assays Pb, %	Zn, %	Cd, ppm
39467	High Grade Ore	-	0.634	6.18	380
39468	Low Grade Ore	07-01-01	0.075	2.70	95
39849-1	Low Grade Chat	01-02	0.053	0.035	2
39849-2	Low Grade Chat	07-01-02	0.043	0.154	4

The feed materials responded well to a differential metal sulfides and oxides flotation treatment system. The conceptual milling flowsheet is shown in Figure 1, page 15 of this report. Saleable grade lead and zinc sulfide concentrates were readily produced, using conventional reagents and simple rougher-single cleaner stages. Oxidized lead and zinc minerals were recovered from the sulfide rougher tailings using chelating, selective flocculating, and sulfidizing techniques and dodecyl mercaptan as collector. The best flotation results (for the specified feed and flotation conditions) are summarized below.

Test No.	Sample	Flotation Conditions	Grind Fineness % -200m	Total Recovery, % Pb	Zn	Final Tailing, % Pb	Zn
10	High Grade Ore	Sulfide/oxide	96.3	94.8	99.8	0.055	0.020
1	High Grade Ore	Conv. sulfide	83.5	60.7	99.2	0.310	0.070
9	Low Grade Ore	Sulfide/oxide	100	85.9	87.7	0.025	0.88
2	Low Grade Ore	Conv. sulfide	85.3	18.8	42.5	0.065	1.65
14	Classified (-20M) Low Grade Chat	Sulfide/oxide	80	69.9	96.8	0.095	0.047

Typical lead and zinc cleaner concentrates produced in the sulfide flotation circuits were as follows:

Test No.	Sample	Sulfide Pb Concentrate Wt %	Grade %	Distribution	Sulfide Zn Concentrate Wt %	Grade %	Distribution
1	High Grade Ore	0.5	51.0	39.6	11.9	57.0	97.0
2	Low Grade Ore	0.1	3.6	5.9	1.6	59.5	35.5

Oxide flotation of the sulfide circuit tailings accounted for the difference in the total rougher recoveries shown above and the distributions reporting to the sulfide circuit rougher concentrates. These data may be expressed most clearly in the following circuit analyses:

	Lead Circuits			Zinc Circuits		
Product	Wt %	Grade, %	Distr. %	Wt %	Grade, %	Distr. %
Test 10, High Grade						
Sulfide cleaner conc 1/	0.5	51.0	39.6	11.9	57.0	97.0
Sulfide circuit conc	18.50	2.05	60.8	18.50	36.4	98.7
Oxide rougher conc	22.09	0.96	34.0	22.09	0.35	1.1
Final tailings	59.41	0.055	5.2	59.41	0.020	0.2
Test 9, Low Grade						
Sulfide cleaner conc 1/	0.1	3.6	5.9	1.6	59.5	35.5
Sulfide rougher conc	16.32	0.16	38.9	16.32	9.0	54.5
Oxide rougher conc	46.14	0.07	47.0	46.14	1.9	33.2
Final tailings	37.54	0.025	14.1	37.54	0.88	12.3

1/ Estimated, based on Test 1 or 2 sulfide cleaner flotation results since sulfide cleaner flotation was not directly conducted on the sulfide rougher concentrates in Tests 9 and 10.

There are several possible options for disposition of the oxide circuit concentrates. Although the grades are low enough that they could not be smelted directly, it is possible that they could be shipped as middlings products for further milling at other lead-zinc operations in the area. The oxide concentrates also could be blended in small amounts with high-grade sulfide concentrates. Leaching of the concentrates for selective lead and zinc recovery is also possible, or they may have to be disposed of off-site.

Standard EP Toxicity Tests were conducted on final tailings to determine whether they would pass EP Toxicity criteria. Resultant data are summarized below.

Tailings Leachate Assays, ppm

Element	EP Toxicity Standards	High Grade Ore Tests: Conventional Sulfide T-1	Medium Grind T-7	Finest Grind T-10	Coarsest Grind T-11	Low Grade Ore Tests: Conventional Sulfide T-2	Finest Grind T-9	Coarsest Grind T-13
As	5.0	ND	ND	ND	ND	ND	ND	ND
Ba	100.0	0.39	0.27	0.48	0.28	0.2	.0.18	0.1
Cd	1.0	0.06	0.33	0.02	0.03	1.7	0.98	1.0
Cr	5.0	0.06	ND	ND	ND	ND	0.04	ND
Pb	5.0	25	8.1	2.9	46	1.0	0.4	1.3
Hg	0.2	ND	ND	ND	ND	ND	ND	ND
Se	1.0	ND	0.001	ND	ND	ND	ND	ND
Ag	5.0	ND	ND	ND	ND	ND	ND	ND

Notes: Zinc assays are not included in a standard EP Toxicity Test.
ND = not detected
T-1, etc., refers to Test No.

The tailings from each of the high and low grade ore samples in at least one test contained metals below the EP Toxicity standards. The best tailings were produced in Tests 10 (High Grade) and 9 (Low Grade). Both tests included a 3-stage sequential flotation oxide circuit. Test 7 on the High Grade Ore, conducted at a coarser grind, narrowly missed the standard for Pb. Test 13 on the Low Grade Ore, conducted at a grind of 70.1% minus 200 mesh, also passed the standards. The tailings from the conventional sulfide flotation circuit, i.e., without the oxide step, did not meet EP Toxicity test specifications.

Flotation tailings thickened readily to 50 to 60% solids using lime and polyacrylamide flocculant. Thickener unit area requirements were as low as 0.1 ft^2/ton of dry tailings/day.

Assay/screen analyses of the Low Grade "Chat Sample" showed that 80 to 90 weight percent could be separated at +20 to 28 mesh and possibly discarded without milling or flotation. The fraction assayed 0.02% Pb and 0.1% Zn. Additional tests would have to be conducted on the classified Chat Sample rejects to confirm whether they would satisfy EP Toxicity standards.

Flotation of a composite of High Grade ore and classified Low Grade Chat Sample (upgraded portion) resulted in lower recoveries compared to other samples. Additional testing needs to be conducted on composited high/low grade samples to better define the reagent conditions.

TEST WORK

Samples and Preparation

Two principal test work samples were hand delivered by CH_2M Hill on July 11, 1988, for the testing program. They were described as High Grade (HRI No. 39467) and Low Grade (39468). The samples weighed approximately 27 kg each. The as-received minus 4-inch High Grade Ore was described as hand-picked. Low Grade Sample 39468 was received at minus 3/8 inch.

In preparation for testing, the samples were individually stage crushed to 6 mesh. A 10-kg split was reserved for dry grindability testing, and the remainder then crushed to passing 10 mesh. The 10-mesh material was blended and split into 2-kg test charges and two head assay portions. The prepared charges were bagged in plastic and stored in a freezer to reduce surface oxidation.

Two additional Low Grade Chat Samples were received on August 11 from CH_2M Hill, accompanied by chain of custody record No. 067900, and described as Chat Samples 01-02 and 07-01-02. The 13-kg samples were minus 1/4 inch and were assigned HRI Nos. 39849-1 and -2, respectively. The samples were blended and split into 4- x 1-kg splits which were intended for assay-particle size analyses. The rejects were crushed to minus 10 mesh; head assay portions were prepared and the remainder stored for possible compositing with high grade ore.

Head Assays

Detailed head analyses for all received samples are shown in Table 1. All analyses were performed using standard industrial practice atomic absorption procedures. CLP (Contract Lab Program) protocol analysis procedures were requested for specific portions of the work, and are designated where used hereafter. The various analytical procedures used in the work are appended for reference. Original analytical certificates have been given to CH_2M Hill. Copies of the certificates are appended for completeness.

Grindability Tests

One standard, dry, Bond grindability test was conducted on the original high grade and low grade samples. The results, summarized below, showed that the

materials were relatively hard. Details of the grindability tests are appended (Tables A-1 and A-2) for reference.

Sample	Grinding Test No.	100% Passing Size of Production, µm	P_{80}, µm	F_{80}, µm	Work Index
39467	536	147	116	2567	20.17
39468	537	147	118	2147	18.81

Grinding of Ore Samples

Grinding of the ore samples for flotation testing was conducted using 1- or 2-kilogram 10-mesh ore charges, as required. Grinding was conducted in a 5- x 12-inch diameter Denver steel ball mill containing a seasoned 12.7-kg ball charge. Grinds were conducted to determine the length of time required to produce ground products sizing approximately 60, 85, and 100% passing 200 mesh. Particle size analyses for the flotation feeds are appended for reference (Tables A-3 to A-8).

Chat Sample Assay-Size Analyses

Five hundred-gram portions of the Low Grade Chat samples 01-02 (39849-1) and 07-01 (39849-2) were wet screened on a standard sieve series of 17 screens from 1/4 inch to 400 mesh. Each fraction was dried and prepared for assay. The water collected from the screening was measured and sampled for analysis. All of the products were submitted for Pb, Zn, and Cd analyses, and the water for Pb only. Analyses were performed using CLP protocols. A metallurgical balance was calculated for each sample for Pb, Zn, and Cd. Results are shown in Tables 2 and 3.

The data showed that the Chat Samples could conceivably be classified at 20 to 28 mesh, with the oversize portion comprising a significant weight percentage of from 81 to 92 percent, and assaying very low in lead and zinc, i.e., <0.02% Pb and <0.10% Zn.

It was thought that the oversize 28-mesh material would be discarded as tailings, and the upgraded 8 to 20 weight percent of the material either floated directly or blended with higher grade ore wastes for flotation. Standard EP Toxicity tests need to be conducted on the plus 20- or 28-mesh fractions to

determine if they will pass these standards before they can be dismissed as tailings material.

Water from the screening procedure showed no detectable lead (detection limit was 0.1 ppm).

Flotation response of the classified Chat Samples is discussed in the following flotation section.

Flotation Testing

Conventional Sulfide Flotation

An initial sulfide flotation test was conducted on both the High Grade (39467) and Low Grade (39468) ore samples to determine sulfide lead and zinc recoveries using conventional milling practice. Metallurgical balances are shown in Tables 4a/4b and 5a/5b and flotation conditions in Tables 6 and 7.

Lead was first selectively recovered using diaryldithiophosphoric acid (R242) and either potassium amyl xanthate (AX350) or sodium ethyl xanthate (AX325) collectors, with small additions of sodium cyanide and zinc sulfate as zinc depressants. The lead rougher concentrate was cleaned once by reflotation.

Zinc was then activated by copper sulfate addition (reversing the effect of the cyanide and zinc sulfate) and separately collected using two collectors: sodium isopropyl xanthate (AX343) and ethyl isopropyl thionocarbamate (M-1661). The pH was increased in the zinc circuit to about 11 with $Ca(OH)_2$ to control potential pyrite flotation. The zinc rougher concentrate was upgraded in one or two cleaner stages by reflotation at high pH.

Flotation products from these and all subsequent flotation tests were oven dried, weighed, and pulverized for analysis. Products were assayed for Pb, Zn, Cd, Cu, S^{2+}, and As. Standard atomic absorption techniques were used for all analyses. Complete metallurgical balances were calculated for each test. The flotation final tailing pulp was filtered and the tailings water sampled, with portions stabilized at pH values >13 and <2 for possible impurities analyses.

Results of the conventional flotation showed that a lead concentrate was produced easily from the High Grade sample, assaying 51.0% Pb and 4.5% Zn. A lead concentrate was not successfully produced from the Low Grade material due to the considerably lower head grade and lack of sufficient material for cleaning.

Lead recovery for the High Grade sample was 52.2% in the lead rougher, with an additional 8.5% recovered in the zinc circuit, for a total lead recovery of 60.7%. On the Low Grade sample, lead rougher recovery was only 11.1%, with an overall lead recovery of 18.8% in both lead and zinc sulfide flotation.

Excellent zinc sulfide concentrates were produced from each sample. The High Grade sample showed very good sulfide zinc promotion, with an 11.9 weight percent zinc concentrate assaying 57.0% Zn. This represents a zinc recovery of 97.0%. The zinc rougher distribution was 97.3%, and overall zinc recovery from the feed was 99.2% (1.9 points recovered in the lead circuit). From the Low Grade ore, a zinc concentrate was produced which assayed 59.5% Zn, for a zinc recovery of 35.5%. Overall zinc recovery from the sample was 42.5%.

The sulfide flotation results are summarized as follows:

	Calculated Head Assays, %		Final Tailing, %		Overall Recovery, %	
Sample	Pb	Zn	Pb	Zn	Pb	Zn
High Grade	0.66	6.97	0.310	0.070	60.7	99.2
Low Grade	0.076	2.70	0.065	1.65	18.8	42.5

Based on prior test work by others on materials from Galena, CH_2M Hill indicated that it was their knowledge that these final tailings would not pass EP Toxicity test standards for lead and cadmium (the estimated end result for cadmium was based on the zinc assay, since cadmium is most often intimately associated with zinc mineralization). They indicated that a final tailing produced from these types of materials would need to assay approximately 0.05% Pb and 0.10% Zn to pass the EP Toxicity test for lead and cadmium. These estimates were based on comparisons of tailings assays and resultant EP Toxicity results. There was no established standard for zinc discharge, and Zn assays are not included in a standard EP Toxicity test.

Based on these observations, a series of tests was conducted to study oxide flotation conditions on the sulfide tailings, in order to further lower the final tailing lead and zinc contents. Description and development of the oxide circuitry are discussed in the following section.

Oxide Flotation

Summary results for all flotation tests conducted are shown in Table 8. The conceptual milling flowsheet including oxide flotation circuitry is previously shown in Figure 1. Individual test metallurgical balances and test operating sheets are also appended for reference (Tables A-9 to A-67).

The initial oxide flotation effort involved conventional conditioning of the sulfide tailings with sodium sulfide, followed by flotation using M-1661 and AX343 collectors. This procedure was conducted on both the Low and High Grade samples in Tests 3 and 4, respectively. Results on the Low Grade sample showed an increase in overall lead and zinc recoveries from 18.8 and 42.5% (Test 2) to 44.8 and 53.6%, respectively, in Test 3, with the final tailing reduced to 0.044% Pb and 1.50% Zn. Although a significant improvement, the tailing zinc assay was still too high. The High Grade sample also showed increases in overall lead and zinc recoveries from 60.7 and 99.2% in Test 1 to 80.9 and 99.3%. The final tailing assayed 0.160% Pb and 0.059% Zn. The tailing zinc assay was reduced only slightly from that obtained from the sulfide flotation, due to the unusually high zinc recovery in sulfide flotation. The tailing lead assay was reduced by more than half by the sulfidization operation, but remained higher than the desirable 0.05% Pb level.

Subsequent oxide mineral recovery efforts in later tests involved use of chelating and flocculating agents, sulfidization, and use of a strong oxide-sulfide mineral collector. The general oxide circuit conditions developed for overall oxide lead and zinc recovery involved the following elements:

Stage 1	Conditioning with a chelating agent followed by a hydroxamate collector.
Stage 2	Selective flocculation of the metallic oxides, with a hydrophobic oxygenated polymer, followed by collection by an oxide mineral collector.
Stage 3	Conventional sulfuridization, collected by oxide and sulfide collectors.

Chelating agents are generally water-miscible organic compounds that complex with the metallic oxide minerals to change their ionic character. The resultant complexes can then be collected by a specialized collector or, in some instances, by thiol collectors. The chelating agent utilized here was a Lubrizol product, designated 81469C. The collector was alkyl hydroxymate (American Cyanamid S-6410). Flocculation of metallic oxides for collection was accomplished by a

metallic-oxide specific flocculating agent, polyethylene oxide. The strong, multi-purpose, sulfide-oxide collector used here was dodecyl mercaptan (Pennwalt PF-3).

High Grade Sample Results

The best initial flotation results on the High Grade sample (HRI 39467) using the 3-stage oxide flotation conditions were achieved in Test 7, on a primary grind product of 83.5% minus 200 mesh. Overall lead and zinc recoveries were 89.8% and 99.6% (see Table 8), respectively, with the final tailings assaying 0.110% Pb and 0.045% Zn. Lead and zinc recoveries by the oxide circuits alone were 27.6% lead and 0.7% zinc, respectively, in addition to the sulfide circuit recoveries of 62.2% lead and 99.0% zinc. The tailing, however, did not pass EP Toxicity test standards.

The best results on the High Grade sample were obtained in Test 10 which utilized a finer primary grind of 96.3% minus 200 mesh and the same flotation conditions as in Test 7. Overall lead and zinc recoveries were 94.8 and 99.8%, respectively, of which 34.0% lead distribution and 1.1% zinc were produced by the oxide circuits. The final tailing assayed 0.055% Pb and 0.020% Zn. This tailing passed EP Toxicity test standards which is discussed in a following section.

A number of other tests were conducted to investigate various conditions in the oxide circuit to simplify the flotation conditions and reduce reagent conditioning times. None of the other conditions resulted in improved lead recoveries. Two tests, Nos. 19 and 20, showed nearly the same overall recoveries (93.3% and 91.6% lead), with the final tailings assaying 0.115% Pb; however, weight recoveries in the oxide circuit were considerably higher. Test 11, conducted at the coarsest primary grind of 67.3% minus 200 mesh, showed a definite relationship of grind fineness to lead recovery in the oxide circuit. Tests 10, 7, and 11, conducted at grinds of 96.3%, 83.5%, and 67.3% minus 200 mesh, respectively, showed lead recoveries of 34.0, 27.6, and 19.1% in the oxide circuit. An additional test at an even finer grind should be conducted to determine whether the final tailing lead assay can be further reduced.

Low Grade Sample Results

Only three tests were conducted (Tests 8, 9, 13) on the Low Grade sample HRI No. 39468 to determine lead/zinc recoveries as a function of grind fineness. Conditions of Test 7, as used on the High Grade sample, were utilized. Results showed that lead/zinc recoveries were related to the fineness of the primary grind,

similar to that demonstrated on the high-grade sample. Results may be summarized as follows:

Test No.	Grind Fineness % -200m	Overall Recovery, % Pb	Overall Recovery, % Zn	Oxide Circuit Recovery, % Pb	Oxide Circuit Recovery, % Zn	Final Tailing Assay Pb, %	Final Tailing Assay Zn, %
13	70.1	57.8	60.5	33.5	15.8	0.039	1.40
8	85.3	77.3	82.3	32.5	21.5	0.028	0.88
9	88.5	85.9	87.7	47.0	33.1	0.025	0.88

As suggested for the High Grade sample, finer grinding may well produce an even lower grade final tailing. No further tests were conducted on this particular low-grade sample, although additional tests were conducted on classified fractions of two other Chat samples described in a later section.

EP Toxicity Testing

Final tailings from selected flotation tests were submitted for standard EP Toxicity tests. These tests were conducted to determine if the final tailing products would pass EP Toxicity test standards, to enable their use as mine fill. A list of the flotation tailings submitted and the EP Toxicity test results is shown in Table 9.

Results showed that at least one tailing for each of the High and Low Grade samples passed the allowable standards. The best resultant final tailing for the High Grade sample was from Test 10, conducted at the finest primary grind. The final tailing assayed 0.055% Pb and 0.020% Zn. For the Low Grade sample, two tailings passed the standards, i.e., Tests 9 and 13. These two tests were conducted at widely varied primary grind finenesses, but the final tailing Pb assay was apparently low enough in the more coarsely ground test to pass the standard for lead. Tailings assays were 0.025% Pb and 0.88% Zn for Test 9, and 0.039% Pb and 1.40% Zn for Test 13.

Final tailings zinc assays were considerably higher for the Low Grade sample than those shown for the High Grade ore tested. EP Toxicity test procedures and regulations do not specify a discharge limit for zinc. However, cadmium assays were also considerably higher in the Low Grade tailing EP Toxicity results. This was to be expected considering the common association of zinc and cadmium mineralization in geologic samples. The cadmium assays for the Low Grade sample were at the allowable limit, suggesting that additional test work to lower the tailing

zinc assay may be desirable on that sample. There was no concern regarding cadmium results on the High Grade sample.

Classified Low Grade Chat Flotation

As previously discussed, the Low Grade Chat sample 07-01 (HRI 39849-2, averaging 0.044% Pb, 0.15% Zn) was screen classified on 20 mesh, and the lower grade oversize material discarded as tailings. The upgraded minus 20-mesh material, assaying 0.260% Pb and 0.974% Zn, was then floated directly in Test 14 using the sulfide/oxide process flowsheet conditions. The final flotation tailing assayed 0.095% Pb and 0.047% Zn. If the classified feed results are compared to the best High Grade ore test results, which resulted in tailings of 0.055 to 0.11% Pb and 0.020 to 0.045% Zn, these are fairly good results. The Test 14 tailing should conceivably pass EP Toxicity standards, although the tailing was not tested.

We recommend that this classification approach be pursued in additional testing, since a large weight percentage of Low Grade material (in the case of this sample, 92.40 weight percent) may be classified out and discarded as final tailings, thereby bypassing the milling operation. Testing should be conducted on additional classified fractions of varied lead and zinc head assays, to determine classification parameters. These tests should be accompanied by EP Toxicity testing of the classified waste portions and the resultant flotation tailings.

Flotation of a composite of High Grade ore and classified Low Grade Chat was studied in Test 15. The composite makeup was as follows:

			Assays, %		Distribution, %	
Sample		Wt %	Pb	Zn	Pb	Zn
High Grade ore	39467	71.5	0.634	6.18	96.9	99.8
01-02 sample -28M fraction	39849-1	28.5	0.050	0.03	3.1	0.2
Composite calc'd head		100.0	0.465	4.42	100.0	100.0
Test calc'd head			0.546	5.11		

The composite was ground to 85% minus 200 mesh and floated using the conditions of High Grade Test 10. The final tailing assayed 0.250% Pb and 0.07% Zn. Overall lead and zinc recoveries were 65.3% and 99.0%, respectively. Results were excellent for zinc, as might be expected, since the composite was basically a High Grade ore sample. There was no immediate explanation for the poor lead

recovery. Some additional improvement in lead recovery may result from finer grinding. It is possible that there was something peculiar to the 01-02 sample that caused the overall poor lead response. The 01-02 sample was not floated in any of the other tests conducted. Should this be true, however, additional composites should be prepared using other Low-Grade Chat samples of varying assays and property location, to highlight any future problems in flotation of ore blends.

Thickening

Standard settling tests were conducted on selected final tailings. Results are summarized in Table 10. Test results were achieved using a combination of lime and non-ionic polyacrylamide flocculant. Unit areas as low as 0.2 ft^2/tpd were obtained from both the High and Low Grade tailings.

Slightly different conditions were used on the Low Grade sample than for the High Grade. On the High Grade, it was necessary to add lime to the tailings to raise the pH to 11.75, followed by flocculant addition. For the Low Grade tailings, adding only flocculant to the oxide circuit tailings proved to be the best condition. Raising the pH of the Low Grade tailings with additional lime increased the unit area requirements.

The best thickening results were achieved on tailings from very fine primary grinds. Therefore, one would expect similar results should a coarser primary grind be used ultimately.

Details of the thickening tests are attached for reference (Tables A-58 to A-67).

RECOMMENDATIONS

The following recommendations are made for further study.

1. EP Toxicity tests should be conducted on the degraded, classified portions of Low Grade Chat Samples to assess their ability to pass toxicity test standards without additional treatment.
2. Assuming that blending of High Grade material with either Low Grade materials or classified (upgraded) Chat may be an economic processing alternative, additional flotation tests on various anticipated compositions of these materials should be conducted and followed by EP Toxicity testing of resultant tailings.
3. Conditioning times and reagent dosages in the oxide flotation circuits are not sufficiently optimized. Also, froth phases in the oxide stages were somewhat tenacious and sensitive to control. The role of the flocculating agent polyethylene oxide has not been defined completely. There are a number of other chelating agents available which may offer significant improvements in any of the aforementioned areas.
4. Other flotation conditions, such as the use of lime in place of soda ash, have not been investigated. Other conventional oxide lead and zinc flotation techniques should be studied, which may have the potential to simplify the oxide circuits or improve lead/zinc recoveries. These techniques may include desliming after sulfide flotation, which may reduce sulfidization reagent dosages and other oxide circuit reagent requirements. Use of a variety of amine-type cationic collectors or fatty acids for collection of probable carbonate and silicate lead/zinc minerals have also not been explored. Sodium sulfite may also be used as a zinc depressant to reduce cyanide dependence.
5. Some thought should be given to the marketability of the oxide rougher concentrates. Should they not be saleable directly (to a smelter, for instance), treatment by an alternative method, such as leaching, may be necessary.
6. The effect of recirculation of water and intermediate flowsheet products on the overall metallurgy and, more importantly, on the final tailing product, should be studied as soon as possible. The plant flowsheet can be simulated easily in the laboratory by locked-cycle testing. These tests will provide invaluable information on the effect of buildups in reagent concentrations due to water recirculation on process control.

CONCEPTUAL CONCENTRATOR FLOWSHEET

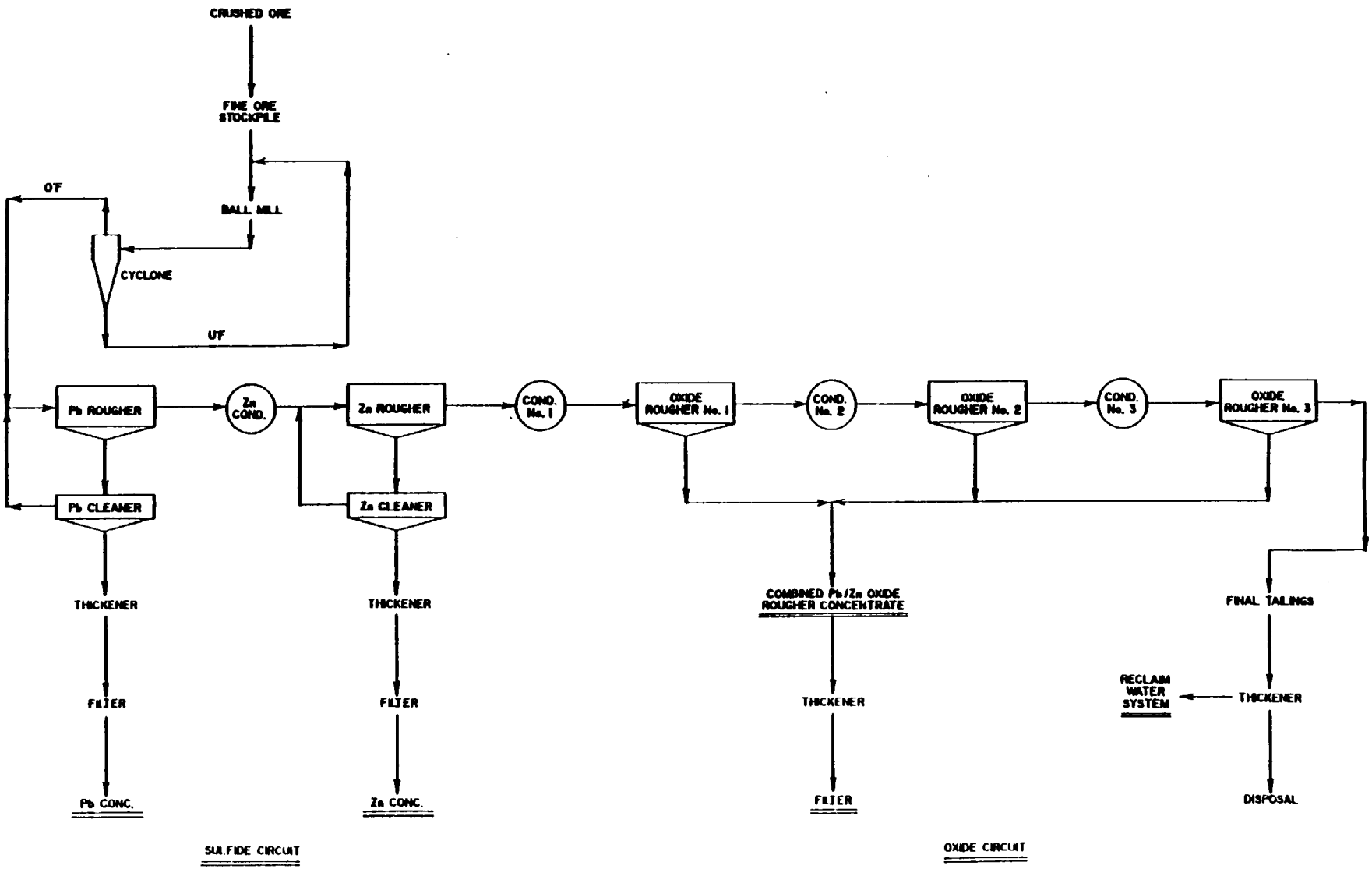


FIGURE 1

Table 1

Head Assays

Sample	HRI ID No.	Assays, % Pb	Zn	Cu	Cd	S	SO_4	Assay, As, ppm
High grade ore	39467A 1/	0.480	5.55	0.009	0.034	-	<0.05	-
High grade ore	39467B 2/	0.634	6.18	0.013	0.038	3.82	0.07	8
Low grade ore	39468A	0.071	2.90	0.008	0.009	0.65	<0.05	-
Low grade ore	39468B	0.075	2.70	0.009	0.010	0.75	0.05	5
Low grade chat	39849-1	0.050	0.02	-	0.0002	<0.02	-	-
	39849-1 R_1	0.053	0.046	-	0.0003	-	-	-
	39849-1 R_2	0.055	0.040	-	<0.001	-	-	-
Low grade chat	39849-2	0.035	0.10	-	0.0003	<0.02	-	3
	39849-2 R_1	0.042	0.160	-	0.0006	-	-	-
	39849-2 R_2	0.052	0.201	-	<0.001	-	-	-
		Pb(t)	Pb(ox)	Pb(s) 4/	Zn(t)	Zn(ox)	Zn(s) 5/	
High grade	39467B	0.634	0.126	0.508	6.18	0.022	6.16	-
Low grade	39468B	0.073	0.044	0.029	2.70	0.903	1.80	-
Chat	39849-2	0.052	<0.001	0.051	0.201	0.006	0.195	-

1/ Heads A assayed at Skyline Labs, Inc.
2/ Heads B assayed at Hazen.
3/ Heads R were new heads prepared from a different split of the materials at a later date to verify initial analyses.
R_1 assayed at Analytica, R_2 assayed at Hazen.
4/ Pb(t) = total Pb, Pb(ox) = oxide Pb, Pb(s) = sulfide Pb. Pb(s) is a calculated value of Pb(t) minus Pb(ox).
5/ Zn(t) = total Zn, Zn(ox) = oxide Zn, Zn(s) = sulfide Zn. Zn(s) is calculated value of Zn(t) minus Zn(ox).
Note: Dashes indicate assay not conducted.

Table 2

Metallurgical Results

Test No.: Assay Screen Analysis

Ore: Chat Sample No. 07-01, HRI No. 39849-2

Conditions: Wet screened a representative split of the as-received ore on a gyratory sieve shaker, and collected all water used.

		Weight		Assays, mg/kg			Distribution, %		
	Product	g	%	Pb	Zn	Cd	Pb	Zn	Cd
1	+3 mesh	0	0.00				0.0	0.0	0.0
2	-3 +4 mesh	9.76	2.47	210	930	3	1.2	1.4	1.3
3	-4 +6 mesh	164.13	41.53	240	1,200	4	23.6	30.5	29.4
4	-6 +8 mesh	131.61	33.30	250	780	3	19.7	15.9	17.7
5	-8 +10 mesh	40.51	10.25	230	460	2	5.6	2.9	3.6
6	-10 +14 mesh	10.27	2.60	240	780	5	1.5	1.2	2.3
7	-14 +20 mesh	8.88	2.25	280	2,000	12	1.5	2.8	4.8
8	-20 +28 mesh	2.64	0.67	330	7,400	35	0.5	3.0	4.1
9	-28 +35 mesh	3.63	0.92	350	14,000	48	0.8	7.9	7.8
10	-35 +48 mesh	1.50	0.38	350	20,000	65	0.3	4.6	4.4
11	-48 +65 mesh	1.31	0.33	400	22,000	70	0.3	4.5	4.1
12	-65 +100 mesh	0.72	0.18	400	21,000	60	0.2	2.3	1.9
13	-100 +150 mesh	0.3	0.08	600	34,000	70	0.1	1.6	0.9
14	-150 +200 mesh	0.14	0.04	700	46,000	20	0.1	1.0	0.1
15	-200 +270 mesh	0.18	0.05	0	100,000	0	0.0	2.8	0.0
16	-270 +400 mesh	0.05	0.01	0	0	0	0.0	0.0	0.0
17	-400 mesh	19.55	4.95	3,800	5,800	20	44.6	17.6	17.5
18	Water	1.356 L	-	0	-	-	0.0	0.0	0.0
Head calc'd		395.18	100.00	421.7	1,500.0	5.7	100.0	100.0	100.0
Head, direct				435.0	1,500.0	<10			
Calculated products									
Plus 4 mesh		1 and 2	2.47	210	930	3	1.2	1.4	1.3
Plus 6 mesh		1 to 3	44.00	238	1,185	4	24.9	31.9	30.7
Plus 8 mesh		1 to 4	77.31	243	1,010	4	44.6	47.8	48.4
Plus 10 mesh		1 to 5	87.56	242	946	3	50.2	50.7	52.0
Plus 14 mesh		1 to 6	90.16	242	941	3	51.7	58.0	54.3
Plus 20 mesh		1 to 7	92.40	243	967	4	53.2	54.7	59.1
Plus 28 mesh		1 to 8	93.07	243	1,013	4	53.7	57.7	63.2
-20 mesh classified		8 to 17	7.60	2,600	9,738	30	46.8	45.3	40.9
-28 mesh classified		9 to 17	6.93	2,819	9,963	30	46.3	42.3	36.8

Table 3

Metallurgical Results

Test No.: Assay Screen Analysis

Ore: Chat Sample No. 01-02, HRI No. 39849-1

Conditions: Wet screened a representative split of the as-received ore on a gyratory sieve shaker, and collected all water used.

		Weight		Assays, mg/kg			Distribution, %		
	Product	g	%	Pb	Zn	Cd	Pb	Zn	Cd
1	+3 mesh	5.78	1.63	110	590	0	0.4	1.9	0.0
2	-3 +4 mesh	63.01	17.75	100	460	3	3.9	16.3	25.5
3	-4 +6 mesh	67.88	19.12	100	450	2	4.2	17.1	18.3
4	-6 +8 mesh	10.47	2.95	100	240	0	0.6	1.4	0.0
5	-8 +10 mesh	94.54	26.63	200	190	0	11.7	10.1	0.0
6	-10 +14 mesh	17.26	4.86	150	230	0	1.6	2.2	0.0
7	-14 +20 mesh	24.62	6.93	190	320	2	2.9	4.4	6.6
8	-20 +28 mesh	3.59	1.01	200	730	3	0.4	1.5	1.5
9	-28 +35 mesh	21.84	6.15	170	820	3	2.3	10.1	8.8
10	-35 +48 mesh	6.07	1.71	240	1,100	7	0.9	3.7	5.7
11	-48 +65 mesh	6.47	1.82	230	1,100	7	0.9	4.0	6.1
12	-65 +100 mesh	4.23	1.19	250	1,300	8	0.7	3.1	4.6
13	-100 +150 mesh	2.60	0.73	460	1,400	9	0.7	2.0	3.2
14	-150 +200 mesh	1.24	0.35	300	1,400	9	0.2	1.0	1.5
15	-200 +270 mesh	1.51	0.43	320	1,300	8	0.3	1.1	1.6
16	-270 +400 mesh	0.61	0.17	360	1,200	9	0.1	0.4	0.7
17	-400 mesh	23.34	6.57	4,700	1,500	5	68.0	19.7	15.8
18	Water	1.800 L	-	0	-	-	0.0	0.0	0.0
Head, calc'd		355.06	100.00	454.4	501.8	2.1	100.0	100.0	100.0
Head, direct				550.0	400.0	<10			
Calculated products									
Plus 4 mesh		1 and 2	19.37	101	471	3	4.3	18.2	25.5
Plus 6 mesh		1 to 3	38.49	100	461	2	8.5	35.3	43.8
Plus 8 mesh		1 to 4	41.44	100	445	2	9.2	36.7	43.8
Plus 10 mesh		1 to 5	68.07	139	345	1	20.9	46.8	43.8
Plus 14 mesh		1 to 6	72.93	140	337	1	22.5	49.0	43.8
Plus 20 mesh		1 to 7	79.86	144	336	1	25.4	53.5	50.5
Plus 28 mesh		1 to 8	80.87	145	341	1	25.8	54.9	51.9
-20 mesh classified		8 to 17	20.14	1,684	1,159	5	74.6	46.5	49.5
-28 mesh classified		9 to 17	19.13	1,762	1,182	5	74.2	45.1	46.1

Table 4a

Flotation Test Metallurgical Results

Test No.: 1

Ore: HRI 39467 High Grade

Grind: 83.5% -200 mesh, P_{80} = 69.9

Conditions: Preliminary selective flowsheet test, using conventional Pb/Zn milling conditions

Product	Weight g	Weight %	Assays, % Pb	Assays, % Zn	Assays, % Cu	Distribution, % Pb	Distribution, % Zn	Distribution, % Cu
1. Pb concentrate	10.32	0.51	51.000	4.50	0.105	39.6	0.3	1.4
2. Pb 1st cleaner tailing	28.24	1.41	5.900	7.30	0.135	12.5	1.5	4.9
3. Zn concentrate	237.64	11.85	0.280	57.00	0.165	5.0	97.0	49.9
4. Zn 1st cleaner tailing	44.91	2.24	1.050	1.20	0.205	3.5	0.4	11.7
5. Zn rougher tailing	1,683.90	83.98	0.31	0.07	0.015	39.3	0.8	32.1
Head, calc'd	2,005.01	100.00	0.66	6.97	0.039	100.0	100.0	100.0
Head, direct			0.63	6.18	0.013			
Calculated products								
Pb rougher concentrate	1 and 2	1.92	17.97	6.55	0.13	52.2	1.8	6.2
Zn rougher concentrate	3 and 4	14.09	0.40	48.13	0.17	8.6	97.3	61.6
Total rougher recovery	1 to 4	16.02	2.51	43.14	0.17	60.7	99.2	67.9

Table 4b

Flotation Test Metallurgical Results

Test No.: 1

Ore: HRI 39467 High Grade

Grind: 83.5% -200 mesh, P_{80} = 69.9

Conditions: Preliminary selective flowsheet test, using conventional Pb/Zn milling conditions

Product	Weight		Assays, %			Distribution, %		
	g	%	$S_{(t)}$	Cd	As	$S_{(t)}$	Cd	As
1. Pb concentrate	10.32	0.51	11.80	0.025	0.001	1.6	0.3	0.9
2. Pb 1st cleaner tailing	28.24	1.41	5.20	0.040	0.003	2.0	1.3	6.0
3. Zn concentrate	237.64	11.85	29.00	0.351	0.001	92.8	97.1	10.1
4. Zn 1st cleaner tailing	44.91	2.24	1.50	0.006	0.003	0.9	0.3	11.5
5. Zn rougher tailing	1,683.90	83.98	0.12	0.001	0.001	2.7	1.0	71.6
Head, calc'd	2,005.01	100.00	3.71	0.043	0.0006	100.0	100.0	100.0
Head, direct			3.82	0.038	0.0008			
Calculated products								
Pb rougher concentrate	1 and 2	1.92	6.97	0.04	0.002	3.6	1.6	6.9
Zn rougher concentrate	3 and 4	14.09	24.63	0.30	0.001	93.7	97.4	21.6
Total rougher recovery	1 to 4	16.02	22.51	0.26	0.001	97.3	99.0	28.4

Table 5a

Flotation Test Metallurgical Results

Test No.: 2

Ore: HRI 39468 Low Grade

Grind: 85.3% -200 mesh, P_{80} = 67.9

Conditions: Preliminary selective flowsheet test, using conventional Pb/Zn milling conditions

Product	Weight g	Weight %	Assays, % Pb	Assays, % Zn	Assays, % Cu	Distribution, % Pb	Distribution, % Zn	Distribution, % Cu
1. Pb concentrate	2.53	0.13	3.550	5.15	0.035	5.9	0.2	0.2
2. Pb 1st cleaner tailing	22.79	1.14	0.345	6.40	0.020	5.2	2.7	1.2
3. Zn concentrate	32.39	1.61	0.035	59.50	0.150	0.7	35.5	13.0
4. Zn 2nd cleaner tailing	7.13	0.36	0.255	5.05	0.165	1.2	0.7	3.1
5. Zn 1st cleaner tailing	48.89	2.44	0.18	3.75	0.050	5.8	3.4	6.5
6. Zn rougher tailing	1,894.00	94.34	0.065	1.65	0.015	81.2	57.5	75.9
Head, calc'd	2,007.73	100.00	0.076	2.70	0.019	100.0	100.0	100.0
Head, direct			0.073	2.70	0.009			
Calculated products								
Pb rougher concentrate	1 and 2	1.26	0.67	6.28	0.021	11.1	2.9	1.5
Zn rougher concentrate	3 to 5	4.40	0.13	24.28	0.096	7.7	39.5	22.7
Total rougher recovery	1 to 5	5.66	0.25	20.27	0.079	18.8	42.5	24.1

Table 5b

Flotation Test Metallurgical Results

Test No.: 2

Ore: HRI 39468 Low Grade

Grind: 85.3% -200 mesh, P_{80} = 67.9

Conditions: Preliminary selective flowsheet test, using conventional Pb/Zn milling conditions

	Weight		Assays, %			Distribution, %		
Product	g	%	$S_{(t)}$	Cd	As	$S_{(t)}$	Cd	As
1. Pb concentrate	2.53	0.13	2.100	0.017	0.003	0.4	0.2	0.6
2. Pb 1st cleaner tailing	22.79	1.14	1.000	0.019	0.002	1.7	2.4	4.1
3. Zn concentrate	32.39	1.61	30.300	0.265	0.001	75.0	49.6	1.5
4. Zn 2nd [illegible]er tailing	7.13	0.36	1.300	0.019	0.003	0.7	0.8	1.9
5. Zn 1st cleaner tailing	48.89	2.44	0.52	0.011	0.002	1.9	3.1	6.6
6. Zn rougher tai[illegible]g	1,894.00	94.34	0.14	0.004	0.001	20.3	43.8	85.3
Head, calc'd	2,007.73	100.00	0.652	0.009	0.0006	100.0	100.0	100.0
Head, direct			0.750	0.010	0.0005			
Calculated products								
Pb rougher concentrate	1 and 2	1.26	1.11	0.02	0.002	2.1	2.7	4.7
Zn rougher concentrate	3 to 5	4.40	11.49	0.10	0.001	77.6	53.5	10.0
Total rougher recovery	1 to 5	5.66	9.18	0.09	0.001	79.7	56.2	14.7

Table 6

Flotation Test Report

Test No. 1 Project No. 6905 Operator R. Rak Date 7/21/88

Purpose: Preliminary selective flowsheet test, high-grade sample

Feed: 2000 g of minus 10-mesh ore, HRI 39467

Grind: 60 min at 65% solids = 83.5% minus 200 mesh

Conditions:

	Reagents Added, Pounds/Ton						Time, Minutes			
Stage	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX350	MIBC/ DF250	Grind	Cond	Froth	Pulp pH
Grind	1.5						60			7.6
Pb rougher (1)	1.5	1.0	0.33	0.030	0.01	0.020/-		5+1	3	9.0
(2)				0.015	-			2	3	
	$Ca(OH)_2$									
Pb 1st cleaner	0.20	0.10	0.01	0.01				3	3	9.5
		$CuSO_4$	M-1661	AX343						
Zn condition	5.8	1.5						5		11.0
Zn rougher (1)			0.02	0.02		0.015/0.015		2	3	11.0
(2)			0.02	0.02		0.005/0.005		2	3	10.1
(3)			0.02	0.02		-/0.005		1	3	10.1/10.8
Zn 1st cleaner	0.20		0.01					-	5	11.0

Table 7

Flotation Test Report

Test No. 2 Project No. 6905 Operator R. Rak Date 7/21/88

Purpose: Preliminary selective flowsheet test, low-grade sample

Feed: 2000 g of minus 10-mesh ore, HRI 39468, low grade

Grind: 60 min at 65% solids = 85.3% minus 200 mesh

Conditions:

	Reagents Added, Pounds/Ton						Time, Minutes			
Stage	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC/ DF250	Grind	Cond	Froth	Pulp pH
Grind	3.0						60			9.20
Pb rougher (1)		1.0	0.33	0.03	0.01	0.015/-		5+1	3	9.00
(2)				0.02	-			1	3	
	$Ca(OH)_2$									
Pb 1st cleaner	0.20	0.10	0.01					3	2	9.5
		$CuSO_4$	M-1661	AX343						
Zn condition		1.0						5		11.0
Zn rougher (1)			0.02	0.02		0.010/0.030		1	3	11.0
(2)			0.02	0.02				1	3	10.4
(3)			0.03	-				1	3	11.0
Zn 1st cleaner	0.10		0.01					1	5	11.0
Zn 2nd cleaner	0.10		0.005					1	3	11.0

Table 8

Flotation Testing Summary

High grade sample

Test No.	Conditions	Pb Circuit: Oxide Ro Conc, % Wt	Pb Oxide Ro Conc, % Pb	Pb Oxide Ro Conc, % Recovery	Pb Ro Conc, % Wt	Pb Ro Conc, % Pb	Pb Ro Conc, % Recovery	Zn Circuit: Oxide Ro Conc, % Wt	Zn Oxide Ro Conc, % Zn	Zn Oxide Ro Conc, % Recovery	Zn Ro Conc, % Wt	Zn Ro Conc, % Zn	Zn Ro Conc, % Recovery	Ro Tail, % Pb	Ro Tail, % Zn	Total Recovery, % Pb	Total Recovery, % Zn
1	Conventional 1/	0.5	51.0	39.6	1.9	18.0	52.2	11.9	57.0	97.0	14.1	48.1	97.3	0.310	0.070	60.7	99.2
4	Oxide scav: sulfidized	3.2	3.5	16.8	2.5	14.5	54.9	3.2	0.48	0.2	15.4	43.5	96.8	0.160	0.059	80.9	99.3
5	Oxide scav: BHQ	3.1	1.2	5.5	2.4	15.5	56.7	3.1	0.89	0.4	16.4	43.5	96.7	0.255	0.049	69.0	99.5
6	Oxide scav: R-6410 & DMG	4.9	1.0	7.3	2.7	15.0	58.1	4.9	0.92	0.5	17.2	43.0	96.5	0.265	0.045	71.7	99.6
7	Oxide: 81459C, PEO, sulfdz	22.5	0.8	27.6	3.0	12.0	56.0	22.5	0.22	0.7	14.4	48.0	95.8	0.110	0.045	89.8	99.6
10	Test 7, finest PG	22.1	1.0	34.0	2.7	12.0	51.0	22.1	0.35	1.1	15.9	41.5	96.3	0.055	0.020	94.8	99.8
11	Test 7, coarsest P6	13.1	1.0	19.1	1.9	20.5	55.6	13.1	0.32	0.5	16.9	43.5	96.7	0.185	0.055	82.0	99.5
16	Deleted 81469C	3.7	1.2	5.6	4.0	11.5	60.3	3.7	1.2	0.6	13.5	50.0	86.0	0.290	0.071	70.4	99.3
17	Deleted R-6410	13.8	0.4	9.2	3.3	11.0	58.7	13.8	0.29	0.6	12.8	50.0	88.0	0.230	0.065	74.2	99.4
18	Sulfdz before PEO	15.9	0.4	12.0	3.6	10.5	65.2	15.9	0.18	0.4	11.6	50.0	89.3	0.145	0.033	82.6	99.6
19	Sulfdz at start of oxide	47.1	0.4	28.9	2.7	14.0	58.9	47.1	0.14	0.9	12.9	50.0	92.5	0.115	0.025	93.3	99.9
20	Sulfdz + chelate together	34.0	0.5	24.8	3.4	12.5	62.7	34.0	0.21	1.0	13.3	50.5	91.0	0.115	0.029	91.6	99.8
21	Sulfdz + R-6410 together	5.7	1.1	9.4	2.8	13.5	57.6	5.7	0.34	0.3	16.0	42.5	91.7	0.215	0.040	75.6	99.6
22	Sulfdz + PEO together	13.7	0.5	10.4	4.9	7.5	62.3	13.7	0.23	0.5	13.7	46.0	89.5	0.185	0.029	78.7	99.7

Low grade sample

Test No.	Conditions	Pb Circuit: Oxide Ro Conc, % Wt	Pb Oxide Ro Conc, % Pb	Pb Oxide Ro Conc, % Recovery	Pb Ro Conc, % Wt	Pb Ro Conc, % Pb	Pb Ro Conc, % Recovery	Zn Circuit: Oxide Ro Conc, % Wt	Zn Oxide Ro Conc, % Zn	Zn Oxide Ro Conc, % Recovery	Zn Ro Conc, % Wt	Zn Ro Conc, % Zn	Zn Ro Conc, % Recovery	Ro Tail, % Pb	Ro Tail, % Zn	Total Recovery, % Pb	Total Recovery, % Zn
2	Conventional 1/	0.1	3.6	5.9	1.3	0.7	11.1	1.6	59.5	35.5	4.4	24.3	39.5	0.065	1.650	18.8	42.5
3	Oxide scav: sulfidized	5.3	0.31	23.4	1.1	0.6	8.6	5.3	5.0	9.2	5.7	20.5	41.6	0.044	1.500	44.8	53.6
8	Oxide: 81469C, PEO, sulfdz	24.8	0.09	32.5	8.2	0.2	26.4	24.8	2.5	21.5	8.8	15.0	45.5	0.028	0.880	77.3	82.3
9	Test 8, finest PG	46.1	0.07	47.0	9.7	0.2	27.0	46.1	1.9	33.1	6.6	16.0	39.4	0.025	0.880	85.9	87.7
12	Test 9, medium PG	15.8	0.19	28.1	2.3	1.1	22.9	15.8	3.2	14.7	6.0	27.5	47.7	0.052	1.450	63.5	68.2
13	Test 9, coarsest PG	11.5	0.22	33.5	1.7	0.6	12.9	11.5	4.0	15.8	4.4	26.5	40.3	0.039	1.400	57.8	60.5
14	-20m classif. chat 07-01	15.7	0.4	28.4	9.2	0.3	12.9	15.7	0.3	4.3	12.6	5.9	80.4	0.095	0.047	69.9	96.8
15	71.5% high grade & 28.5% -28m chat 01-02	9.6	0.5	8.8	3.4	7.8	48.8	9.6	0.4	0.7	11.3	42.5	93.7	0.250	0.070	65.3	99.0

1/ On the conventional Tests 1 and 2, oxide rougher concentrate data are for Pb and Zn cleaner concentrates (there were no oxide circuits conducted in these tests).

Table 9

EP Toxicity Test Results

EP Toxicity Standards ppm		Flotation Test No.								
		High-grade Sample					Low-grade Sample			
		1	11	7	7r	10	2	13	9	12
5	As	ND	ND	0.001	ND	ND	ND	ND	ND	ND
100	Ba	0.39	0.28	0.37	0.27	0.48	0.20	0.1	0.18	1.0
1	Cd	0.06	0.03	0.04	0.33	0.02	1.7	1.0	0.98	1.6
5	Cr	0.06	ND	0.02	ND	ND	ND	ND	0.04	ND
5	Pb	25	46	23	8.1	2.9	1.0	1.3	0.4	3.0
0.2	Hg	ND	ND	ND	ND	ND	ND	ND	ND	ND
1	Se	ND	ND	ND	0.001	ND	ND	ND	ND	ND
5	Ag	ND	ND	ND	ND	ND	ND	ND	ND	ND

1. ND = not detected.
2. Test 7r values are repeat analyses on a new split from those reported previously in Test 7.

List of Flotation Tests

1	High grade	Initial Pb and Zn sulfide flotation only - Sulfide tails
11	High	3-stage oxide conditions, 67.3% - 200M
7	High	3-stage oxide conditions, 83.5% - 200M
10	High	3-stage oxide conditions, 96.3% - 200M
2	Low grade	Initial Pb and Zn sulfide flotation only - Sulfide tails
13	Low	3-stage oxide conditions, 70.1% - 200M
9	Low	3-stage oxide conditions, 85.3% - 200M
12	Low	3-stage oxide conditions, 88.5% - 200M

Table 10

Thickening Test Summary Results

Test	Oxide Tailing	Test Conditions	Grind Fineness, Mesh	Flocculant Dosage Type	Flocculant Dosage lb/ton	Additive Dosage Type	Additive Dosage lb/ton
10a	High grade	Lime --> pH 11.6	96.3% -200			Lime	1.23
10b	High grade	pH = 11.75 flocced	"	P-351	0.047	Lime	0.50
11a	High grade	Lime --> pH 11.6	65.8% -200			Lime	1.04
11b	High grade	pH = 11.75 flocced	"	P-351	0.024	Lime	0.50
12a	Low grade	pH = 10.5 flocced	88.5% -200	P-351	0.074	None	
12b	Low grade	pH = 10.5 flocced	"	P-351	0.082	None	
12c	Low grade	Lime --> pH = 11.6	"			Lime	0.56
12d	Low grade	Lime --> pH = 11.75	"			Lime	0.40
13a	Low grade	Lime --> pH = 11.6	70.1% -200			Lime	0.65
13b	Low grade	pH = 11.75 flocced	"	P-351	0.103	Lime	0.35

Test	Percent Solids Initial	Percent Solids Final	Feed Concentration Zone Settling Rate, ft/hr	Feed Concentration Zone Unit Area $ft^2/t/day$	Compression Zone Settling Rate, ft/hr	Compression Zone Unit Area $ft^2/t/day$
10a	25.5	67.4	4.2	0.8	1.8	1.3
10b	24.4	54.9	30.0	0.1	6.3	0.1
11a	26.0	54.3	3.6	0.7	1.3	1.3
11b	26.1	54.1	27.0	0.1	1.4	0.4
12a	26.9	51.3	4.1	0.6	10.8	0.1
12b	26.9	49.1	12.0	0.2	5.9	0.1
12c	28.7	55.1	1.2	1.9	0.2	8.6
12d	29.1	54.8	1.1	2.0	1.1	1.9
13a	31.3	57.6	1.1	1.7	0.9	1.9
13b	31.3	58.4	12.4	0.2	0.3	1.9

APPENDIX

Table A-1a

STANDARD BOND BALL MILL GRINDABILITY
Hazen Research, Inc.

HRI Grindability Test No. 536

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond Work Index number.

Procedure: The equipment and procedure used duplicate the Bond method for determining ball mill work indices.

Sample: Client identified High Grade "Ore"
The sample (HRI 39467) was stage crushed to minus 6 mesh. This material was used as the feed for the grindability test.

Results: Calculation of a Bond Work Index

$$W_i = \frac{44.5}{(P_1)^{0.23} \times (Gbp)^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}}\right)}$$

W_i = Bond Work Index, kwhr/ton to 80% minus 100 microns

P_1 = 100% passing size of product 147 microns

Gbp = Grams per revolution 0.96 grams

P_{80} = 80% passing size of product 118 microns

F_{80} = 80% passing size of feed; 2567 microns

W_i = 20.17

Table A-1b

BOND GRINDABILITY TEST RESULTS
Hazen Research, Inc.

HRI Grindability Test No. 536
HRI Project No.: 6905
Date: July 15, 1988

HRI Sample No. 39467
100% mesh of grind: 100
Operator: DAN

Test Conditions and Notes:

Test Feed Weight (700 cc) = 1187.4 g

% Minus 100 M in Feed = 5.7 %

Target product weights: +100 M = 848.1 g
-100 M = 339.3 g

Test Ball Charge

Nominal Ball Size Inch	No. of Balls	Wt, g
1.50	32	7523
1.25	49	6772
1.00	10	796
0.75	28	3210
0.50	10.4	1826
Total		20,127

Test Cycle	Number of Revolutions	Product +100M g	Product -100M g	Feed -100M g	Product Produced -100M g	Grams per Revolution	% Circulating Load
1	175	936.8	250.6	67.7	182.9	1.05	374
2	305	889.0	298.4	14.3	284.1	0.93	298
3	344	854.4	333.0	17.0	316.0	0.92	257
4	348	848.4	339.0	19.0	320.0	0.92	250
5	348	838.5	348.9	19.3	329.6	0.95	240
6	337	842.6	344.8	19.9	324.9	0.96	244
7	333	848.6	338.8	19.7	319.1	0.96	250
8	333	847.7	339.7	19.3	320.4	0.96	250
					Average last 3 cycles	0.96	248

Table A-1c

FEED AND PRODUCTS PARTICLE SIZE ANALYSES

HRI Grindability Test No. 536

Particle Size Mesh (Tyler)		-6M Feed				+100M Circulating Load				-100M Test Product			
		Direct		Cumulative		Direct		Cumulative		Direct		Cumulative	
Passing	Retained	Wt g	Wt %	% Retained	% Passing	Wt g	Wt %	% Retained	% Passing	Wt g	Wt %	% Retained	% Passing
6	8	302.01	25.4	25.4	74.6	84.17	9.9	9.9	90.1				
8	10	326.34	27.5	52.9	47.1	105.17	12.4	22.3	77.7				
10	14	172.03	14.5	67.4	32.6	61.83	7.3	29.6	70.4				
14	20	107.46	9.1	76.5	23.5	48.95	5.8	35.4	64.6				
20	28	72.77	6.1	82.6	17.4	47.84	5.6	41.0	59.0				
28	35	47.88	4.0	86.6	13.4	54.00	6.4	47.4	52.6				
35	48	42.22	3.6	90.2	9.8	92.20	10.9	58.3	41.7				
48	65	28.15	2.4	92.6	7.4	161.64	19.1	77.4	22.6				
65	100	20.51	1.7	94.3	5.7	191.90	22.6	100	0				
100	150	68.03	5.7	100	0					34.49	29.2	29.2	70.8
150	200									21.06	17.9	47.1	52.9
200	270									8.68	7.4	54.5	45.5
270	400									18.13	15.4	69.9	30.1
400	PAN									35.44	30.1	100	0
		1187.4				847.7				117.8			

1/ Last three cycles of product.

HAZEN RESEARCH, INC. Table A-1d

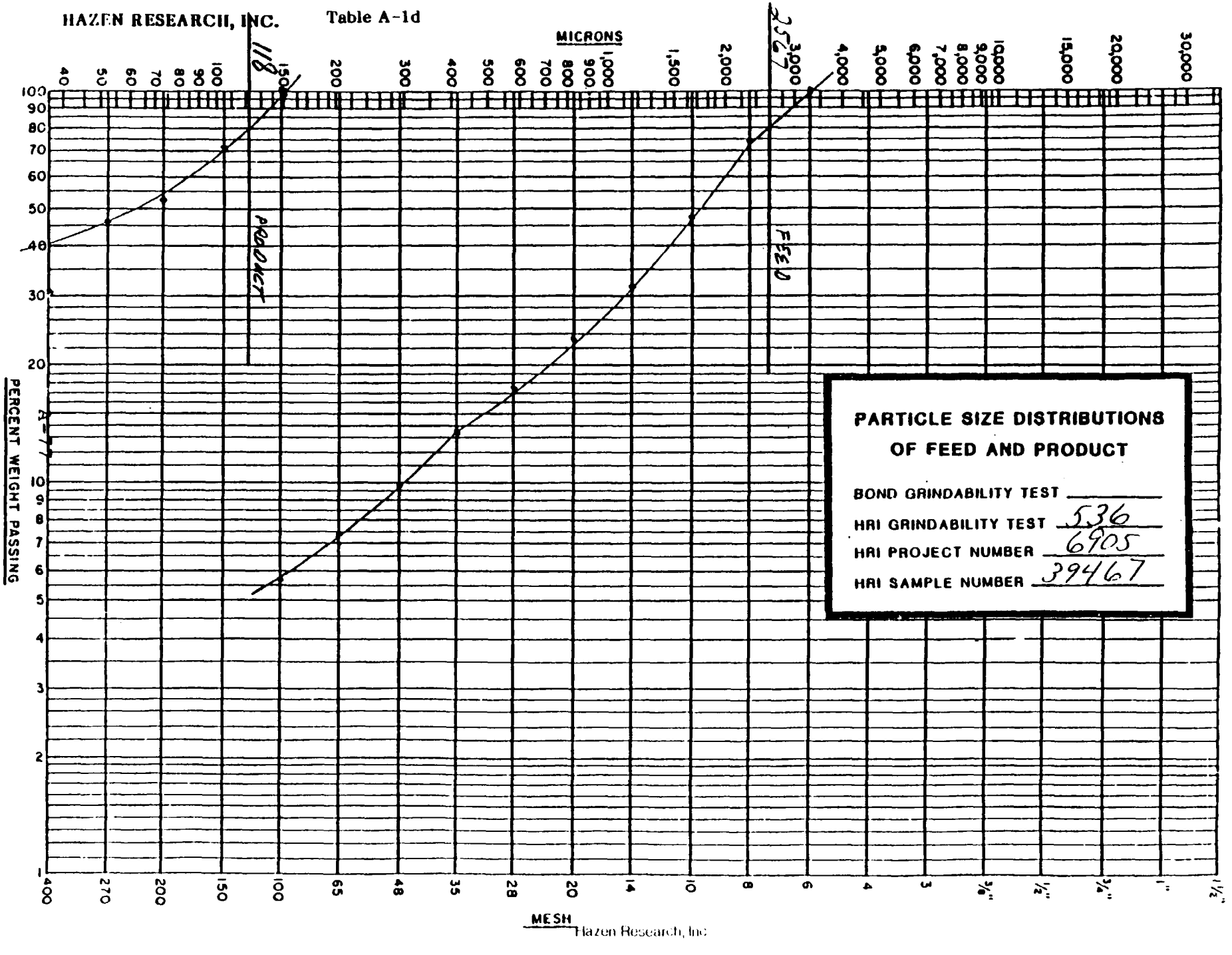


Table A-2a

STANDARD BOND BALL MILL GRINDABILITY
Hazen Research, Inc.

HRI Grindability Test No. 537

Purpose: To determine the ball mill grindability of the test sample in terms of a Bond Work Index number.

Procedure: The equipment and procedure used duplicate the Bond method for determining ball mill work indices.

Sample: Client identified Low Grade "Chat"
The sample (HRI 39468) was stage crushed to minus 6 mesh.
This material was used as the feed for the grindability test.

Results: Calculation of a Bond Work Index

$$W_i = \frac{44.5}{(P_1)^{0.23} \times (Gbp)^{0.82} \times \left(\frac{10}{\sqrt{P_{80}}} - \frac{10}{\sqrt{F_{80}}}\right)}$$

W_i = Bond Work Index, kwhr/ton to 80% minus 100 microns
P_1 = 100% passing size of product 147 microns
Gbp = Grams per revolution 1.08 grams
P_{80} = 80% passing size of product 118 microns
F_{80} = 80% passing size of feed; 2147 microns

W_i = 18.81

Table A-2b

BOND GRINDABILITY TEST RESULTS
Hazen Research, Inc.

HRI Grindability Test No. 537
HRI Project No.: 6905
Date: July 15, 1988

HRI Sample No. 39468
100% mesh of grind: 100
Operator: DAN

Test Conditions and Notes:

Test Feed Weight (700 cc) = 1172.3 g
% Minus 100 M in Feed = 13.2%
Target product weights: + 100 M = 837.4 g
- 100 M = 334.9 g

Test Ball Charge

Nominal Ball Size Inch	No. of Balls	Wt, g
1.50	32	7523
1.25	48	6772
1.00	10	796
0.75	88	3210
0.50	104	1826
Total		20,127

Test Cycle	Number of Revolutions	Product +100M g	Product -100M g	Feed -100M g	Product Produced -100M g	Grams per Revolution	% Circulating Load
1	175	774.7	397.6	154.7	242.9	1.39	195
2	209	866.1	306.2	52.5	253.7	1.21	283
3	240	867.5	304.8	40.4	264.4	1.10	285
4	264	855.5	316.8	40.2	276.6	1.05	270
5	277	840.8	331.5	41.8	289.7	1.05	254
6	277	829.2	343.1	43.8	299.3	1.08	242
7	269	838.9	333.4	45.3	288.1	1.07	252
8	269	838.1	334.2	44.0	290.2	1.08	251
					Average last 3 cycles	1.08	248

Table A-2c

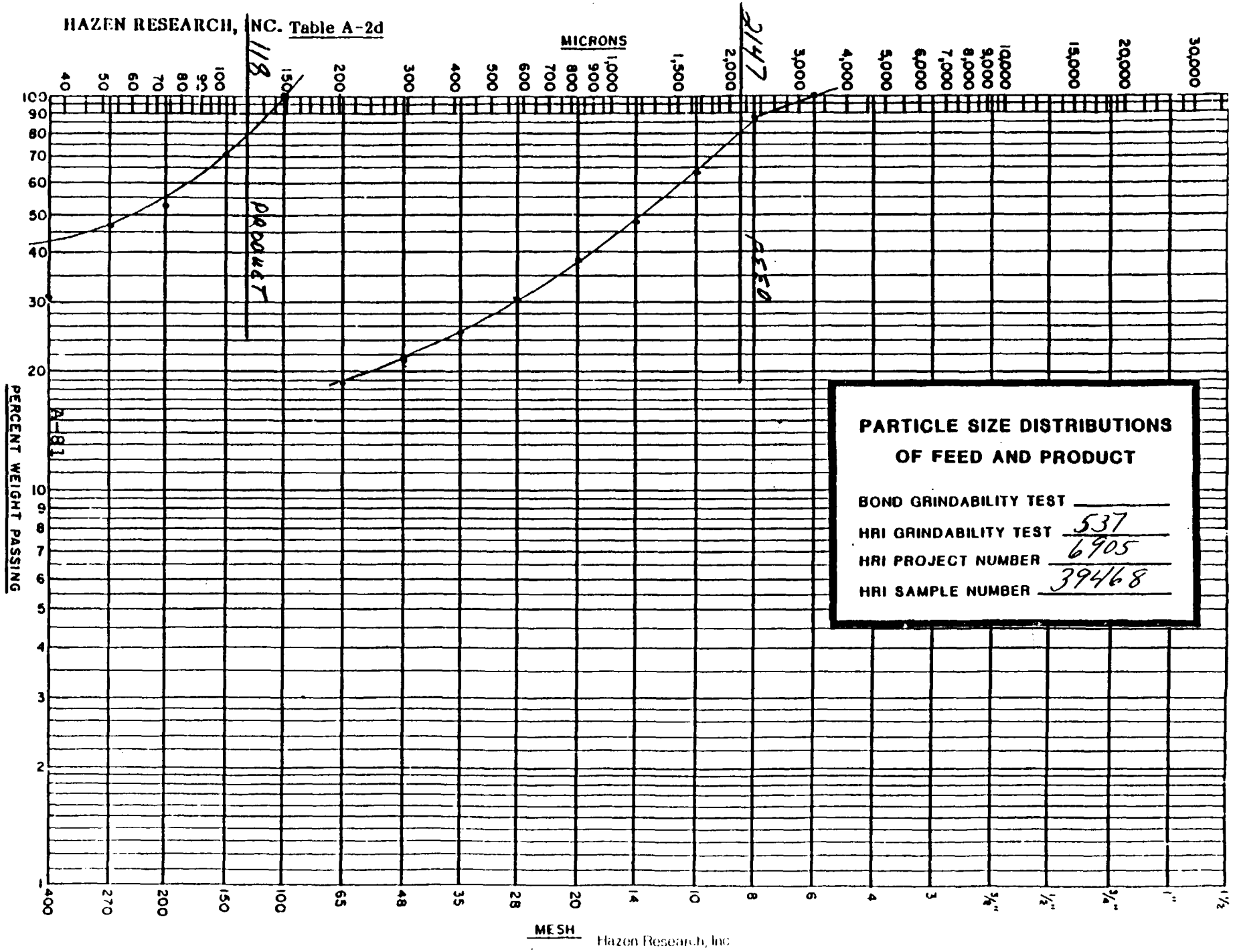
FEED AND PRODUCTS PARTICLE SIZE ANALYSES

HRI Grindability Test No. 537

Particle Size Mesh (Tyler)		-6M Feed				+100M Circulating Load				-100M Test Product			
		Direct		Cumulative		Direct		Cumulative		Direct		Cumulative	
Passing	Retained	Wt g	Wt %	% Retained	% Passing	Wt g	Wt %	% Retained	% Passing	Wt g	Wt %	% Retained	% Passing
6	8	152.12	13.0	13.0	87.0	35.38	4.2	4.2	95.8				
8	10	271.29	23.1	36.1	63.9	68.37	8.2	12.4	87.6				
10	14	176.65	15.1	51.2	48.8	48.73	5.8	18.2	87.8				
14	20	121.44	10.4	61.6	38.4	45.19	5.4	23.6	76.4				
20	28	87.84	7.5	69.1	30.9	48.95	5.9	29.5	70.5				
28	35	61.10	5.2	74.3	25.7	60.36	7.2	36.7	63.3				
35	48	45.12	3.8	78.1	21.9	120.00	14.3	51.0	49.0				
48	65	34.89	3.0	81.1	18.9	222.98	26.6	77.6	22.4				
65	100	67.10	5.7	86.8	13.2	188.14	22.4	100	0				
100	150	154.75	13.2	100	0					42.57	29.6	29.6	70.4
150	200									24.38	17.0	46.6	53.4
200	270									8.52	5.9	52.5	47.5
270	400									23.78	16.5	69.0	31.0
400	PAN									44.62	31.0	100	0
		1172.3				838.1				143.87			

1/ Last three cycles of product.

HAZEN RESEARCH, INC. Table A-2d



PARTICLE SIZE DISTRIBUTIONS OF FEED AND PRODUCT

BOND GRINDABILITY TEST _____

HRI GRINDABILITY TEST 537

HRI PROJECT NUMBER 6905

HRI SAMPLE NUMBER 39468

Hazen Research, Inc

Table A-3

PARTICLE SIZE ANALYSIS

PROJECT: 6905
TEST NO: 40 min Ball Mill Primary Grind, Test 11
% SOLIDS: 65
GRIND TIME: 40 MINUTES
WEIGHT: 2000 GRAMS
SAMPLE: 39467 High Grade Ore

SCREEN SIZE, MESH	MICRON	DIRECT WEIGHT, GRAMS	%	CUMULATIVE WEIGHT, % RET	% PASS
65	208	6.48	4.9	4.9	95.1
100	149	3.16	2.4	7.3	92.7
150	104	16.73	12.7	20.0	80.0
200	75	16.64	12.6	32.7	67.3
270	52	14.87	11.3	44.0	56.0
325	44	7.56	5.7	49.7	50.3
400	37	5.84	4.4	54.2	45.8
500	25	0.00	0.0	54.2	45.8
	20	0.00	0.0	54.2	45.8
	10	0.00	0.0	54.2	45.8
	5	0.00	0.0	54.2	45.8
	PAN	60.34	45.8	100.0	0.0
TOTAL		131.62	100.0		

LOWEST % PASSING ABOVE 80%	92.68	AT	149 MICRONS
HIGHEST % PASSING BELOW 80%	79.97	AT	104 MICRONS
P80 IN MICRONS	104.1		

Table A-4

PARTICLE SIZE ANALYSIS

PROJECT: 6905
TEST NO: 60 min Ball Mill Primary Grind, Test 1 In Ro Tail
% SOLIDS: 65
GRIND TIME: 60 MINUTES
WEIGHT: 2000 GRAMS
SAMPLE: 39467 High Grade Ore

SCREEN SIZE, MESH	MICRON	DIRECT WEIGHT, GRAMS	%	CUMULATIVE WEIGHT, % RET	% PASS
65	208	0.00	0.0	0.0	100.0
100	149	8.69	2.5	2.5	97.5
150	104	15.39	4.4	6.8	93.2
200	75	34.20	9.7	16.5	83.5
270	52	55.68	15.8	32.3	67.7
325	44	18.44	5.2	37.5	62.5
400	37	27.40	7.8	45.3	54.7
PAN		192.80	54.7	100.0	0.0
TOTAL		352.60	100.0		

LOWEST % PASSING ABOVE 80%	83.47	AT	75 MICRONS
HIGHEST % PASSING BELOW 80%	67.68	AT	52 MICRONS
P80 IN MICRONS	69.9		

Table A-5

PARTICLE SIZE ANALYSIS

PROJECT: 6905
TEST NO: 85 min Ball Mill Primary Grind, Test 10
% SOLIDS: 65
GRIND TIME: 85 MINUTES
WEIGHT: 2000 GRAMS
SAMPLE: 39467 High Grade Ore

SCREEN SIZE, MESH	MICRON	DIRECT WEIGHT, GRAMS	%	CUMULATIVE WEIGHT, % RET	% PASS
65	208	0.00	0.0	0.0	100.0
100	149	0.00	0.0	0.0	100.0
150	104	0.00	0.0	0.0	100.0
200	75	5.61	3.7	3.7	96.3
270	52	15.59	10.3	14.0	86.0
325	44	10.17	6.7	20.7	79.3
400	37	8.62	5.7	26.4	73.6
500	25	0.00	0.0	26.4	73.6
	20	0.00	0.0	26.4	73.6
	10	0.00	0.0	26.4	73.6
	5	0.00	0.0	26.4	73.6
	PAN	111.68	73.6	100.0	0.0
TOTAL		151.67	100.0		

LOWEST % PASSING ABOVE 80% 86.02 AT 52 MICRONS
HIGHEST % PASSING BELOW 80% 79.32 AT 44 MICRONS
P80 IN MICRONS 44.8

Table A-6

PARTICLE SIZE ANALYSIS

PROJECT: 6905
TEST NO: 38 min Ball Mill Primary Grind, Test 13
% SOLIDS: 65
GRIND TIME: 38 MINUTES
WEIGHT: 2000 GRAMS
SAMPLE: 39468 Low Grade Ore

SCREEN SIZE, MESH	MICRON	DIRECT WEIGHT, GRAMS	%	CUMULATIVE WEIGHT, % RET	% PASS
65	208	1.79	1.1	1.1	98.9
100	149	2.40	1.5	2.7	97.3
150	104	19.39	12.4	15.1	84.9
200	75	23.17	14.8	29.9	70.1
270	52	21.18	13.5	43.4	56.6
325	44	8.86	5.7	49.1	50.9
400	37	6.93	4.4	53.5	46.5
500	25	0.00	0.0	53.5	46.5
	20	0.00	0.0	53.5	46.5
	10	0.00	0.0	53.5	46.5
	5	0.00	0.0	53.5	46.5
	PAN	72.82	46.5	100.0	0.0
TOTAL		156.54	100.0		

LOWEST % PASSING ABOVE 80% 84.94 AT 104 MICRONS
HIGHEST % PASSING BELOW 80% 70.14 AT 75 MICRONS
P80 IN MICRONS 94.3

Table A-7

PARTICLE SIZE ANALYSIS

PROJECT: 6905
TEST NO: 60 min Ball Mill Primary Grind, Test 2 Zn Ro Tail
% SOLIDS: 65
GRIND TIME: 60 MINUTES
WEIGHT: 2000 GRAMS
SAMPLE: 39468 Low Grade Ore

SCREEN SIZE, MESH	MICRON	DIRECT WEIGHT, GRAMS	%	CUMULATIVE WEIGHT, % RET	% PASS
65	208	0.00	0.0	0.0	100.0
100	149	6.56	1.7	1.7	98.3
150	104	16.43	4.2	5.9	94.1
200	75	34.36	8.8	14.7	85.3
270	52	67.48	17.2	31.9	68.1
325	44	17.97	4.6	36.5	63.5
400	37	19.27	4.9	41.4	58.6
PAN		229.33	58.6	100.0	0.0
TOTAL		391.40	100.0		

LOWEST % PASSING ABOVE 80%	85.35	AT	75 MICRONS
HIGHEST % PASSING BELOW 80%	68.11	AT	52 MICRONS
P80 IN MICRONS	67.9		

Table A-8

PARTICLE SIZE ANALYSIS

PROJECT: 6905
TEST NO: 80 min Ball Mill Primary Grind, Test 12
% SOLIDS: 65
GRIND TIME: 80 MINUTES
WEIGHT: 2000 GRAMS
SAMPLE: 39468 Low Grade Ore

SCREEN SIZE, MESH	MICRON	DIRECT WEIGHT, GRAMS	%	CUMULATIVE WEIGHT, % RET	% PASS
65	208	0.00	0.0	0.0	100.0
100	149	0.00	0.0	0.0	100.0
150	104	0.00	0.0	0.0	100.0
200	75	14.11	11.5	11.5	88.5
270	52	18.31	14.9	26.4	73.6
325	44	9.15	7.5	33.9	66.1
400	37	7.42	6.1	39.9	60.1
500	25	0.00	0.0	39.9	60.1
	20	0.00	0.0	39.9	60.1
	10	0.00	0.0	39.9	60.1
	5	0.00	0.0	39.9	60.1
	PAN	73.65	60.1	100.0	0.0
TOTAL		122.64	100.0		

LOWEST % PASSING ABOVE 80% 88.49 AT 75 MICRONS
HIGHEST % PASSING BELOW 80% 73.56 AT 52 MICRONS
P80 IN MICRONS 61.9

Table A-9a

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 1 PROJECT NO.: 6905

ORE: HRI 39467 High Grade

GRIND: 83.5% -200 mesh, P80 = 69.9

Conditions: Preliminary selective flowsheet test, using conventional Pb/Zn milling conditions

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	ASSAYS, % Cu	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn	DISTRIBUTION, % Cu
1. Pb concentrate	10.32	0.51	51.00	4.50	0.105	39.6	0.3	1.4
2. Pb 1st cleaner tailing	28.24	1.41	5.90	7.30	0.135	12.5	1.5	4.9
3. Zn concentrate	237.64	11.85	0.28	57.00	0.165	5.0	97.0	49.9
4. Zn 1st cleaner tailing	44.91	2.24	1.05	1.20	0.205	3.5	0.4	11.7
5. Zn rougher tailing	1,683.90	83.98	0.31	0.070	0.015	39.3	0.8	32.1
HEAD, CALC'D	2,005.01	100.00	0.66	6.97	0.039	100.0	100.0	100.0
HEAD, DIRECT			0.63	6.18	0.013			
CALCULATED PRODUCTS								
Pb rougher conc	1 and 2	1.92	17.97	6.55	0.13	52.2	1.8	6.2
Zn rougher conc	3 and 4	14.09	0.40	48.13	0.17	8.6	97.3	61.6
Total rougher recovery	1 to 4	16.02	2.51	43.14	0.17	60.7	99.2	67.9

Table A-9b

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 1 PROJECT NO.: 6905

ORE: HRI 39467 High Grade

GRIND: 83.5% -200 mesh, P80 = 69.9

Conditions: Preliminary selective flowsheet test, using conventional Pb/Zn milling conditions

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % S(t)	ASSAYS, % Cd	ASSAYS, % As	DISTRIBUTION, % S(t)	DISTRIBUTION, % Cd	DISTRIBUTION, % As
1. Pb concentrate	10.32	0.51	11.80	0.025	0.001	1.6	0.3	0.9
2. Pb 1st cleaner tailing	28.24	1.41	5.20	0.040	0.003	2.0	1.3	6.0
3. Zn concentrate	237.64	11.85	29.00	0.351	0.001	92.8	97.1	10.1
4. Zn 1st cleaner tailing	44.91	2.24	1.50	0.006	0.003	0.9	0.3	11.5
5. Zn rougher tailing	1,683.90	83.98	0.12	0.001	0.001	2.7	1.0	71.6
HEAD, CALC'D	2,005.01	100.00	3.71	0.043	0.0006	100.0	100.0	100.0
HEAD, DIRECT			3.82	0.038	0.0008			
CALCULATED PRODUCTS								
Pb rougher conc	1 and 2	1.92	6.97	0.04	0.002	3.6	1.6	6.9
Zn rougher conc	3 and 4	14.09	24.63	0.30	0.001	93.7	97.4	21.6
Total rougher recovery	1 to 4	16.02	22.51	0.26	0.001	97.3	99.0	28.4

Table A-11

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 3 PROJECT NO.: 8905

ORE: HRI 39468 Low Grade

GRIND: 65.3 % -200 mesh, P80 = 67.9

Conditions: Improve lead/zinc recoveries by use of sulfidizing agent Na2S in separate oxide rougher stage.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	31.28	1.05	0.570	6.35	[illegible]	[illegible]
2. Zn rougher concentrate	115.80	3.73	0.150	20.50	18.3	41.6
3. Oxide rougher concentrate	106.45	3.27	0.310	4.950	33.4	9.2
4. Magnetic concentrate	9.42	0.47	0.067	0.955	1.4	1.2
5. Oxide rougher tailings	1,787.20	87.48	0.044	1.500	55.2	46.4
HEAD, CALCULATED	2020.15	100.00	0.070	2.83	100.0	100.0
HEAD, DIRECT			0.073	3.70		

CALCULATED PRODUCTS

Sulfide circuit concentrates	1 and 2	4.79	[illegible]	13.4	[illegible]	44.8
Sulfide rougher tailings	1 to 5	95.21	[illegible]	1.331	[illegible]	53.3
Oxide concentrate	No. 3	5.27	0.31	4.95	33.4	9.2
Oxide ro tail 1	4 to 5	87.94	0.044	1.497	55.5	46.5
Oxide + magnetic concentrates	3 and 4	5.74	0.29	4.63	33.9	9.4
Oxide ro tail 2	No. 5	87.48	0.044	1.500	55.2	46.4

Table A-12

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 4 PROJECT NO.: 6505

ORE: HRI 39467 High Grade

GRIND: 63.5 % -200 mesh, P80 = 69.9

Conditions: Improve lead/zinc recoveries by use of sulfidizing agent Na2S in separate oxide rougher stage.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	50.33	1.48	14.500	1.40	34.9	1.3
2. Zn rougher concentrate	312.50	15.43	0.380	43.50	8.9	65.8
3. Oxide rougher concentrate	54.04	1.16	3.300	1.480	13.8	1.2
4. Magnetic concentrate	10.09	0.50	0.345	0.180	0.3	0.0
5. Oxide rougher tailings	1,588.70	78.42	0.160	0.059	13.1	1.7
HEAD, CALCULATED	2025.75	100.00	0.657	6.33	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	16.91	1.34	13.4	33.8	33.1
Sulfide rougher tailings	3 to 5	81.09	0.330	0.373	66.2	9
Oxide concentrate	No. 3	1.16	3.30	0.48	13.8	0.2
Oxide ro tail 1	4 to 5	78.92	0.161	0.060	19.4	0.7
Oxide + magnetic concentrates	3 and 4	1.66	3.07	0.44	17.1	0.2
Oxide ro tail 2	No. 5	78.42	0.160	0.059	13.1	0.7

Table A-13

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 5 PROJECT NO.: 6305

ORE: HRI 39467 High Grade

GRIND: 83.5 % -200 mesh, P80 = 89.9

Conditions: Improve lead/zinc recoveries by use of chelating agent 8-hydroxy quinoline in separate oxide rougher stage.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	ASSAYS, % Cu	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn	DISTRIBUTION, % Cu
1. Pb rougher concentrate	46.38	2.33	15.500	7.55	0.140	53.7	2.4	11.3
2. Zn rougher concentrate	325.85	16.39	0.265	43.50	0.110	6.7	96.7	55.4
3. Oxide rougher concentrate 1	17.75	0.89	1.600	1.550	0.195	2.2	0.3	2.3
4. Oxide rougher concentrate 2	43.31	2.17	0.985	0.630	0.190	3.3	0.3	13.9
5. Oxide rougher tailings	1,559.70	78.20	0.255	0.049	0.005	31.0	0.5	13.3
HEAD, CALCULATED	1994.57	100.00	0.643	7.37	0.032	100.0	100.0	100.0
HEAD, DIRECT			0.630	6.16	0.013			
CALCULATED PRODUCTS								
Sulfide circuit concentrates	1 and 2	18.74	3.19	19.0	0.11	93.5	99.1	66.7
Sulfide rougher tailings	3 to 5	81.26	0.133	0.081	0.013	16.5	0.9	33.3
Oxide concentrate 1	No. 3	0.89	1.60	1.55	0.20	2.2	0.2	5.3
Oxide ro tail 1	4 to 5	80.37	0.275	0.065	0.010	34.3	0.7	13.1
Oxide concentrate 1 + 2	3 and 4	3.06	1.16	0.90	0.22	5.5	0.4	21.1
Oxide ro tail 2	No. 5	78.20	0.255	0.049	0.005	31.0	0.5	13.3

Table A-14

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 1 PROJECT NO.: 1905

ORE: HRI 39467 High Grade

GRIND: 83.5 % -200 mesh, P80 = 89.3

Conditions: Improve lead/zinc recoveries by use of chelating agents alkylhydroximate and dimethylglyoxime in separate oxide rougher stages.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	ASSAYS, % Cu	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn	DISTRIBUTION, % Cu
1. Pb rougher concentrate	35.31	1.72	15.00	7.00	0.145	38.1	1.5	11.7
2. Zn rougher concentrate	350.04	17.22	0.255	43.00	0.105	6.2	56.5	53.5
3. Oxide rougher concentrate 1	41.81	2.06	1.100	0.385	0.160	3.2	1.3	5.7
4. Oxide rougher concentrate 2	21.01	1.03	0.875	1.000	0.180	1.4	0.1	5.5
5. Oxide rougher concentrate 3	37.25	1.83	1.05	0.800	0.155	3.7	0.2	5.4
6. Oxide rougher tailings	1,536.90	75.12	0.265	0.045	0.005	28.3	0.4	11.1
HEAD, CALCULATED	2032.32	100.00	0.705	7.68	0.134	100.0	100.0	100.0
HEAD, DIRECT			0.630	6.18	0.010			
CALCULATED PRODUCTS								
Sulfide circuit concentrates	1 and 2	18.93	8.27	18.1	0.1	24.4	38.0	15.2
Sulfide rougher tailings	3 to 6	80.05	0.314	0.099	0.015	75.6	1.0	74.8
Oxide concentrate 1	No. 3	2.06	1.10	0.39	0.16	3.2	0.3	5.7
Oxide rougher tail 1	4 to 6	77.99	0.293	0.075	0.011	32.4	0.8	35.0
Oxide concentrates 1 & 2	3 and 4	3.09	1.06	0.39	0.17	4.6	0.4	15.2
Oxide rougher tail 2	5 to 6	75.95	0.284	0.063	0.009	31.0	0.6	19.5
Oxide concentrates 1 to 3	3 to 5	4.92	1.06	0.92	0.16	7.4	0.6	33.7
Oxide rougher tail 3	No. 6	75.12	0.265	0.045	0.005	28.3	0.4	11.1

Table A-15

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 7 PROJECT NO.: 6305

ORE: HRI 75/67 High Grade

GRIND: 33.5 % -200 mesh, P80 = 69.9

Conditions: Improve lead/zinc recoveries by use of chelating and sulfidizing agents in separate oxide rougher stages.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	60.73	3.02	12.00	7.65	38.0	3.2
2. Zn rougher concentrate	289.46	14.39	0.280	48.00	6.2	95.7
3. Oxide rougher concentrate 1	379.12	18.84	0.650	0.210	18.9	0.5
4. Oxide rougher concentrate 2	38.70	1.92	0.320	0.240	0.9	0.1
5. Oxide rougher concentrate 3	34.37	1.71	2.95	0.280	7.8	0.1
6. Oxide rougher tailings	1,209.60	60.12	0.110	0.045	10.2	0.4
HEAD, CALCULATED	2012.04	100.00	0.648	7.21	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	17.41	2.31	41.0	62.2	98.9
Sulfide rougher tailings	3 to 6	82.59	0.297	0.092	37.8	1.1
Oxide concentrate 1	No. 3	18.84	0.65	0.21	18.9	0.5
Oxide ro tail 1	4 to 6	63.75	0.192	0.057	18.9	0.5
Oxide concentrates 1 & 2	3 and 4	20.77	0.62	0.21	19.9	0.6
Oxide ro tail 2	5 to 6	61.83	0.188	0.051	18.0	0.4
Oxide concentrates 1 to 3	3 to 5	22.47	0.80	0.22	27.6	0.7
Oxide ro tail 3	No. 6	60.12	0.110	0.045	10.2	0.4

Table A-16

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 5 PROJECT NO.: 6905

ORE: HRI 39468 Low Grade

GRIND: 85.3 % -200 mesh, P80 = 67.5

Conditions: Improve lead/zinc recoveries by use of chelating and sulfidizing agents in separate oxide rougher stages.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	164.55	8.22	0.23	5.40	26.4	15.4
2. Zn rougher concentrate	175.42	8.76	0.150	15.00	18.3	45.5
3. Oxide rougher concentrate 1	233.36	11.68	0.115	2.500	18.7	10.5
4. Oxide rougher concentrate 2	179.20	8.95	0.074	2.350	9.2	7.0
5. Oxide rougher concentrate 3	83.67	4.18	0.079	2.750	4.6	4.0
6. Oxide rougher tailings	1,165.50	58.21	0.028	0.880	22.7	17.7
HEAD, CALCULATED	2002.3	100.00	0.072	2.89	100.0	100.0
HEAD, DIRECT			0.073	2.70		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	16.98	0.19	10.35	44.7	60.8
Sulfide rougher tailings	3 to 6	83.02	0.048	1.364	55.3	39.2
Oxide concentrate 1	No. 3	11.68	0.12	2.60	18.7	10.5
Oxide ro tail 1	4 to 6	71.34	0.037	1.161	36.6	28.7
Oxide concentrates 1 & 2	3 and 4	20.63	0.10	2.45	28.0	17.5
Oxide ro tail 2	5 to 6	62.39	0.031	1.005	27.3	21.7
Oxide concentrates 1 to 3	3 to 5	24.81	0.09	2.50	32.6	21.5
Oxide ro tail 3	No. 6	53.21	0.028	0.880	22.7	17.7

Table A-17

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 3 PROJECT NO.: 5905

ORE: HRI 39468 Low Grade

GRIND: ~100 % -200 mesh

Conditions: Improve lead/zinc recoveries by use of chelating and sulfidizing agents in separate oxide rougher stages.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	194.11	9.70	0.19	4.20	27.0	15.2
2. Zn rougher concentrate	132.33	6.61	0.120	16.00	11.9	38.4
3. Oxide rougher concentrate 1	376.14	18.80	0.064	2.050	18.1	14.3
4. Oxide rougher concentrate 2	460.41	23.01	0.059	1.700	20.4	14.5
5. Oxide rougher concentrate 3	58.10	2.90	0.150	2.750	6.5	3.0
6. Oxide rougher concentrate 4	28.47	1.42	0.089	2.55	1.9	1.3
7. Oxide rougher tailings	751.10	37.54	0.025	0.880	14.1	12.3
HEAD, CALCULATED	2000.66	100.00	0.067	2.89	100.0	100.0
HEAD, DIRECT			0.073	2.70		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	16.32	0.16	8.98	38.9	54.5
Sulfide rougher tailings	3 to 7	83.68	0.049	1.462	61.1	45.5
Oxide concentrate 1	No. 3	18.80	0.06	2.05	18.1	14.3
Oxide ro tail 1	4 to 7	64.88	0.044	1.291	43.0	31.2
Oxide concentrates 1 & 2	3 and 4	41.81	0.06	1.86	38.5	28.9
Oxide ro tail 2	5 to 7	41.87	0.036	1.066	22.6	16.6
Oxide concentrates 1 to 3	3 to 5	44.72	0.07	1.92	45.1	31.9
Oxide ro tail 3	6 to 7	38.37	0.027	0.941	16.0	13.6
Oxide concentrates 1 to 4	3 to 6	46.14	0.07	1.93	47.0	33.2

Table A-18

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 10 PROJECT NO.: 5905

ORE: HRI 39467 High Grade

GRIND: 55.3 % -200 mesh, P80 = 44.8

Conditions: Improve lead/zinc recoveries by use of chelating and sulfidizing agents in separate oxide rougher stages.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	53.11	2.65	13.00	1.05	51.0	1.4
2. Zn rougher concentrate	317.13	15.85	0.385	41.50	9.3	95.3
3. Oxide rougher concentrate 1	309.52	15.47	0.875	0.300	21.7	0.7
4. Oxide rougher concentrate 2	67.99	3.40	0.850	0.440	4.6	0.2
5. Oxide rougher concentrate 3	15.54	0.78	1.400	0.585	1.7	0.1
6. Oxide rougher concentrate 4	49.01	2.45	1.450	0.465	5.7	0.2
7. Oxide rougher tailings	1,189.00	59.41	0.055	0.020	5.2	0.2
HEAD, CALCULATED	2001.3	100.00	0.624	5.83	100.0	100.0
HEAD, DIRECT			0.630	5.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	18.50	2.05	35.41	60.8	98.7
Sulfide rougher tailings	3 to 7	81.50	0.300	0.109	39.2	1.3
Oxide concentrate 1	No. 3	15.47	0.88	0.30	21.7	0.7
Oxide ro tail 1	4 to 7	66.03	0.166	0.065	17.5	0.6
Oxide concentrates 1 & 2	3 and 4	18.86	0.88	0.33	25.5	0.9
Oxide ro tail 2	5 to 7	62.64	0.126	0.044	12.7	0.4
Oxide concentrates 1 to 3	3 to 5	19.64	0.90	0.34	28.3	1.0
Oxide ro tail 3	6 to 7	61.86	0.110	0.038	10.9	0.3
Oxide concentrates 1 to 4	3 to 6	22.09	0.96	0.35	34.0	1.1

Table A-19

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 11 PROJECT NO.: 6905

ORE: HRI 39467 High Grade

GRIND: 67.3 % -200 mesh, P80 = 104.1

Conditions: Improve lead/zinc recoveries by use of chelating and sulfidizing agents in separate oxide rougher stages.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	37.75	1.89	20.50	3.15	55.6	1.3
2. Zn rougher concentrate	337.37	16.91	0.300	43.50	7.3	95.7
3. Oxide rougher concentrate 1	206.77	10.36	0.625	0.305	9.3	0.4
4. Oxide rougher concentrate 2	68.93	3.45	1.150	0.325	5.0	0.1
5. Oxide rougher concentrate 3	5.80	0.29	14.000	0.690	5.3	0.0
6. Oxide rougher tailings	1,359.00	68.10	0.185	0.055	18.0	0.5
HEAD, CALCULATED	1995.63	100.00	0.698	7.61	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	18.80	2.33	40.04	62.6	99.0
Sulfide rougher tailings	3 to 6	81.20	0.320	0.097	37.2	1.0
Oxide concentrate 1	No. 3	10.36	0.63	0.31	9.3	0.4
Oxide ro tail 1	4 to 6	70.84	0.275	0.067	27.9	0.6
Oxide concentrates 1 & 2	3 and 4	12.81	0.73	0.31	13.3	0.5
Oxide ro tail 2	5 to 6	68.39	0.244	0.058	23.9	0.5
Oxide concentrates 1 to 3	3 to 5	13.10	1.02	0.32	19.1	0.5
Oxide ro tail 3	No. 6	68.10	0.185	0.055	18.0	0.5

Table A-20

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 12 PROJECT NO.: 6905

ORE: HRI 39468 Low Grade

GRIND: 38.5 % -200 mesh. P80 = 61.3

Conditions: Improve lead/zinc recoveries by use of chelating and sulfidizing agents in separate oxide rougher stages.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	43.69	2.25	1.10	6.85	11.3	5.8
2. Zn rougher concentrate	116.57	6.00	0.225	37.50	12.5	47.7
3. Oxide rougher concentrate 1	135.83	7.00	0.180	3.550	11.6	7.2
4. Oxide rougher concentrate 2	120.22	6.19	0.165	2.650	9.4	4.7
5. Oxide rougher concentrate 3	50.13	2.58	0.295	3.700	7.0	3.8
6. Oxide rougher tailings	1,474.90	75.97	0.052	1.450	36.5	31.8
HEAD, CALCULATED	1941.34	100.00	0.108	3.46	100.0	100.0
HEAD, DIRECT			0.073	2.70		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	8.26	0.46	22.42	35.4	53.5
Sulfide rougher tailings	3 to 6	91.74	0.076	1.754	64.6	46.5
Oxide concentrate 1	No. 3	7.00	0.18	3.55	11.6	7.2
Oxide ro tail 1	4 to 6	84.75	0.068	1.606	53.0	39.3
Oxide concentrates 1 & 2	3 and 4	13.19	0.17	3.13	21.1	11.9
Oxide ro tail 2	5 to 6	78.56	0.060	1.524	43.6	34.6
Oxide concentrates 1 to 3	3 to 5	15.77	0.19	3.22	28.1	14.7
Oxide ro tail 3	No. 6	75.97	0.052	1.450	36.5	31.8

Table A-21

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 13 PROJECT NO.: 6905

ORE: HRI 39468 Low Grade

GRIND: 70.1 % -200 mesh, P80 = 94.3

Conditions: Improve lead/zinc recoveries by use of chelating and sulfidizing agents in separate oxide rougher stages.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	33.81	1.70	0.58	7.50	12.3	4.4
2. Zn rougher concentrate	38.32	4.43	0.195	25.50	11.4	40.3
3. Oxide rougher concentrate 1	109.33	5.49	0.190	4.200	13.7	7.9
4. Oxide rougher concentrate 2	105.26	5.28	0.235	3.450	16.3	5.3
5. Oxide rougher concentrate 3	14.82	0.74	0.360	6.550	3.5	1.7
6. Oxide rougher tailings	1,640.30	82.35	0.039	1.400	42.2	33.5
HEAD, CALCULATED	1991.9	100.00	0.076	2.92	100.0	100.0
HEAD, DIRECT			0.073	2.70		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	6.13	0.30	31.24	24.3	44.6
Sulfide rougher tailings	3 to 6	93.87	0.061	1.720	75.7	55.4
Oxide concentrate 1	No. 3	5.49	0.19	4.20	13.7	7.9
Oxide ro tail 1	4 to 6	88.38	0.053	1.566	62.0	47.4
Oxide concentrates 1 & 2	3 and 4	10.78	0.21	3.83	30.0	14.2
Oxide ro tail 2	5 to 6	83.09	0.042	1.446	45.7	41.2
Oxide concentrates 1 to 3	3 to 5	11.52	0.22	4.01	33.5	15.8
Oxide ro tail 3	No. 6	82.35	0.039	1.400	42.2	33.5

TAble A-22

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 14 PROJECT NO.: 6905

ORE: HRI 39849-2 Minus 20-mesh fraction of Low Grade chat sample 07-01

GRIND: ~80 % -200 mesh

Conditions: Test 10 (advanced sulfidization of Test 7 to 2nd oxide rougher, and used additional xanthate in 3rd oxide rougher.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	46.00	9.23	0.275	1.20	12.9	12.1
2. Zn rougher concentrate	62.70	12.58	0.450	5.85	28.7	90.4
3. Oxide rougher concentrate 1	35.70	7.16	0.370	0.345	13.4	2.7
4. Oxide rougher concentrate 2	35.56	7.13	0.305	0.155	11.0	1.2
5. Oxide rougher concentrate 3	7.19	1.44	0.535	0.220	3.9	0.3
6. Oxide rougher tailings	311.33	62.46	0.095	0.047	30.1	3.2
HEAD, CALCULATED	498.48	100.00	0.197	0.915	100.0	100.0
HEAD, DIRECT			0.260	0.974		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	21.81	0.38	3.88	41.6	92.5
Sulfide rougher tailings	3 to 6	78.19	0.147	0.087	58.4	7.5
Oxide concentrate 1	No. 3	7.16	0.37	0.35	13.4	2.7
Oxide ro tail 1	4 to 6	71.03	0.125	0.061	45.0	4.8
Oxide concentrates 1 & 2	3 and 4	14.30	0.34	0.25	24.5	3.9
Oxide ro tail 2	5 to 6	63.90	0.105	0.051	34.0	3.6
Oxide concentrates 1 to 3	3 to 5	15.74	0.36	0.25	28.4	4.3
Oxide ro tail 3	No. 6	62.46	0.095	0.047	30.1	3.2

Table A-23

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 15 PROJECT NO.: 6905

ORE: Composite of 71.5 wt % high grade ore and 28.5% minus 28-mesh low grade chat
High grade ore = HRI 39467 , Chat (01-02) = HRI 39849-1

GRIND: 85 % -200 mesh

Conditions: Test 10 conditions.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	65.47	3.41	7.80	6.85	48.8	4.6
2. Zn rougher concentrate	215.74	11.25	0.370	42.50	7.6	93.7
3. Oxide rougher concentrate 1-3	183.18	9.55	0.505	0.385	8.8	0.7
4. Oxide rougher tailings	1,453.10	75.78	0.250	0.070	34.7	1.0
HEAD, CALCULATED	1917.49	100.00	0.546	5.11	100.0	100.0
HEAD, DIRECT			0.465	4.42		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	14.67	2.10	34.20	56.4	98.2
Sulfide rougher tailings	3 to 6	85.33	0.279	0.105	43.6	1.8
Oxide concentrates 1 — 3	No. 3	9.55	0.51	0.39	8.8	0.7
Oxide ro tail 3	4 to 6	75.78	0.250	0.070	34.7	1.0

Table A-24

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 16 PROJECT NO.: 6905

ORE: HRI 39467 High Grade ore

GRIND: 75.3 % -200 mesh

Conditions: Deleted 81469C chelating agent (and its corresponding 5 min conditioning time). Test 10 conditions.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	39.98	4.04	11.50	24.50	60.3	12.7
2. Zn rougher concentrate	133.11	13.45	0.255	50.00	4.5	86.0
3. Oxide rougher concentrate 1	16.59	1.68	1.300	2.050	2.8	0.4
4. Oxide rougher concentrate 2	12.72	1.28	0.885	0.455	1.5	0.1
5. Oxide rougher concentrate 3	7.74	0.78	1.300	0.615	1.3	0.1
6. Oxide rougher tailings	779.80	78.77	0.290	0.071	29.6	0.7
HEAD, CALCULATED	989.94	100.00	0.770	7.81	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	17.48	2.85	44.11	64.7	98.7
Sulfide rougher tailings	3 to 6	82.52	0.329	0.122	35.3	1.3
Oxide concentrate 1	No. 3	1.68	1.30	2.05	2.8	0.4
Oxide ro tail 1	4 to 6	80.84	0.309	0.082	32.4	0.9
Oxide concentrates 1 & 2	3 and 4	2.96	1.12	1.36	4.3	0.5
Oxide ro tail 2	5 to 6	79.55	0.300	0.076	31.0	0.8
Oxide concentrates 1 to 3	3 to 5	3.74	1.16	1.20	5.6	0.6
Oxide ro tail 3	No. 6	78.77	0.290	0.071	29.6	0.7

Table A-25

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 17 PROJECT NO.: 6905

ORE: HRI 39467 High Grade ore

GRIND: 75.3 % -200 mesh

Conditions: Deleted R-6410 collector (and its corresponding 10 min conditioning time). Test 10 conditions.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	31.64	3.34	11.00	23.50	58.7	10.8
2. Zn rougher concentrate	121.00	12.75	0.305	50.00	6.2	88.0
3. Oxide rougher concentrate 1	72.21	7.61	0.320	0.295	3.9	0.3
4. Oxide rougher concentrate 2	26.82	2.83	0.550	0.315	2.5	0.1
5. Oxide rougher concentrate 3	32.18	3.39	0.520	0.255	2.8	0.1
6. Oxide rougher tailings	664.80	70.08	0.230	0.065	25.8	0.6
HEAD, CALCULATED	948.65	100.00	0.625	7.25	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	16.09	2.52	44.51	65.0	98.8
Sulfide rougher tailings	3 to 6	83.91	0.261	0.102	35.0	1.2
Oxide concentrate 1	No. 3	7.61	0.32	0.30	3.9	0.3
Oxide ro tail 1	4 to 6	76.30	0.255	0.083	31.1	0.9
Oxide concentrates 1 & 2	3 and 4	10.44	0.38	0.30	6.4	0.4
Oxide ro tail 2	5 to 6	73.47	0.243	0.074	28.6	0.7
Oxide concentrates 1 to 3	3 to 5	13.83	0.42	0.29	9.2	0.6
Oxide ro tail 3	No. 6	70.08	0.230	0.065	25.8	0.6

Table A-26

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 18 PROJECT NO.: 6905

ORE: HRI 39467 High Grade ore

GRIND: 75.3 % -200 mesh

Conditions: In oxide rougher stage 2, sulfidized first, then added PEO (reversed from Test 10).

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	34.97	3.57	10.50	18.00	65.2	9.9
2. Zn rougher concentrate	113.25	11.56	0.270	50.00	5.4	89.3
3. Oxide rougher concentrate 1	94.89	9.69	0.405	0.190	6.8	0.3
4. Oxide rougher concentrate 2	40.79	4.17	0.415	0.135	3.0	0.1
5. Oxide rougher concentrate 3	20.04	2.05	0.625	0.200	2.2	0.1
6. Oxide rougher tailings	675.40	68.96	0.145	0.033	17.4	0.4
HEAD, CALCULATED	979.34	100.00	0.575	6.48	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	15.13	2.68	42.45	70.6	99.2
Sulfide rougher tailings	3 to 6	84.87	0.200	0.060	29.4	0.8
Oxide concentrate 1	No. 3	9.69	0.41	0.19	6.8	0.3
Oxide ro tail 1	4 to 6	75.18	0.173	0.043	22.6	0.5
Oxide concentrates 1 & 2	3 and 4	13.85	0.41	0.17	9.8	0.4
Oxide ro tail 2	5 to 6	71.01	0.159	0.038	19.6	0.4
Oxide concentrates 1 to 3	3 to 5	15.90	0.44	0.18	12.0	0.4
Oxide ro tail 3	No. 6	68.96	0.145	0.033	17.4	0.4

Table A-27

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 19 PROJECT NO.: 6905

ORE: HRI 39467 High Grade ore

GRIND: 75.3 % -200 mesh

Conditions: Sulfidized at start of oxide stages, then repeated Test 10 oxide circuit sequence.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	27.00	2.71	14.00	16.50	58.9	6.4
2. Zn rougher concentrate	128.04	12.85	0.275	50.00	5.5	92.5
3. Oxide rougher concentrate 1	287.96	28.90	0.275	0.110	12.3	0.5
4. Oxide rougher concentrate 2	109.84	11.02	0.635	0.190	10.9	0.3
5. Oxide rougher concentrate 3	71.13	7.14	0.515	0.160	5.7	0.2
6. Oxide rougher tailings	372.36	37.37	0.115	0.025	6.7	0.1
HEAD, CALCULATED	996.33	100.00	0.644	6.95	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	15.56	2.67	44.17	64.4	98.9
Sulfide rougher tailings	3 to 6	84.44	0.271	0.087	35.6	1.1
Oxide concentrate 1	No. 3	28.90	0.28	0.11	12.3	0.5
Oxide ro tail 1	4 to 6	55.54	0.270	0.075	23.3	0.6
Oxide concentrates 1 & 2	3 and 4	39.93	0.37	0.13	23.2	0.8
Oxide ro tail 2	5 to 6	44.51	0.179	0.047	12.4	0.3
Oxide concentrates 1 to 3	3 to 5	47.07	0.40	0.14	28.9	0.9
Oxide ro tail 3	No. 6	37.37	0.115	0.025	6.7	0.1

Table A-28

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 20 PROJECT NO.: 6905

ORE: HRI 39467 High Grade ore

GRIND: 75.3 % -200 mesh

Conditions: At start of the oxide circuit, sulfidized simultaneously with the 81469C chelating agent.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	33.81	3.38	12.50	17.00	62.7	7.8
2. Zn rougher concentrate	133.31	13.33	0.205	50.50	4.1	91.0
3. Oxide rougher concentrate 1	112.50	11.25	0.595	0.200	9.9	0.3
4. Oxide rougher concentrate 2	184.44	18.45	0.430	0.210	11.8	0.5
5. Oxide rougher concentrate 3	42.58	4.26	0.495	0.270	3.1	0.2
6. Oxide rougher tailings	493.14	49.32	0.115	0.029	8.4	0.2
HEAD, CALCULATED	999.78	100.00	0.674	7.40	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	16.72	2.69	43.72	66.8	98.8
Sulfide rougher tailings	3 to 6	83.28	0.269	0.105	33.2	1.2
Oxide concentrate 1	No. 3	11.25	0.60	0.20	9.9	0.3
Oxide ro tail 1	4 to 6	72.03	0.218	0.090	23.3	0.9
Oxide concentrates 1 & 2	3 and 4	29.70	0.49	0.21	21.7	0.8
Oxide ro tail 2	5 to 6	53.58	0.145	0.048	11.5	0.3
Oxide concentrates 1 to 3	3 to 5	33.96	0.49	0.21	24.8	1.0
Oxide ro tail 3	No. 6	49.32	0.115	0.029	8.4	0.2

Table A-29

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 21 PROJECT NO.: 6905

ORE: HRI 39467 High Grade ore

GRIND: 75.3 % -200 mesh

Conditions: In oxide rougher stage 1, sulfidized simultaneously with the R-6410 collector. Test 10 conditions.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	28.21	2.84	13.50	20.00	57.6	7.7
2. Zn rougher concentrate	158.99	16.02	0.355	42.50	8.5	91.7
3. Oxide rougher concentrate 1	24.89	2.51	0.655	0.260	2.5	0.1
4. Oxide rougher concentrate 2	16.21	1.63	1.350	0.370	3.3	0.1
5. Oxide rougher concentrate 3	15.06	1.52	1.600	0.435	3.6	0.1
6. Oxide rougher tailings	749.30	75.48	0.215	0.040	24.4	0.4
HEAD, CALCULATED	992.66	100.00	0.666	7.42	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	18.86	2.34	39.11	66.2	99.3
Sulfide rougher tailings	3 to 6	81.14	0.277	0.061	33.8	0.7
Oxide concentrate 1	No. 3	2.51	0.66	0.26	2.5	0.1
Oxide ro tail 1	4 to 6	78.63	0.265	0.054	31.3	0.6
Oxide concentrates 1 & 2	3 and 4	4.14	0.93	0.30	5.8	0.2
Oxide ro tail 2	5 to 6	77.00	0.242	0.048	28.0	0.5
Oxide concentrates 1 to 3	3 to 5	5.66	1.11	0.34	9.4	0.3
Oxide ro tail 3	No. 6	75.48	0.215	0.040	24.4	0.4

Table A-30

FLOTATION TEST METALLURGICAL RESULTS

TEST NO.: 22 PROJECT NO.: 6905

ORE: HRI 39467 High Grade ore

GRIND: 75.3 % -200 mesh

Conditions: In oxide rougher stage 2, sulfidized simultaneously with the PEO. Test 10 conditions.

PRODUCT	WEIGHT g	WEIGHT %	ASSAYS, % Pb	ASSAYS, % Zn	DISTRIBUTION, % Pb	DISTRIBUTION, % Zn
1. Pb rougher concentrate	48.76	4.92	7.45	14.00	62.3	3.8
2. Zn rougher concentrate	135.57	13.68	0.255	46.00	5.9	89.5
3. Oxide rougher concentrate 1	96.67	9.76	0.375	0.230	6.2	0.3
4. Oxide rougher concentrate 2	27.93	2.82	0.570	0.215	2.7	0.1
5. Oxide rougher concentrate 3	11.35	1.15	0.765	0.310	1.5	0.1
6. Oxide rougher tailings	670.62	67.68	0.185	0.029	21.3	0.3
HEAD, CALCULATED	990.9	100.00	0.588	7.03	100.0	100.0
HEAD, DIRECT			0.630	6.18		
CALCULATED PRODUCTS						
Sulfide circuit concentrates	1 and 2	18.60	2.16	37.54	68.3	99.3
Sulfide rougher tailings	3 to 6	81.40	0.229	0.063	31.7	0.7
Oxide concentrate 1	No. 3	9.76	0.38	0.23	6.2	0.3
Oxide ro tail 1	4 to 6	71.64	0.209	0.041	25.5	0.4
Oxide concentrates 1 & 2	3 and 4	12.57	0.42	0.23	9.0	0.4
Oxide ro tail 2	5 to 6	68.82	0.195	0.034	22.8	0.3
Oxide concentrates 1 to 3	3 to 5	13.72	0.45	0.23	10.4	0.5
Oxide ro tail 3	No. 6	67.68	0.185	0.029	21.3	0.3

Table A-31

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 1 Project No. 3905 Operator [illegible] Date [illegible]/88

Purpose [illegible] high grade sample.

Procedure

Feed 2000g [illegible] minus 10-mesh ore HRI [illegible]

Grind 60 min at 65% solids = 87.5% -200 mesh.

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX343	MIBC/[illegible]	Grind	Cond	Froth	
Grind	1.5						30			[illegible]
Pb Rougher (1)	1.5	1.0	0.33	0.030	0.01	0.020/—		5+1	3	9.0
(2)				0.015	—			2	3	
	Calcine									
Pb 1st Cleaner	0.25	0.10	0.01	0.01				3	3	9.5
~~Pb 2nd Cleaner~~										
	Calcine	$CuSO_4$	M-801	AX343						
Zn Condition	5.7	1.5						5		[illegible]
Zn Rougher (1)			.02	.02		.015/0.5		2	3	10.0
(2)			.02	.02		.005/.005		2	3	10.1
(3)			.02	.02		—/.005		1	3	10.1/10.8
Zn 1st Cleaner	0.50		0.01			—		2	5	11.0

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-32

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 2 Project No. 6905 Operator R. Park Date 7/21/88

Purpose Preliminary selective flowsheet test, low grade sample.

Procedure

Feed 2000g of -10 mesh ore, HRI 39468, low grade

Grind 60 min at 65% solids = 85.3% -100 mesh

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	MIBC	R242	AX325	MIBC/R242	Grind	Cond	Froth	
Grind	3.0						60			9.20
Pb Rougher (1)		1.0	0.33	0.03	0.01	0.015/–		5+1	3	9.00
(2)				0.02	–			1	3	
	Cal/OxL									
Pb 1st Cleaner	0.10	0.10	0.01					3	2	9.5
	Cal/OxL	$CuSO_4$	MIBC	AX342						
Zn Conditioner		1.0						5		11
Zn Rougher (1)			0.02	0.02		0.010/0.020		1	3	11
(2)			0.02	0.02				1	3	10.4
(3)			0.03	–				1	3	11.0
Zn 1st Cleaner	0.10		0.01					1	5	11.0
Zn 2nd Cleaner	0.10		0.005					1	3	11.0

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-33

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 3 Project No. 6905 Operator R. Rick Date 8/1/88

Purpose Add sulfidization to selective flotation of Test 2. Low grade ore.

Procedure

Feed HRI 30468 Low grade, 2000g of minus 6 mesh ore.

Grind 60 min. at 65% solids in the Denver lab steel ball mill.

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R-242	AX325	~~R-250~~ 0.006	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				60			9.6
Pb Rougher				0.030	0.01	~~0.024~~ 0.006		2	3	9.6
	$CaCl_2$	$CuSO_4$	M-Hexyl	AX343	Na_2S					
Zn Conditioner	3.1	1.5						5		11.15
Zn Rougher (1)	–		0.02	0.02		~~0.030~~ 0.02		1	3	11.15
(2)	0.–		0.02	0.01		–			3	10.9
(3)	–		0.02	0.01					3	10.5
Sulfidization					5.0			20		11.7
Bulk Rougher (1)			0.03	0.03				2	2	11.7
(2)			0.02	–				1	1	8.7
(3)		0.5	–	0.01				3-1	2	8.5

Stage	Roughers	Cleaners		
Equipment	D-1 4.5 l	D-1 1.2 l		
Speed, rpm	1900	1000		
% Solids	25			

Observations

Page 2

TAble A-34

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 3 Project No. _____ Operator _____ Date _____

Purpose _____

Procedure _____

Feed _____

Grind _____

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	PF-3	PEO	AX343				Grind	Cond	Froth	
Bulk Rougher (4)	0.05							1	2	8.5
(5)		0.05	0.05					2+1		8.7
Magnetic conc										

Stage				
Equipment				
Speed				
% Solids				

Observations _____

Table A-35

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 4 Project No. .005 Operator R. Tak Date 5/1/88

Purpose Add sulfidization to selective flowsheet of Test 1. High grade ore.

Procedure

Feed HRT 39467 High grade. 2000g of minus 10-mesh ore.

Grind 60 min. at 65% solids in the Denver 1/6 steel ball mill.

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	[illegible]	R240	AX705	MIBC/DF250	Grind	Cond	Froth	
Grind	3.5	1.0	0.33				60			8.6
Bulk Rougher (1)	0.5			0.030	0.01	.020/.012		2	3	9.0
(2)				0.015	—	—		1	3	
	CaO	$CuSO_4$	[illegible]	AX343	Na_2S	NaHS				
Zn Conditioner	3.4	1.5						5		11.0
Zn Rougher (1)			0.02	0.02				1	3	11.0/10.3
(2)	0.7		0.02	0.02				1	—	11.0
(3)			0.02	0.02				1	3	10.8
Sulfidization 1					5.0			5		9.0
Bulk Rougher (1)				0.10					2	8.8
Sulfidization 2				0.3		5.0		5		9.5
(2)				0.05				1	4	9.9

Stage	Rougher	Cleaner		
Equipment	D-1 4.5-1	D-1 1.2-1		
Speed, rpm	1900	1500		
% Solids	34			

Observations

Table A-36

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 5 Project No. 6905 Operator R. Pak Date 2/4/88

Purpose Attempt improved recoveries of lead & zinc by use of a chelating agent in a separate bulk rougher float stage.

Procedure

Feed HRI 39467 High grade ore. 2000g of minus 10-mesh.

Grind 60 min. at 65% solids in the Denver 1.5 L steel ball mill.

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC / DF250	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				60			9.20
Pb Rougher (1)				0.030	0.01	0.09 / 0.02		2	3	9.18
(2)				0.015				1	3	
	Cu(NO3)2	CuSO4	[illegible]	AX343	8-HQ					
Zn Conditioner	0.01	1.5						5		11.0
Zn Rougher (1)			[illegible]	[illegible]		[illegible]		1	3	11.06 / [illegible]
(2)	[illegible]		0.02	[illegible]		0.20		1		11.3/11.[illegible]
(3)	0.34		[illegible]	[illegible]		0.10		1	[illegible]	[illegible]
Pb Cleaner Conditioner			PES		0.1			5		7.0
Pb Cleaner 1								—	3	
2			0.03	0.01					[illegible]	
3			0.03	0.01						7.5

Stage	Rougners	Cleaners		
Equipment	D-1 4.5 l	D-1 1.2 l		
Speed, rpm	1900	1900		
% Solids	[illegible]			

Observations Zn rougher froth stability was poor. Higher frother dosage doesn't help. Will be considered on [illegible] initially, with no frother [illegible]

Table A-37

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 6 Project No. 6905 Operator R Park Date 2/4/88

Purpose Attempt improved recoveries of lead & zinc by use of a chelating agent in a separate bulk rougher float stage

Procedure

Feed

Grind 60 min. [illegible]

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	[illegible]	[illegible]	[illegible]	[illegible]	Grind	Cond	Froth	
Grind	4.0	[illegible]	[illegible]				60			9.3
[illegible]				0.030	0.01	[illegible]		[illegible]	[illegible]	[illegible]
[illegible]				0.015		[illegible]		1	5	9.13
	(010[illegible]	[illegible]	[illegible]	AR[illegible]	S-[illegible]U					
[illegible] Conditioner	2.[illegible]	[illegible]						5		11.[illegible]
[illegible] Rougher [illegible]	[illegible]		0.04	[illegible]		[illegible]		[illegible]	[illegible]	2[illegible].7
	[illegible]		[illegible]	[illegible]				[illegible]	[illegible]	[illegible]
[illegible]	0.26		[illegible]	[illegible]					[illegible]	[illegible]
Bulk Re Conditioner					0.1			5		7.25
Bulk Rougher (1)			DEC					-	3	7.2[illegible]
(2)			0.02	0.01	DMC			1	2	
(3)			0.02	0.01	0.1			5	3	7.0

Stage				
Equipment				
Speed				
% Solids				

Observations Re Rougher: froths were not improved with [illegible] initial dosages of MIBC.

Table A-38

Page 2

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 5 Project No. 2795 Operator _____ Date _____

Purpose _____

Procedure _____

Feed _____

Grind _____

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
			PAX				Grind	Cond	Froth	
(4)			.03					1	1	7.2

Stage				
Equipment				
Speed				
% Solids				

Observations _____

PAGE

Table A-39

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 7 Project No. 6905 Operator LP Date 8/5/88

Purpose

Procedure

Feed HRI [illegible] 10 mesh

Grind 60 min [illegible] 65% [illegible] 200 mesh [illegible] 2

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	Grind	Cond	Froth	
GRIND	4.0	1.5	0.37				60			9.24
Pb Rougher				0.02	0.01	0.02		2	2	9.23
2				0.015				1	2	9.23
	CaO	CuSO4	[illegible]	AX [illegible]						
Zn Conditioner	2.32	1.5						5		9.24 → 11.24
Zn Rougher 1			0.02	0.02		SHO 0.02		1	2	10.63
2	0.5		0.02	0.02		0.02		1	3	[illegible]
3	0.15		0.02	0.02		0.02		1	2	10.45 → 10.79

Stage				
Equipment				
Speed				
% Solids				

Observations

PAGE 2

Table A-40

HAZEN RESEARCH, INC.

(Cont) Flotation Test Report

Test No. 7 Project No. 3905 Operator LP Date 9/5/83

Purpose

Procedure

Feed

Grind

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH	
	[illegible]	6410	[illegible]	[illegible]	[illegible]		Grind	Cond	Froth		
Rock Sorting	[illegible]							7	—	10.58	
[illegible] 1)		[illegible]						10	5	10.44	1
			0.05					10	—	10.36	
[illegible] 2)				0.5				2	5	9.95	2
					3.0			10	—	9.92 / 11.52	
3)				[illegible]				2	5	11.56	3

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-41

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 3 Project No. 6905 Operator LP Date 8/10/88

Purpose

Procedure

Feed 2000g [illegible] mesh [illegible] 68 [illegible] grams

Grind [illegible]

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	[illegible]	[illegible]	NaCN	[illegible]	AX225	[illegible]	Grind	Cond	Froth	
Grind	3.0	1.0	0.33				60			
[illegible]				0.02	0.01	[illegible]			3	9.8
				0.02					[illegible]	9.7
	Ca(OH)	[illegible]	[illegible]	[illegible]						
[illegible]	2.35	1.0						5		9.7→11.0
[illegible]	0.12		0.02	0.02		0.01 / 0.023		1	3	10.9→11.0
[illegible]	0		0.02	0.02				1	3	10.95
[illegible]	[illegible]		[illegible]					1	3	[illegible]

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-42

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. [illegible] Project No. [illegible] Operator [illegible] Date [illegible]

Purpose

Procedure

Feed

Grind

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	814[illegible] C	6410	PAC	FF-[illegible]	Na[illegible]	[illegible]	Grind	Cond	Froth	
Rough Cond	[illegible]							5		[illegible]
Rough [illegible]		[illegible]						[illegible]	[illegible]	[illegible]
[illegible]			[illegible]					[illegible]		[illegible]
[illegible]				[illegible]		[illegible]		[illegible]	[illegible]	[illegible]
[illegible]					[illegible]			[illegible]		[illegible]
[illegible]				[illegible]				[illegible]	[illegible]	[illegible]

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-43

Page 1

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 9 Project No. 5905 Operator _____ Date 2/6/88

Purpose Improve zinc recovery on low grade ore

Procedure
1. Finer primary grind (~100% - 200 mesh)
2. More intensive use of chelating agent during bulk rougher step.
3. More intensive sulfidizing during bulk rougher step.

Feed HRI 39468 Low grade ore 2000g, -10 mesh

Grind 85 minutes at 65% solids (~100% - 200 mesh)

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	PAX	AX325		Grind	Cond	Froth	
Grind	4.0	1.0	0.33				85			
Pb Rougher (1)				.03	.01			1	3	
(2)				.02				1	3	
	CaloHin	$CuSO_4$	Nitrogel	AX343						
Zn Conditioner	2.1	1.5						5		→ 11.0
Zn Rougher (1)	0.20		.02	.02				1	3	11.0
(2)	0.30	0.5	.02	.02				2	3	
(3)	0.42	0.25	.03	—				5	3	
	81469 Cyanex			PF-3						
Bulk Conditioner 1	.25	0.10						5		11.
Bulk Rougher 1	-	-	-	-					5	
Bulk Conditioner 2	.20	-	-	-				5		
Bulk Rougher 2	-	-	-	-					5	

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-44

HAZEN RESEARCH, INC.

Page 2

Flotation Test Report

Test No. 9 Project No. _____ Operator _____ Date _____

Purpose _____

Procedure _____

Feed _____

Grind _____

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2S		AX343	PF-3			Grind	Cond	Froth	
Bulk Condition 3	2.5							10		11.
			.02	.05				2	5	
Bulk Condition 4	1.5							5		
			.01	.05				2	5	

Stage				
Equipment				
Speed				
% Solids				

Observations _____

Table A-45

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 10 Project No. 1905 Operator [illegible] Date [illegible]

Purpose Fineness of primary grind variable.

Procedure

Feed HRI 33467 High grade ore, 2000g -10 mesh

Grind ~100% -200 mesh: 85 min @ 65% solids in the 5x12" Denver steel ball mill.

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	MBC	R242	AX325	MIBC / AF-65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				85			
Pb Roucher (1)	0.90			0.030	0.01	.020 / 0.008		2	3	9.25
" (2)				0.015				1	3	
	CaO/H_2O	$CuSO_4$	M-1661	AX343		MIBC / AF-65				
Zn Conditioner	2.0	1.5						5		11.0
Zn Roucher (1)			0.02	0.02		.020 / .021			3	10.75
" (2)	0.6		0.03	0.02		— / .021		1	3	10.00
" (3)			0.02	0.02				1	3	10.70
	814690	6410	R-301	Na_2S	PE3	AX303				
Bulk Conditioner	0.20							5		10.70
Bulk Roucher 1		0.4						10	5	
" 2			0.05	3.0	0.05			10+5	5	
" 3					0.05	0.02		1	5	11.3
Stage 4			0.005					1	2	
Equipment										
Speed										
% Solids										

Observations Zn roucher froth was excellent with use of AF-65 and MIBC combination.

Table A-46

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 11 Project No. 6905 Operator R. Rak Date 8/31/88

Purpose

Procedure

Feed 4RI 39467 High grade ore

Grind 40 min at 65% solids

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC / AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				40			9.07
Pb Rougher (1)	0.21			0.030	0.01	.025 / –		2	3	9.25
" (2)				0.015		.005 / –		1	3	
	Calcul	$CuSO_4$	AF-166	AX343		MIBC / AF-65				
Zn Condition	2.23	1.5						5		11.0
Zn Rougher (1)			0.02	0.02		.005 / .042		1	3	10.86
" (2)			0.03	0.02		– / .021			3	
" (3)			0.02	0.02		/ .021		1	–	
	8146	6410	[illegible]	Na_2S	PF-3	AX343				
Bulk Conditioner	0.20							5		10.36
Bulk Rougher 1	0.10	0.4						10	5	10.50
" 2			0.05	3.0	0.05			10+5	5	11.5
" 3					0.015	0.02		1	3	11.4

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-47

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 12 Project No. 6905 Operator R. Tolk Date 9/1/88

Purpose Fineness of primary grind variable.

Procedure

Feed HRI 39468 Low grade ore, 2000g, -100 mesh BWI product.

Grind ~100% -200 mesh: Took the -100m dry ground BWI #537 product, and ground 20 additional minutes at 65% solids in the

Conditions 5" x 12" Denver steel ball mill.

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC/AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				80			
Pb Rougher (1)				0.030	0.01			1	3	9.53
" (2)				0.015				1	3	
	$CaO+H_2$	$CuSO_4$	M-1661	AX343		MIBC/AF65				
Zn Condition	3.77	1.5						5		11.5
Zn Rougher (1)			0.02	0.02				1	3	
" (2)			0.03	0.02				1	3	
" (3)			0.02	—				1	3	
	814640	6410	FEO Rol	Na_2S	PAC-3	AX343				
Bulk Condition	0.20							5		
Bulk Rougher 1		0.4						10	5	11.11 ✓
" 2			0.05	3.0	0.05			10+5	5	11.6 ✓
" 3					0.015	0.02		1	3	✓
" 4			0.05					1		

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-48

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 13 Project No. 6905 Operator PPK Date 9/12/88

Purpose

Procedure

Feed HRT 39468 Low Grade Ore, 2000g

Grind 38 min. at 65% solids (~ 60% -200 mn)

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	M113C / AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				38			9.75
Pb Rougher (1)				0.030	0.01			1	3	9.55
" (2)				0.015				1	3	
	$CaCO_3$	$CuSO_4$	M-1661	AX343		M113C / AF65				
Zn Conditioner	2.00	1.5						5		11.5
Zn Rougher (1)			0.02	0.02		0.01		1	3	11.45
" (2)			0.02	0.02		0.01		1	3	
" (3)			0.01	-				1	3	11.23
	814SC	64.0	PAC 221	No 5	PF-3	AX343				
Bulk Conditioner	0.20							5		
Bulk Rougher 1		0.40						10	3	11.01
" 2			0.09	3.0	0.05			10+5	3	11.5
" 3					0.015	0.02		1	3	11.4

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-49

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 14 Project No. 6905 Operator LP Date 9/16/88

Purpose Study flotation response of classified (-20 mesh) low grade chat sample 07-01

Procedure Conditions of Test 10 (advanced sulfidization to 2nd Oxide Rougher, and used additional xanthate in 3rd oxide rougher.)

Feed 500 g of minus 20 mesh sample 07-01 HRI #37849-C

Grind 15 min / 500 g at 50% solids in the 5×12" steel ball mill. ~80% -200 mesh

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC / AF-65	Grind	Cond	Froth	
Grind	3.5	1.0	0.33				15			
Pb Rougher (1)				0.030	0.01	0.005 / 0.02		1	3	
" (2)				0.015				1	3	
	$Ca(OH)_2$	$CuSO_4$	M-1661	AX343						
Zn Condition	592	1.5						5		11.5 → 8.17
Zn Rougher (1)			0.02	0.02				1	3	11.5
" (2)			0.02	0.02					3	
" (3)			0.01					1	3	
	STARCH	3410	PRO 301	Na_2S	PF-3	AX343				
Oxide Condition	0.20							5		
Oxide Rougher 1		0.40						10	—	11.2
" 2			0.05	2.5	0.05			10 + 5	5	
" 3					0.015	0.02		1	5	11.4

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-50

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 15 Project No. 6905 Operator LP Date 9/16/88

Purpose Flotation of composite of 71.5% high grade ore and 28.5% of -28 mesh low grade chat

Procedure Same as Test 14.

Feed 1430g of 39467 High grade, 570g of -28m 39849-1 01-0

Grind 60 min at 65% solids

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC / AE65	Grind	Cond	Froth	
Grind	3.5	1.0	0.33				60			
Pb Roughter (1)				0.03	0.01	0.021		1	3	
(2)				0.015		0.010		1	3	
	$Ca(OH)_2$	$CuSO_4$	M-661	AX343						
Zn Conditioner	2.2	1.5						5		9.15→11.3
Zn Roughter (1)			0.02	0.02				1	3	11-13
(2)			0.02	0.02				1	3	
(3)			0.01					1	3	
	81469C	6410	PEO 301	No. 5	PF-2	AX242				
Oxide Conditioner	0.20							5		
Oxide Roughner 1		0.40						10	5	10.7→11
2			0.05	2.5	0.05			10 + 5	5	11.43
3					0.015	0.02		1	5	

0.36g

Stage				
Equipment				
Speed				
% Solids				

Observations

Table A-51

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 16 Project No. 6905 Operator LP Date 9/19/88

Purpose Optimization of conditions in the oxide rougher with emphasis on reducing conditioning times and simplifying the reagent balance.

Procedure See below. Deleted 81469 C chelating agent (and its corresponding 5 min. conditioning time). Test 10 conditions.

Feed (1000g of minus 10-mesh High Grade ore, HRT 39467

Grind 25 min at 50% solids in the laboratory steel ball mill. (75.3% -200 mesh)

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC/AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				25			9.
Pb Rougher (1)				0.030	0.01	0.028		1	3	
" (2)				0.015	–	0.014		1	3	
	$Ca(OH)_2$	$CuSO_4$	M-1661	AX343						
Zn Condition	3.46	1.5						5		11.5 9.2
Zn Rougher (1)			0.02	0.02		0.014		1	3	11.46
" (2)			0.02	0.02		0.014		1	3	
" (3)			0.01	–				1	3	
	6410	PED 301	Na.S	PF-3	AX343					
Oxide Condition	0.40							10		11.19
Oxide Rougher 1									5	
" 2		0.05	2.5	0.05				10+5	5	
" 3				0.015	0.02			1	5	

Stage	All			
Equipment	D-1 2.5ℓ			
Speed, rpm	1500 (steel imp.)			
% Solids	34			

Observations

Table A-52

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 17 Project No. 6905 Operator LD Date 9/19/88

Purpose Optimization of conditions in the oxide rougher with emphasis on reducing conditioning times and simplifying the reagent balance.

Procedure See below. Deleted 6410 (and its corresponding 10 min conditioning time). Test 10 conditions.

Feed (1000g of minus 10-mesh High Grade ore, HRI 39467

Grind 25 min at 50% solids in the laboratory steel ball mill. (75.3 % -200 mesh)

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC/AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				25			9.2
Pb Rougher (1)				0.030	0.01	0.028		1	3	
" (2)				0.015	–	0.014		1	3	
	$Ca(OH)_2$	$CuSO_4$	M-1661	AX343						
Zn Condition	3.64	1.5						5		11.5 9.13
Zn Rougher (1)			0.02	0.02				1	3	11.47
" (2)			0.02	0.02		0.014		1	3	
" (3)			0.01	–		0.014		1	3	
	814690	PED 301	Na S	PF-3	AX343					
Oxide Condition	0.20							5	–	11.21
Oxide Rougher 1									5	
" 2		0.09	2.5	0.05				10+5	5	
" 3				0.015	0.02			1	5	11.5

Stage	All			
Equipment	D-1 2.5 l			
Speed, rpm	1500 (steel imp.)			
% Solids	34			

Observations

Table A-53

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 18 Project No. 6905 Operator LP Date 9/19/88

Purpose Optimization of conditions in the oxide rougher with emphasis on reducing conditioning times and simplifying the reagent balance.

Procedure See below. In Ox Rougher stage 2, sulfidized first, then added PEO (reversed from Test 10).

Feed (1000g of minus 10-mesh High Grade ore, HRI 39467

Grind 25 min at 50% solids in the laboratory steel ball mill. (75.3% -200 mesh)

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC/AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				25			9.4
Pb Rougher (1)				0.030	0.01			1	3	
" (2)				0.015	–			1	3	
	$Ca(OH)_2$	$CuSO_4$	M-1661	AX343						
Zn Condition	3.30	1.5						5		11.5 9.5
Zn Rougher (1)			0.02	0.02				1	3	11.49
" (2)			0.02	0.02				1	3	
" (3)			0.01	–				1	3	
	8146X	6410	Na_2S	PEO 301	PF-3	AX343				
Oxide Condition	0.30							5		11.17
Oxide Rougher 1		0.40						10	5	
" 2			2.5	0.05	0.05			5 + 10	5	
" 3					0.015	0.02		1	5	11.27

Stage	All			
Equipment	D-1 2.5 l			
Speed, rpm	1500 (steel imp.)			
% Solids	34			

Observations

Table A-54

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 19 Project No. 6905 Operator [illegible] Date 9/19/88

Purpose Optimization of conditions in the oxide rougher with emphasis on reducing conditioning times and simplifying the reagent balance.

Procedure See below. Sulfidized at start of Oxide stages, then repeated Test 10 oxide sequence.

Feed (1000g of minus 10-mesh High Grade ore, HRI 39467

Grind 25 min at 50% solids in the laboratory steel ball mill. (75.3% -200 mesh)

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC/AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				25			9.14
Pb Rougher (1)				0.030	0.01	0.028		1	3	
" (2)				0.015	–	0.014		1	3	
	$Ca(OH)_2$	$CuSO_4$	M-1661	AX343						
Zn Condition	4.34	1.5						5		11.5 9.06
Zn Rougher (1)			0.02	0.02				1	3	11.47
" (2)			0.02	0.02		0.014		1	3	
" (3)			0.01	–				1	3	
	Na_2S	814694	6410	PEO 351	PF3	AX343				
Oxide Condition	2.5	0.20						5+5		11.5
Oxide Rougher 1			0.40					10	5	9.9
" 2				0.05	0.05			10	5	
" 3					0.05	0.02		1	5	

Stage	All			
Equipment	D-1 2.5l			
Speed, rpm	1500 (steel imp.)			
% Solids	34			

Observations

Table A-55

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 20 Project No. 6905 Operator LP Date 9/2/88

Purpose Optimization of conditions in the oxide rougher with emphasis on reducing conditioning times and simplifying the reagent balance.

Procedure See below. At start of oxide circuit, sulfidized together with 81469C chelating agent

Feed (1000g of minus 10-mesh High Grade ore, HRI 39467

Grind 25 min at 50% solids in the laboratory steel ball mill. (75.3% -200 mesh)

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC/AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				25			9.29
Pb Rougher (1)				0.030	0.01	0.028		1	3	
" (2)				0.015	–	0.014		1	3	
	$Ca(OH)_2$	$CuSO_4$	M-1661	AX343						
Zn Condition	3.68	1.5						5		11.5 9.1
Zn Rougher (1)			0.02	0.02				1	3	11.45
" (2)			0.02	0.02				1	3	
" (3)			0.01	–				1	3	
	Na_2S	81469C	6410	PEO301	PF-3	AX343				
Oxide Condition	(2.5)	0.20						5		11.51
Oxide Rougher 1			0.40					10	5	
" 2				0.05	0.05			10	5	
" 3					0.015	0.02		1	5	11.12

Stage	All			
Equipment	D-1 2.5l			
Speed, rpm	1500 (steel imp.)			
% Solids	34			

Observations

Table A-56

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 21 Project No. 6905 Operator Rak Date 9/21/88

Purpose Optimization of conditions in the oxide rougher with emphasis on reducing conditioning times and simplifying the reagent balance.

Procedure See below. In Oxide Rougher stage 1, sulfidized together with 6410 collector. Test 10 conditions.

Feed (1000g of minus 10-mesh High Grade ore, HRI 39467

Grind 25 min at 50% solids in the laboratory steel ball mill. (75.3% -200 mesh)

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R-242	AX325	MIBC/AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				25			9.24
Pb Rougher (1)				0.030	0.01	.010/.007		1	3	
" (2)				0.015	–	–/.007		1	3	
	$Ca(OH)_2$	$CuSO_4$	M-1661	AX343						
Zn Condition	2.04	1.5						5		11.5
Zn Rougher (1)			0.02	0.02		–/.007		1	3	
" (2)			0.02	0.02		.05/–		1	3	
" (3)			0.01	–				1	3	
	81462C	Na_2S	6410	PEO 301	PF-3	AX343				
Oxide Condition	0.20							5		11.0
Oxide Rougher 1		2.5	0.40					5	5	11.7
" 2				0.05	0.05	0.02*		10	5	
" 3					0.015			1	5	

Stage	All			
Equipment	D-1 2.5l			
Speed, rpm	1500 (steel imp.)			
% Solids	34			

Observations Some zinc began floating in last half of 2nd Pb rougher.

*Accidentally added 0.05 M-1661 in addition to AX343.

Table A-57

HAZEN RESEARCH, INC.

Flotation Test Report

Test No. 22 Project No. 6905 Operator LP Date 9/22/88

Purpose Optimization of conditions in the oxide rougher with emphasis on reducing conditioning times and simplifying the reagent balance.

Procedure See below. In Oxide Rougher stage 2, sulfidized together with the PEO. Test 10 conditions.

Feed (1000g of minus 10-mesh High Grade ore, HRI 39467

Grind 25 min at 50% solids in the laboratory steel ball mill. (75.3% -200 mesh)

Conditions

Stage	Reagents Added, Pounds/Ton						Time, Minutes			Pulp pH
	Na_2CO_3	$ZnSO_4$	NaCN	R242	AX325	MIBC/AF65	Grind	Cond	Froth	
Grind	4.0	1.0	0.33				25			9.20
Pb Rougher (1)	0.5			0.030	0.01	0.028		5+1	3	
" (2)				0.015	–	0.014		1	3	
	$Ca(OH)_2$	$CuSO_4$	M-1661	AX343						
Zn Condition	9.46	1.5						5		11.5 9.20
Zn Rougher (1)			0.02	0.02				1	3	11.40
" (2)			0.02	0.02		0.014		1	3	
" (3)			0.01	–		0.014		1	3	
	8146AC	6410	PEO 301	Na_2S	PF-3	AX242				
Oxide Condition	0.20							5		11.02
Oxide Rougher 1		0.40						10	5	
" 2			0.05	2.5	0.05			5	5	
" 3					0.015	0.02		1	5	

Stage	All			
Equipment	D-1 2.5 l			
Speed, rpm	1500 (steel imp.)			
% Solids	34			

Observations

Table A-58

SAMPLE: Oxide Rougher Tails, Test 10

PULP: As floated, pH 10.72

CONDITIONS: pH = 11.5-11.6 with lime

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 10a
DATE: Sept 7 '8
BY: SM

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	990	0	730	0
0.5	950	40	690	0.046
1	920	70	660	0.080
1.5	890	100	630	0.115
2	870	120	610	0.137
3	810	180	550	0.206
4	750	240	490	0.275
5	700	290	440	0.332
7	590	400	330	0.458
10	450	540	190	0.619
15	410	580	150	0.664
20	340	650	80	0.745
25	290	700	30	0.802
30	270	720	10	0.825
			730	0.000
			730	0.000
			730	0.000
			730	0.000
			730	0.000
			730	0.000
			730	0.000
			730	0.000
			730	0.000
			730	0.000
			730	0.000
			730	0.000
TERM.			730	0.000
LEVEL	260	730	0	0.836

REQUIRED MEASUREMENTS

TUBE, ML/FT 873

PULP WEIGHT, g 1175

DRY SOLIDS WT, g 299.8

SOLIDS S.G. 2.9

COMPRESSION ZONE SETTLING RATE, FT/HR: 1.763
FEED ZONE SETTLING RATE, FT/HR: 4.200
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.230

CALCULATED RESULTS

INITIAL % SOLIDS: 25.51

FINAL % SOLIDS: 67.37

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 30.77

F = 2.250

D = 0.484

R = 1.76 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 1.33
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 0.77

Table A-59

SAMPLE: Oxide Rougher Tails, Test 10

PULP: Repulped solids from thickening Test 10a

CONDITIONS: Ph > 11.75, 0.007g Percol 351 flocculant

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 10b
DATE: Sept 7 '8
BY: SM

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	980	0	685	0
0.5	670	310	375	0.355
1	545	435	250	0.498
1.5	500	480	205	0.550
2	475	505	180	0.578
2.5	450	530	155	0.607
3	430	550	135	0.630
3.5	420	560	125	0.641
4	410	570	115	0.653
4.5	400	580	105	0.664
5	395	585	100	0.670
6	385	595	90	0.682
7	375	605	80	0.693
8	365	615	70	0.704
9	360	620	65	0.710
10	355	625	60	0.716
12	345	635	50	0.727
15	335	645	40	0.739
20	330	650	35	0.745
25	325	655	30	0.750
30	320	660	25	0.756
			685	0.000
			685	0.000
			685	0.000
			685	0.000
			685	0.000
TERM.			685	0.000
LEVEL	295	685	0	0.785

REQUIRED MEASUREMENTS

TUBE, ML/FT 873

PULP WEIGHT, g 1231

DRY SOLIDS WT, g 299.8

SOLIDS S.G. 2.9

COMPRESSION ZONE SETTLING R 6.290
FEED ZONE SETTLING RATE, FT/HR: 30.000
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.580

CALCULATED RESULTS

INITIAL % SOLIDS: 24.35

FINAL % SOLIDS: 54.91

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 41.37

F = 1.417

D = 0.821

R = 6.29 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 0.13
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 0.10

Table A-60

SAMPLE: Oxide Rougher Tails, Test 11

PULP: As floated, pH 10.88

CONDITIONS: pH = 11.5-11.6 with lime

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 11a
DATE: Sept 7 '8
BY: SM

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	985	0	655	0
0.5	970	15	640	0.017
1	930	55	600	0.064
1.5	900	85	570	0.098
2	870	115	540	0.133
3	815	170	485	0.197
4	770	215	440	0.249
5	720	265	390	0.306
7	610	375	280	0.434
10	500	485	170	0.561
15	405	580	75	0.671
20	370	615	40	0.711
25	360	625	30	0.723
30	350	635	20	0.734
			655	0.000
			655	0.000
			655	0.000
			655	0.000
			655	0.000
			655	0.000
			655	0.000
			655	0.000
			655	0.000
			655	0.000
			655	0.000
			655	0.000
TERM.			655	0.000
LEVEL	330	655	0	0.757

REQUIRED MEASUREMENTS

TUBE, ML/FT 865

PULP WEIGHT, g 1256.5

DRY SOLIDS WT, g 326.54

SOLIDS S.G. 2.9

COMPRESSION ZONE SETTLING RATE, FT/HR: 1.275
FEED ZONE SETTLING RATE, FT/HR: 3.600
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.305

CALCULATED RESULTS

INITIAL % SOLIDS: 25.99

FINAL % SOLIDS: 54.29

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 32.89

F = 2.040

D = 0.842

R = 1.28 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 1.25
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 0.74

Table A-61

SAMPLE: Oxide Rougher Tails, Test 11

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 11b
DATE: Sept 7 '8
BY: SM

PULP: Repulped solids from thickening Test 11a

CONDITIONS: pH > 11.75 with lime, 0.007g Percol 351 flocculant

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	980	0	645	0
0.5	650	330	315	0.382
1	540	440	205	0.509
1.5	480	500	145	0.578
2	450	530	115	0.613
2.5	440	540	105	0.624
3	425	555	90	0.642
3.5	410	570	75	0.659
4	400	580	65	0.671
4.5	390	590	55	0.682
5	385	595	50	0.688
6	275	705	-60	0.815
7	370	610	35	0.705
8	360	620	25	0.717
9	355	625	20	0.723
10	350	630	15	0.728
12	350	630	15	0.728
15	345	635	10	0.734
20	340	640	5	0.740
25	340	640	5	0.740
30	335	645	0	0.746
			645	0.000
			645	0.000
			645	0.000
			645	0.000
			645	0.000
TERM.			645	0.000
LEVEL	335	645	0	0.746

REQUIRED MEASUREMENTS

TUBE, ML/FT 865

PULP WEIGHT, g 1249

DRY SOLIDS WT, g 326.54

SOLIDS S.G. 2.9

COMPRESSION ZONE SETTLING R 1.350
FEED ZONE SETTLING RATE, FT/HR: 27.000
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.585

CALCULATED RESULTS

INITIAL % SOLIDS: 26.14

FINAL % SOLIDS: 54.06

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 43.95

F = 1.275

D = 0.850

R = 1.35 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 0.42
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 0.10

Table A-62

SAMPLE: Oxide Rougher Tails, Test 12

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 12a
DATE: Sept 7 '8
BY: SM

PULP: As floated, pH 10.50

CONDITIONS: pH = 10.50, 0.074 lb/t Percol 351 flocculant

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	920	0	600	0
0.5	800	120	480	0.139
1	740	180	420	0.208
1.5	680	240	360	0.277
2	610	310	290	0.358
2.5	530	390	210	0.451
3	460	460	140	0.532
3.5	440	480	120	0.555
4	430	490	110	0.566
4.5	420	500	100	0.578
5	410	510	90	0.590
6	395	525	75	0.607
7	385	535	65	0.618
8	370	550	50	0.636
9	365	555	45	0.642
10	360	560	40	0.647
12	350	570	30	0.659
15	340	580	20	0.671
20	330	590	10	0.682
25	325	595	5	0.688
30	320	600	0	0.694
			600	0.000
			600	0.000
			600	0.000
			600	0.000
			600	0.000
TERM.			600	0.000
LEVEL	320	600	0	0.694

REQUIRED MEASUREMENTS

TUBE, ML/FT 865

PULP WEIGHT, g 1263

DRY SOLIDS WT, g 340.02

SOLIDS S.G. 2.8

COMPRESSION ZONE SETTLING RATE, FT/HR: 10.800
FEED ZONE SETTLING RATE, FT/HR: 4.100
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.510

CALCULATED RESULTS

INITIAL % SOLIDS: 26.92

FINAL % SOLIDS: 51.29

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 41.37

F = 1.417

D = 0.950

R = 10.80 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 0.06
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 0.57

Table A-63

SAMPLE: Oxide Rougher Tails, Test 12

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 12b
DATE: Sept 7 '8
BY: SM

PULP: Repulped solids from thickening Test 12a

CONDITIONS: pH = 10.5, 0.082 lb/t Percol 351 flocculant

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	940	0	570	0
0.5	700	240	330	0.277
1	590	350	220	0.405
1.5	550	390	180	0.451
2	520	420	150	0.486
2.5	510	430	140	0.497
3	490	450	120	0.520
3.5	480	460	110	0.532
4	470	470	100	0.543
4.5	460	480	90	0.555
5	455	485	85	0.561
6	445	495	75	0.572
7	435	505	65	0.584
8	430	510	60	0.590
9	425	515	55	0.595
10	420	520	50	0.601
12	410	530	40	0.613
15	400	540	30	0.624
20	390	550	20	0.636
25	385	555	15	0.642
30	385	555	15	0.642
			570	0.000
			570	0.000
			570	0.000
			570	0.000
			570	0.000
TERM.			570	0.000
LEVEL	370	570	0	0.659

REQUIRED MEASUREMENTS

TUBE, ML/FT 865

PULP WEIGHT, g 1263

DRY SOLIDS WT, g 340.02

SOLIDS S.G. 2.8

COMPRESSION ZONE SETTLING R 5.910
FEED ZONE SETTLING RATE, FT/HR: 12.000
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.485

CALCULATED RESULTS

INITIAL % SOLIDS: 26.92

FINAL % SOLIDS: 49.06

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 40.31

F = 1.481

D = 1.038

R = 5.91 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 0.10
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 0.19

Table A-64

SAMPLE: Oxide Rougher Tails, Test 12

PULP: As floated, pH 10.50

CONDITIONS: pH = 11.5-11.6 with lime

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 12c
DATE: Sept 7 '8
BY: SM

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	990	0	620	0
0.5	985	5	615	0.006
1	980	10	610	0.012
1.5	970	20	600	0.023
2	950	40	580	0.046
3	930	60	560	0.069
4	910	80	540	0.092
5	900	90	530	0.104
7	870	120	500	0.139
10	830	160	460	0.185
15	760	230	390	0.266
20	690	300	320	0.347
25	620	370	250	0.428
30	570	420	200	0.486
			620	0.000
			620	0.000
			620	0.000
			620	0.000
			620	0.000
			620	0.000
			620	0.000
			620	0.000
			620	0.000
			620	0.000
			620	0.000
			620	0.000
TERM.			620	0.000
LEVEL	370	620	0	0.717

REQUIRED MEASUREMENTS

TUBE, ML/FT 865

PULP WEIGHT, g 1294

DRY SOLIDS WT, g 371.33

SOLIDS S.G. 2.8

COMPRESSION ZONE SETTLING RATE, FT/HR: 0.175
FEED ZONE SETTLING RATE, FT/HR: 1.200
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.233

CALCULATED RESULTS

INITIAL % SOLIDS: 28.70

FINAL % SOLIDS: 55.09

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 33.99

F = 1.942

D = 0.815

R = 0.18 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 8.58
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 1.85

Table A-65

SAMPLE: Oxide Rougher Tails, Test 12

PULP: Repulped solids from thickening Test 12c

CONDITIONS: pH > 11.75 with lime

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 12d
DATE: Sept 7 '8
BY: SM

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	980	0	600	0
0.5	970	10	590	0.012
1	960	20	580	0.023
1.5	950	30	570	0.035
2	945	35	565	0.040
2.5	940	40	560	0.046
3	930	50	550	0.058
3.5	925	55	545	0.064
4	920	60	540	0.069
4.5	915	65	535	0.075
5	910	70	530	0.081
6	885	95	505	0.110
7	870	110	490	0.127
8	860	120	480	0.139
9	850	130	470	0.150
10	840	140	460	0.162
12	805	175	425	0.202
15	750	230	370	0.266
20	680	300	300	0.347
25	610	370	230	0.428
30	550	430	170	0.497
			600	0.000
			600	0.000
			600	0.000
			600	0.000
			600	0.000
TERM.			600	0.000
LEVEL	380	600	0	0.694

REQUIRED MEASUREMENTS

TUBE, ML/FT 865

PULP WEIGHT, g 1278

DRY SOLIDS WT, g 371.33

SOLIDS S.G. 2.8

COMPRESSION ZONE SETTLING R 1.060
FEED ZONE SETTLING RATE, FT/HR: 1.080
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.050

CALCULATED RESULTS

INITIAL % SOLIDS: 29.06

FINAL % SOLIDS: 54.77

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 30.07

F = 2.325

D = 0.826

R = 1.06 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 1.89
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 1.99

Table A-66

SAMPLE: Oxide Rougher Tails, Test 13

PULP: As floated, pH 11.15

CONDITIONS: pH = 11.5-11.6 with lime

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 13a
DATE: Sept 7 '8
BY: SM

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	995	0	595	0
0.5	985	10	585	0.011
1	980	15	580	0.017
1.5	970	25	570	0.029
2	960	35	560	0.040
3	940	55	540	0.063
4	930	65	530	0.074
5	910	85	510	0.097
7	880	115	480	0.132
10	835	160	435	0.183
15	750	245	350	0.281
20	680	315	280	0.361
25	610	385	210	0.441
30	550	445	150	0.510
			595	0.000
			595	0.000
			595	0.000
			595	0.000
			595	0.000
			595	0.000
			595	0.000
			595	0.000
			595	0.000
			595	0.000
			595	0.000
			595	0.000
			595	0.000
TERM.			595	0.000
LEVEL	400	595	0	0.682

REQUIRED MEASUREMENTS

TUBE, ML/FT 873

PULP WEIGHT, g 1302

DRY SOLIDS WT, g 407.26

SOLIDS S.G. 2.8

COMPRESSION ZONE SETTLING RATE, FT/HR: 0.938
FEED ZONE SETTLING RATE, FT/HR: 1.140
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.050

CALCULATED RESULTS

INITIAL % SOLIDS: 31.28

FINAL % SOLIDS: 57.60

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 32.36

F = 2.090

D = 0.736

R = 0.94 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 1.92
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 1.71

Table A-67

SAMPLE: Oxide Rougher Tails, Test 13

THICKENING TEST REPORT

PROJECT: 6905
TEST NO: 13b
DATE: Sept 7 '8
BY: SM

PULP: Repulped solids from thickening Test 13a

CONDITIONS: pH -> 11.75 with lime, 0.103 lb Percol 351 flocculant

TIME MIN	INTERFACE	SETTLED ML	H-E ML	SETTLED FEET
0	1020	0	605	0
0.5	860	160	445	0.183
1	760	260	345	0.298
1.5	660	360	245	0.412
2	600	420	185	0.481
2.5	570	450	155	0.515
3	550	470	135	0.538
3.5	540	480	125	0.550
4	530	490	115	0.561
4.5	520	500	105	0.573
5	510	510	95	0.584
6	490	530	75	0.607
7	480	540	65	0.619
8	470	550	55	0.630
9	460	560	45	0.641
10	455	565	40	0.647
12	445	575	30	0.659
15	440	580	25	0.664
20	430	590	15	0.676
25	425	595	10	0.682
30	420	600	5	0.687
		1020	-415	1.168
		1020	-415	1.168
		1020	-415	1.168
		1020	-415	1.168
		1020	-415	1.168
TERM.		1020	-415	1.168
LEVEL	415	605	0	0.693

REQUIRED MEASUREMENTS

TUBE, ML/FT 873

PULP WEIGHT, g 1303

DRY SOLIDS WT, g 407.76

SOLIDS S.G. 2.8

COMPRESSION ZONE SETTLING RATE, FT/HR: 0.250
FEED ZONE SETTLING RATE, FT/HR: 12.360
POINT OF ENTRY TO COMPR. ZONE, SETTLED FEET: 0.525

CALCULATED RESULTS

INITIAL % SOLIDS: 31.29

FINAL % SOLIDS: 58.42

% SOLIDS AT ENTRY TO COMPRESSION ZONE = 48.27

F = 1.072

D = 0.712

R = 0.25 FT/HR

COMPRESSION ZONE UNIT AREA, SQ FT/TON/DAY = 1.92
FEED ZONE UNIT AREA, SQ FT/TON/DAY = 0.16

Appendix B
Chat and Sediment Data

APPENDIX B
CHAT AND SEDIMENT DATA

Lead (galena, PbS) was mined at the Galena, Kansas subsite during the middle to late 1800s into the early 1900s. Most ore was hand cobbed to increase the lead grade before being fed to the milling operations. Milling and concentration processes then were not as efficient as present operations, but with the high density of the galena mineral and by using gravity concentration methods, satisfactory recovery was achieved. Zinc was not considered a valuable mineral at that time and the majority of the mineral was removed during the hand cobbing process and was left as surface mine wastes.

Chat is a waste product from past mining and ore milling operations. The chat resembles a tailing material from present milling operations, but it is much coarser with a top size of about 1/4 inch ranging down to very fine minus 400 mesh (38 micron) material. Typical size analyses of two chat samples from the Galena subsite are in Table B-1.

Chat tonnage estimates completed by the Galena PRPs are approximately 1 million tons. During past field sampling events, visual estimates were made of several large chat piles by CH2M HILL staff. The chat piles were estimated to have ranged about 70,000 to 100,000 tons each. Therefore, it is felt that 1 million tons of chat at the Galena site is a reasonable estimate. This appendix presents the results of field sampling and analyses of chat and sediment samples collected by CH2M HILL during three separate events in 1988.

Table B-1
SCREEN ASSAY OF CHAT SAMPLE NO. 01-02-01

Product Size Mesh	Weight		Cumulative		Assay, mg/kg			Distribution, Percent					
	Gram	Percent	Percent	Percent	Pb	Zn	Cd	Pb	Cumul	Zn	Cumul	Cd	Cumul
+3	5.78	1.63	1.63	100.0	110	590	0	0.4	100	1.9	100	0	100
-3+4	63.01	17.75	19.37	98.37	100	460	3	3.9	99.6	16.3	98.1	25.5	100
-4+6	67.88	19.12	38.49	80.63	100	450	2	4.2	95.7	17.1	81.8	18.3	74.5
-6+8	10.47	2.95	41.44	61.51	100	240	0	0.6	91.5	1.4	64.7	0	56.2
-8+10	94.54	26.63	68.07	58.56	200	190	0	11.7	90.9	10.1	63.3	0	56.2
-10+14	17.26	4.86	72.93	31.93	150	230	0	1.6	79.2	2.2	53.2	0	56.2
-14+20	24.62	6.93	79.86	27.07	190	320	2	2.9	77.6	4.4	51	6.6	56.2
-20+28	3.59	1.01	80.87	20.14	200	730	3	0.4	74.7	1.5	46.6	1.5	49.6
-28+35	21.84	6.15	87.02	19.13	170	820	3	2.3	74.3	10.1	45.1	8.8	48.1
-35+48	6.07	1.71	88.73	12.98	240	1,100	7	0.9	72	3.7	35	5.7	39.3
-48+65	6.47	1.82	90.56	11.27	230	1,100	7	0.9	71.1	4	31.3	6.1	33.6
-65+100	4.23	1.19	91.75	9.44	250	1,300	8	0.7	70.2	3.1	27.3	4.6	27.5
-100+150	2.6	0.73	92.48	8.25	460	1,400	9	0.7	69.5	2	24.2	3.2	22.9
-150+200	1.24	0.35	92.83	7.52	300	1,400	9	0.2	68.8	1	22.2	1.5	19.7
-200+270	1.51	0.43	93.25	7.17	320	1,300	8	0.3	68.6	1.1	21.2	1.6	18.2
-270+400	0.61	0.17	93.43	6.75	360	1,200	9	0.1	68.3	0.4	20.1	0.7	16.6
-400	23.34	6.57	100.00	6.57	4,700	1,500	5	68.2	68.2	19.7	19.7	15.9	15.9
			355.06	100.00				100		100		100	
Calc Head					454.4	501.8	2.1						

Table B-1
SCREEN ASSAY OF CHAT SAMPLE NO. 07-01-01
(continued)

Product Size Mesh	Weight Gram	Weight Percent	Cumulative Percent	Cumulative Percent	Assay, mg/kg Pb	Assay, mg/kg Zn	Assay, mg/kg Cd	Distribution, Percent Pb	Distribution, Percent Cumul	Distribution, Percent Zn	Distribution, Percent Cumul	Distribution, Percent Cd	Distribution, Percent Cumul
+3	0	0.00	00.00	100.00	0	0	0	0	0	0	0	0	0
-3+4	9.76	2.47	2.47	100.00	210	930	3	1.2	100	1.4	100	1.3	100
-4+6	164.13	41.53	44.00	97.53	240	1,200	4	23.6	98.8	30.5	98.6	29.4	98.7
-6+8	131.61	33.30	77.31	56.00	250	780	3	19.7	75.2	15.9	68.1	17.7	69.3
-8+10	40.51	10.25	87.56	22.69	230	460	2	5.6	55.5	2.9	52.2	3.6	51.6
-10+14	10.27	2.60	90.16	12.44	240	780	5	1.5	49.9	1.[illegible]	49.3	2.3	48
-14+20	8.88	2.25	92.40	9.84	280	2,000	12	1.5	48.4	2.[illegible]	48.1	4.8	45.7
-20+28	2.64	0.67	93.07	7.60	330	7,400	35	0.5	46.9	3	45.3	4.1	40.9
-28+35	3.63	0.92	93.99	6.93	350	14,000	48	0.8	46.4	7.9	42.3	7.8	36.8
-35+48	1.5	0.38	94.37	6.01	350	20,000	65	0.3	45.6	4.6	34.4	4.4	29
-48+65	1.31	0.33	94.70	5.63	400	22,000	70	0.3	45.3	4.5	29.8	4.1	24.6
-65+100	0.72	0.18	94.88	5.30	400	21,000	60	0.2	45	2.3	25.3	1.9	20.5
-100+150	0.3	0.08	94.96	5.12	600	34,000	70	0.1	44.8	1.6	23	0.9	17.7
-150+200	0.14	0.04	94.99	5.04	700	46,000	20	0.1	44.7	1	21.4	0.1	17.7
-200+270	0.18	0.05	95.04	5.01	0	100,000	0	0	44.6	2.8	20.4	0	17.6
-270+400	0.05	0.01	95.05	4.96	0	0	0	0	44.6	0	17.6	0	17.6
-400	19.55	4.95	100.00	4.95	3,800	5,800	20	44.6	44.6	17.6	17.6	17.6	17.6
			395.18	100.00				100		100		100	
Calc Head					421.7	1,633.2	5.7						

SAMPLING

Following completion of a second OUFS, EPA identified a preferred remedy. The preferred remedy would remove and process surface mine wastes to primarily remove lead, zinc, and cadmium, and produce a tailing of sufficient quality to permit disposal in the mine working voids. The removal and processing of the surface mine wastes, as identified in the preferred remedy, would use mining, trucking, crushing, grinding, and mineral flotation to remove metals and produce an acceptable tailing. To provide technical assistance to EPA for the preliminary design of the milling circuit, EPA authorized the development of the Mine Waste Characterization Program. This program would provide additional information on critical design parameters, such as fineness of grind, retention times, number of flotation stages, flotation reagents and amounts of additions, and expected recoveries and tailing quality.

In June of 1988, field sampling provided test material for the Mine Waste Characterization Program (see Appendix A). Surface mine waste materials, with both high and low lead and zinc concentrations, were collected for metallurgical test work. The field sampling program used an x-ray fluorescence (XRF) instrument to provide semiquantitative analyses for lead and zinc to focus the sampling on areas of more significant contamination. In addition to spot sampling of the mine waste rock, some chat piles were sampled and measured with the XRF to better define metals content in the chat. XRF and subsequent laboratory wet chemical analyses indicated that some chat material contains lead concentrations in excess of 1,000 milligrams per kilogram (mg/kg). As a result, the metallurgical test work program was altered to include chat as well as the other surface-deposited mine waste materials.

Chat samples were taken again during two subsequent sampling events. A second sample was collected in September 1988 to provide additional material for the metallurgical test work program. The third sampling event was in November 1988; this sampling event provided additional information on metals concentrations at the surface and at depth on several major chat piles. A field portable XRF provided semiquantitative analyses for lead and zinc during this sampling program. All chat samples collected during the three sampling events have been submitted to a laboratory for wet chemical analyses. These results are presented in Table B-2.

The sample numbering system used for the chat samples was developed to help distinguish between the many different chat piles at the Galena subsite. For instance, Sample No. 07-01-02 represents Subsite Zone No. 7, the first pile sampled at this zone during the second sampling event. Figure B-1 shows each subsite zone and the locations of the chat piles sampled.

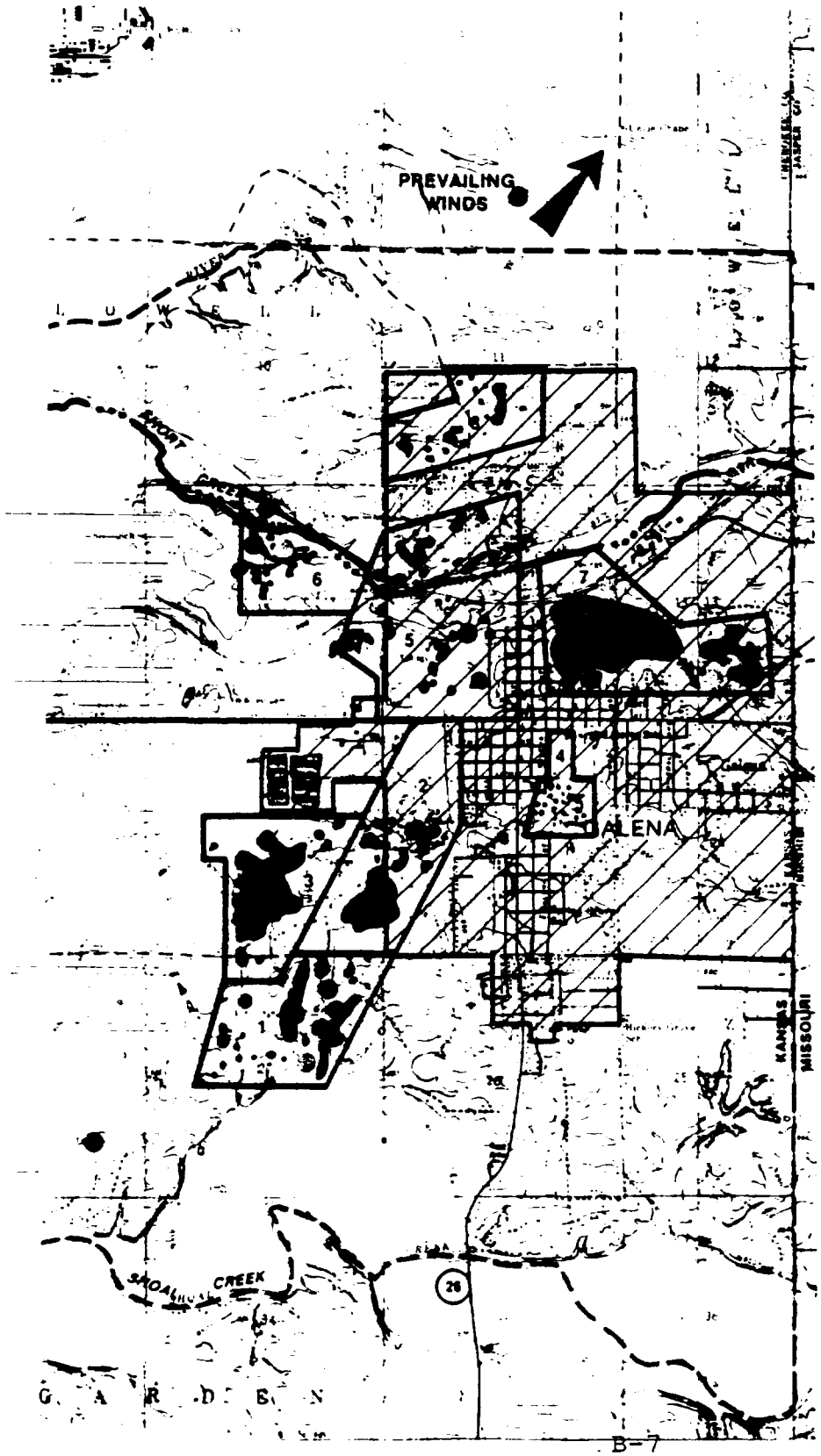
ANALYSES

FIRST SAMPLING EVENT

Four chat samples collected during the first sampling event (June 1988) were dry screened at 80 mesh (180 microns). The bulk and each of the size fractions were analyzed for arsenic, cadmium, lead, and zinc. The minus 80 mesh fraction represents that portion most likely to be windblown and ingested. Dry screening was done to prevent any soluble loss of lead compounds during the screening. The results are shown in Table B-3, which shows that the lead is concentrated in the finer fraction for three of the samples: 01-02-01, 07-01-01, and 07-02-01.

Table B-2
BULK CHAT ANALYTICAL RESULTS

Sample No.	ppm-Pb	ppm-Zn	ppm-Cd
01-01-01	3,950	4,100	20.0
01-02-01	550	400	
01-03-01	700	100	0.6
01-04-01	1,400	1,050	5.0
01-05-01	1,050	850	5.0
01-07-01	2,200	8,350	38.0
01-08-01	2,700	8,350	38.0
01-01-03 Surface	950	1,600	
01-01-03 Depth	920	1,000	
01-10-03 Depth	100	275	
01-10-03 Surface	100	690	
02-01-01	1,350	340	2.0
05-04-01	610	4,000	15.0
06-01-03 Surface	420	4,250	
06-01-03 Depth	495	4,500	
07-01-01	730	27,000	
07-01-02	520	2,010	
07-01-03 Surface	950	5,650	
07-01-03 Depth	730	3,100	
09-01-03	100	9,050	
09-02-03	60	8,250	



LEGEND

- BOUNDARY OF GALENA SUBSITE
- APPROXIMATE GALENA CITY LIMITS
- MINE WASTES
- 5 MINE WASTE STUDY ZONES
- CHAT PILES
- SEDIMENT SAMPLES

N

0 1
SCALE IN MILES

FIGURE B-1
CHAT AND SEDIMENT LOCATION AND SAMPLING POINTS
CHEROKEE COUNTY KANSAS
GALENA SUBSITE—OUFS
GROUNDWATER SURFACE WATER

Table B-3

ANALYSIS AND SCREEN DATA OF CHAT SAMPLES FROM GALENA, KANSAS

Sample No.	Sample No.	Arsenic mg/kg	Cadmium mg/kg	Lead mg/kg	Zinc mg/kg	Weight Grams	Weight Percent	Fraction of Total Lead	Fraction of Total Zinc	Cal Lead mg/kg	Cal Zinc mg/kg
01-02	Bulk	3.9	6.3	1,600	437.8	--	--	--	--	4,565	1,314
01-02	+80 mesh	4.3	5.4	2,870	1,360	401.3	91.66	--	--	--	--
01-02	-80 mesh	8.8	2.4	23,200	805	36.5	8.34	0.42	0.04	--	--
06-01	Bulk	6.5	40.4	104	11,200	286.3	--	--	--	442	10,979
06-01	+80 mesh	7.6	39.8	455	10,200	268.6	93.82	--	--	--	--
06-01	-80 mesh	11.6	33.6	238	22,800	17.7	6.18	0.14	0.13	--	--
07-01	Bulk	5.6	38.2	1,360	23,800	281.5	--	--	--	1,131	20,034
07-01	+80 mesh	7.6	33.6	761	13,400	244.3	86.79	--	--	--	--
07-01	-80 mesh	9.8	34.5	3,560	63,600	37.2	13.21	0.35	0.35	--	--
07-02	Bulk	5.1	9.2	887	2,260	451.8	--	--	--	777	1,804
07-02	+80 mesh	6.1	7.5	661	1,790	437.1	96.75	--	--	--	--
07-02	-80 mesh	6.2	8.4	4,220	2,230	14.7	14.7	0.15	0.03	--	--

To further investigate the distribution of lead and cadmium among the different size fractions, each of the size fractions separated during screen size analyses of Samples 01-02-01 and 07-01-01 were analyzed for lead, cadmium, and zinc. Each sample was wet screened at 16 size intervals between plus 3 mesh (5,000 microns) and minus 400 mesh (38 microns). The results of these tests are seen in Table B-1, which shows the lead and zinc values concentrated in the finer fractions. For example, in Sample 07-01-01, 12 percent of the sample weight (minus 10, mesh) contains approximately 50 percent of the lead, zinc, and cadmium. For Sample 01-02-01, the sample size fractions (minus 10 mesh) contains approximately 80 percent of the lead, 50 percent of the zinc, and 50 percent of cadmium concentrations in only 32 percent (by weight) of the material.

From these screen analyses, it was realized that by screening at a nominal 20 mesh (850 microns) as much as 70 percent of the lead, zinc, and cadmium could be removed by removing 20 percent of the fine fraction material. A processing method was developed so that the "dirty" chat (chat with lead concentrations in excess of 1,000 mg/kg) could be wet screened. The treatment process would produce a fine or "dirty" fraction containing the bulk of the lead, zinc, and cadmium, and a cleaner coarse fraction that would be available for commercial uses. The "dirty" fraction would be fed to the milling treatment process for recovery of lead and zinc.

Three chat samples were provided to the laboratory for the metallurgical test work. The lead and zinc analyses on these samples are shown in Table B-4. Sample 07-01-01 was used in the early phases of test work, but when the supply became low, Sample 07-01-02 was substituted. Sample 01-02-01 was also supplied to the laboratory for the screen analysis testing program for use in conjunction with Sample 07-01-01.

Table B-4
BULK MATERIAL ANALYSIS RESULTS

Description	Identification No.	Lead (Pb) (mg/kg)	Zinc (Zn) (mg/kg)
High grade mine waste	--	6,340	61,800
Low grade mine waste	07-01-01	730	27,000
Chat analysis (second sampling period)	07-01-02	520	2,010
Chat analysis (used in screening tests)	01-02-01	550	400

Results from the metallurgical test work indicated that both types of samples (mine waste rock and chat) were readily amenable to metals removal by conventional flotation. Good quality lead and zinc sulfide concentrates were easily produced using conventional reagents and simple rougher-single stage cleaner stage circuits. Oxidized lead and zinc minerals were recovered in subsequent staged flotation circuits using special chelating agents. In testing the lower grade chat samples, conventional flotation methods produced poor removals of for lead and zinc (18.8 and 42.5 percent, respectively). By using the special chelating agents to enhance oxide lead flotation and a finer grind of 100 percent minus 200 mesh, a lead recovery of 85.9 percent and a zinc recovery of 87.7 percent were achieved. Additional information on the metallurgical testing program can be found in the "Revised Draft, Technical Assistance for the Galena Subsite, Mine Waste Characterization Program, March 1989."

THIRD SAMPLING EVENT

Sampling Event 3 was conducted in November 1988 to further characterize the Galena mine wastes and chat. Chat samples were collected at both the surface and at a depth of 18 to 24 inches. Twenty samples were collected from the surface and at depth for each chat pile and were composited. The two composited samples for each pile were split using a riffle splitter to obtain the final surface and depth composite sample for metals analyses.

Samples 01-10-03 and 01-11-03 were collected from a large chat pile adjacent to Blackjack Road, southwest of Zone 1. At EPA's request, Samples 09-01-03 and 09-02-03 were collected from a chat pile near an old mining and milling site north of Zone 8 near the Union Chapel School.

These samples show little difference in lead and zinc concentrations in the first 2 feet of depth. To determine the average chemical analyses of each chat pile, including deeper locations, samples would have to be collected by drilling or auguring each pile at a much greater depth and at several different locations. This would provide an indication to what degree the metals concentrations at the surface have been reduced by weathering.

Stream sediments were also collected from three drainages within the subsite. Sediment Sample 01-01-03S was collected from the drainage flowing from Zone 1 into Shoal Creek. Sediment Sample 06-01-03S was collected from the Owl Branch flowing from Zone 6 into Short Creek and Sediment Sample 07-01-03S was collected from the drainage flowing from Hell's Half Acre into Short Creek. Approximately 100 grams of sediment was collected at 50-foot intervals along the

stream channel. This composited sample was blended and reduced by splitting using a riffle splitter. The results of the wet chemical analyses on sediments are provided in Table B-5.

Table B-5
SEDIMENT ANALYSIS

Sample No.	Lead (mg/kg)	Zinc (mg/kg)
01-01-03S	580	2,850
06-01-03S	560	4,300
07-01-03S	1,250	4,950
07-02-03S	820	3,800

The test results indicate that the stream sediments from the three sampled streams are not heavily contaminated with lead, with the exception of Sample 07-01-03S. This sediment sample from Hell's Half Acre has a lead analysis of 1,250 mg/kg, which is above the action level of 1,000 mg/kg.

XRF RESULTS

The field portable XRF was used to determine the lead and zinc concentrations in chat within the Galena Subsite zones. The XRF uses a low activity radioactive isotope to excite the elements in a sample. The excitation causes the elements to "fluoresce" at an energy specific to each individual element. The amount of energy given off by each element is measured by the XRF instrument as an index number. The index number is correlated to laboratory determined element concentrations (standards) to calibrate the instrument. The calibration curve developed by correlating the XRF index to

laboratory measured concentrations for specific elements allows field personnel to calculate concentrations from index numbers. Table B-6 provides a comparison of XRF values and wet chemical analyses for samples collected during Field Sampling Event 3.

Table B-6
COMPARISON OF XRF AND WET CHEMICAL CHAT ANALYSES

	Lead Analyses		Zinc Analyses	
Sample No.	Chemical (mg/kg)	XRF (mg/kg)	Chemical (mg/kg)	XRF (mg/kg)
01-01-03	950	620	1,600	1,550
01-02-03	920	920	1,000	1,620
01-10-03	100	60	275	1,150
01-10-03	100	90	690	1,280
01-01-03S	580	590	2,850	2,280
06-01-03	495	720	4,560	4,020
06-02-03	420	460	4,250	3,480
06-01-03S	560	720	4,300	4,150
07-01-03	950	960	5,650	4,550
07-02-03	730	850	3,100	2,150
07-01-03S	1,250	890	4,950	2,880
07-02-03S	820	720	3,800	4,020

Appendix C
Review of Potentially Responsible Parties' Leach Test Report

Section 1
INTRODUCTION

This appendix presents CH2M HILL's comments from review of the potentially responsible parties' (PRP) Results and Analysis of Leaching Tests report dated December 5, 1988. Review comments are presented in the same order of progression as the text in question is presented in the PRP report. In addition, to assist the reader, the review comment is preceded by the PRP report section heading and the text being discussed.

1.1 INTRODUCTION

1.1 BACKGROUND, PAGE 1-1

Report Text

Over geologic time, oxygenated water has reacted with metallic sulfides in mineralized areas and has produced acidic water with relatively high concentrations of dissolved metals, including iron, manganese, cadmium, lead, and zinc. These waters containing metallic ions discharge to local creeks, principally Short Creek.

Response to Text

There are no measurements indicating that acidic water with relatively high concentrations of dissolved metals discharged into local creeks over "geologic time." There are no chemical data before the late 1800s, or even premining, to document that acidic water, let alone dissolved metals, were being generated.

1.1 BACKGROUND, PAGE 1-3

Report Text

Work to date by EPA, primarily in 1987 and 1988, has led EPA to propose a remedy that, in addition to some remediation of

surface water drainage and of potential cross contamination of the deep aquifer due to boreholes or failed well casings, included as a major component the conventional milling of waste materials at the site prior to disposal of milled tailings in mine voids.

Response to Text

Milling and flotation was proposed to permanently remove the lead, zinc, and cadmium from the mine wastes. The resulting tailings were then to be placed into the mine voids.

This flotation step is missing from the PRP's description throughout the text. Both milling and flotation are assumed to be combined under the terms "conventional milling."

1.2 SEPTEMBER 1988, DIRECTION FROM THE USEPA

1.2 SEPTEMBER 1988, DIRECTION FROM THE USEPA, PAGE 1-4

Report Text

Although the original direction from EPA required completion of the PRP effort by mid-September, the Agency ultimately allowed the PRPs until the end of November to complete the experimental program.

Response to Text

The draft of final data analyses was not received until December 27, and the final data on the experimental program were not received until the first part of January 1989.

Report Text

Although the work plan transmitted by EPA did not include testing of tailing material representative of that which would be produced by EPA's remedy, the PRPs suggested inclusion of such testing.

Response to Text

Was the material used for testing the EPA "milling" proposal really representative of the samples EPA used in their work? Specifically, did it include the grind size, reagents, and flotation circuits to produce a tailing that would pass the EP toxicity criteria?

1.3 OBJECTIVES OF THE CURRENT STUDY

1.3.2 SPECIFIC OBJECTIVES, PAGE 1-5

Report Text

The second objective was to estimate impacts on water quality of the PRP's Additional Alternative, both with respect to ground and surface water quality, and mass flux.

Response to Text

Are the PRPs proposing that these tests estimate the impacts to both surface and groundwater quality and the total mass flux? These tests, as designed by EPA, characterize rainfall events and rainwater moving through the wastes under unsaturated and saturated conditions. The impact of the leaching of these materials when backfilled into mine voids containing acidic groundwater was not included in the test program. Therefore, without appropriate test data, it is very difficult to estimate the impact of such occurrences on the surface and groundwater quality and mass flux.

Section 2
PROJECT ACTIVITIES

2.1 SAMPLING

2.1 SAMPLING, PAGE 2-3

Report Text

Following crushing, the minus-100 mesh material was transferred to Hazen Research for bench-scale conventional milling, following the procedures developed by them for EPA.

Response to Text

The sentence is unclear. Was the entire waste rock sample split, crushed to less than 100 mesh, and sent to Hazen, or was the sample crushed and only the less than 100 mesh material sent to Hazen? If the latter, were both fractions weighed and analyzed?

The intent was to produce a tailing using procedures developed for EPA by Hazen. However, no detailed data have been provided to determine if this is, in fact, the case.

2.2 SAMPLE PREPARATION AND HEAD GRADE ANALYSIS

2.2 SAMPLE PREPARATION AND HEAD GRADE ANALYSIS, PAGE 2-4

Report Text

The waste-rock samples were composited and thoroughly mixed in a 55-gallon steel drum; the chat samples were composited and mixed in a 45 gallon drum.

Response to Text

Appendix C says chat samples were composited in a 30- not a 45-gallon drum.

Report Text

Splits of waste rock (in three grain-size/handling fractions), chat, and milled tailings were collected and prepared for various batch leaching tests, Extraction Procedure Toxicity tests, and acid-producing potential/total sulfur (waste rock and chat only) tests;

Response to Text

How were the splits sampled? Was it by random grab samples or use of a sample splitter or other method? What QA/QC procedures were used to document that representative samples (or to establish the variability of sampling) were analyzed and used in the columns?

Report Text

waste rock, chat, and milled tailings samples were prepared for column leaching tests, as directed in the EPA work plan.

Response to Text

The samples were not prepared for column leaching tests as directed in the EPA draft work plan. Several modifications were suggested and accepted.

2.3 LEACHING EXPERIMENTS

2.3.1 BATCH SAMPLING, PAGE 2-5

Report Text

Batch leaching tests were performed to obtain a rapid estimate of the extent to which the samples could be leached, and to compare with the more elaborate column testing to be performed later.

Response to Text

"Perfomed" should be "performed." Batch leaching tests do not necessarily give "the extent to which the samples could

be leached" for all metals, but are probably more representative for zinc and cadmium than for lead--all three are more representative than for iron.

2.3.2 EP TOXICITY TESTING, PAGE 2-5

Report Text

The U.S. EPA Extraction Procedure (EP) Toxicity test (EPA protocol SW846) was performed on splits of the three waste-rock fractions that were batch leached.

Response to Text

How were the splits sampled?

2.3.3 COLUMN TESTS, PAGE 2-7

Report Text

Method 2-"Saturated." Method 2 columns were designed and operated to simulate the effects of fully saturating (in a flow-through environment) waste rock and milled tailings. The columns were flooded with pH 5.5 de-ionized water and allowed to stand for 2 days. On days 2, 5, 9, 15 and 28, an additional 823 cc of de-ionized water was allowed to flow through the system for 2 hours, displacing an equivalent volume, which was collected for analysis.

Response to Text

It is agreed that Method 2 will simulate the effects of a flow-through environment using surface water as the quality of water in that environment. The text does not deal with acidic waters present in numerous locations at the site.

Report Text

The Method 2 (fully saturated) procedure was also adapted for use with the menus-100-mesh milled tailings sample, to simulate the effect of disposing of this material below the water table.

Response to Text

Does minus 100-mesh milled tailings sample indicate the size range that flotation was conducted?

Report Text

The column experiments were designed and executed as directed in the EPA work plan except for the following minor modifications (all of which were discussed in advance with staff of CH2M HILL, and copies of notes taken during these discussions are attached to Appendix A).

Response to Text

"The EPA work plan" is a draft work plan and as such, it was not intended to be a final work plan.

2.3.3 COLUMN TESTS, PAGE 2-8

Report Text

The diameter of the columns was 8-inches, reather than 10-inches.

Response to Text

"Reather" should be "rather."

Report Text

Hydrochloric acid was to be used to adjust pH, as necessary, in lieu of sulfuric acid, because sulfate was one of the analytes for the testing program.

Response to Text

The use of hydrochloric acid was approved, but do the authors care to comment on the effect of hydrochloric versus the effect of sulfuric acid on the metals and their concentrations leached from the samples?

Report Text

The leachant used was de-ionized water, with an initial pH of approximately 5.5 rather than 7; no adjustment with acid was required since the initial pH was below 7.0.

Response to Text

Was the deionized water analyzed for metals?

Report Text

It is important to note that the results for Method 2 experiments at Day 37 are not comparable to the results from Days 0-28 because the EPA Work Plan directed that the protocol be modified to include a final rinse.

Response to Text

Why is it important to note the difference between the results of Day 37 and the results from Days zero through 28? What protocol is being modified? A comparison of the data is valid.

2.4 CHEMICAL ANALYSIS

2.4 CHEMICAL ANALYSIS, PAGE 2-9

Report Text

All chemical analyses (unless otherwise cited) were performed at Core Laboratories, Aurora, Colorado. All analyses were performed in accordance with EPA procedures, for those procedures that have been published.

Response to Text

What analyses were not performed in accordance with EPA procedures?

2.5 FLOW AND GEOCHEMICAL MODELING

2.5.1 FLOW MODELING, PAGE 2-10

Report Text

The additional alternative will place waste rock below the groundwater surface, and cover the remaining depressions with chat. These materials are both of very high permeability, which is not reduced by leaching (see results in Appendix C). Thus, the infiltration of flow to the underground workings will not be changed by this activity, nor will the flow within the workings be changed.

Response to Report

Infiltration should be reduced by surface contouring and rerouting surface water away from mine openings. Is this not planned in the PRP alternative?

The flow within and, therefore, the total mass flux should change because of reduction of both infiltration and open space. An analogy would be taking an open pipe and filling it with gravel. Would the gravel-filled pipe have the same flow and mass flux as the open pipe? Further flow modeling is still proposed unless the PRPs have an alternative method of dealing with the reduced flow.

2.5.2 GEOLOGICAL MODELING, PAGE 2-11

Report Text

If the amount and direction of fluid flux are unchanged, the only factors that can change the flux and the instantaneous concentration of metals in solution from their present values as a result of the additional alternative are changes to the chemistry of the source term and any chemical reactions that may occur in the flow system. These changes were addressed by geochemical modeling.

Response to Text

We agree that the chemistry of the "source term" and "any" chemical reactions that may occur in the flow system are very important (basis of this is physical testing) but geo-

chemical modeling by PHREEQE doesn't address the changes. PHREEQE is an equilibrium model. It compares the water chemistry with a known mineral suite under known physio-chemical conditions (temperature, pH, Eh) to an idealized coefficient measured with individual pure minerals under laboratory conditions. It does a fairly credible job of addressing simple equilibrium-condition calculations with known simple mineralogy. Adsorption processes are not addressed.

No mineralogy was conducted on the samples used. The existing mineralogy is unknown and certainly not simple (few pure minerals totally exposed to leaching conditions remain as a pure, essentially infinite source term). This is a serious problem because PHREEQE requires that the input include all the minerals, compounds, and species. Using the MINTEQ data base improves the mineral suite for PHREEQE to consider, but without knowing the minerals, especially the initial suites of oxidation minerals, the calculations are, at best, able to give an estimate of the propensity for a mineral to go into solution or to precipitate.

Kinetics of the reactions is not considered by PHREEQE. Oxidation reaction kinetics of metal sulfides are very sensitive to pH. For example, iron oxidation at pH of less than 6 takes days to precipitate but at pH 7 only minutes and, at higher pH, iron oxidizes to oxyhydroxides in seconds.

Adsorption is not being considered. Adsorption is also very strongly pH dependent. For example, adsorption on iron oxyhydroxide, lead is almost completely adsorbed above a pH of about 6, but cadmium adsorption is not significant until a pH of about 8.

The pH considered in these calculations is based on the pH generated by the waste, but the actual pH in the mine working

has been documented to be as low as 3.9. Equilibrium, kinetics, and adsorption at a pH between 4 and 5 result in much higher metal concentrations being transported than at column leach pH units. The above discussions relate to inorganic considerations. Bacteria present in the system will accelerate the oxidation process below, and to some extent above, the water level in the mine workings. The oxidation reactions are accelerated almost without regard for the pH, certainly within the pH range 3.5 to 10. The influence of bacteria is not evaluated in the current tests, but comparable testing in other areas indicate that one order of magnitude increase in dissolved metal concentration is not unusual.

2.5.2 GEOCHEMICAL MODELING, PAGE 2-12

Report Text

It is considered that these activity diagrams provide useful, though not necessarily quantitative, information to explain some significant portions of the solution chemistry of the experimental and field data.

Response to Text

The diagrams are useful to explain some simpler portion of the solution chemistry. See the above discussion.

Section 3

RESULTS

3.1 CHARACTERISTICS AND LEACH BEHAVIOR OF WASTE ROCK

3.1 CHARACTERISTICS AND LEACH BEHAVIOR OF WASTE ROCK, PAGE 3-1

Report Text

Evaluation of the data from the various tests performed waste rock provides not only data that can be used to assess source-term characteristics, but also provides a great deal of information about the processes that are likely to be important in the fate of metals.

Response to Report

Some words are missing: "the various test performed" on the (?) "waste rock. . . . "

3.1.1 WASTE ROCK HEAD GRADES, PAGE 3-1

Report Text

Total metals analyses from splits of the waste rock are presented in the first column of Table 3-1.

Response to Text

Are the "Total metals analyses" conducted using EPA CLP procedures?

3.1.1 WASTE ROCK HEAD GRADES, TABLE 3-1, PAGE 3-2

Report Text

Table 3-1
TOTAL METALS CONCENTRATIONS IN VARIOUS WASTE ROCK SAMPLES

Metal		Minus 1" Recomposited Waste Rock	-100 Mesh Mill Feed
Cadmium	mg/kg	58	40
Iron	mg/kg	8,332	8,200
Lead	mg/kg	1,393	4,230
Manganese	mg/kg	1,126	140
Zinc	mg/kg	13,424	7,000
Acid potl	ton/$caCO_3$/kt	9.6	NA
Total S	%	0.66	NA
Total S	ton $CaCO_3$/kt	20.6	NA

Response to Text

The head assay for the minus 1-inch material does not compare well with the analysis of the same material crushed and homogenized at minus 100-mesh. The head assay for the minus 100-mesh is likely the more representative because of the difficulties in sampling coarser materials.

3.1.2 WASTE-ROCK AND EXTRACTION PROCEDURE TOXICITY TASKS, PAGE 3-3

Report Text

As can be seen, lead is the only metallic species that is leached in appreciable quantities from the waste rock under this protocol. Furthermore, both the as-received waste rock and the recomposited waste rock do not possess the characteristic of EP toxicity. The elevated lead concentration in

the test of the waste rock fines (minus 1-inch) emphasizes the importance of grain-size effects in the mobility of metallic species in this setting. The minus 1-inch material constituted about 36 percent of the total sample.

Response to Text

The potential for highly variable analytical results, such as nugget effect, is much greater with samples from these sources, yet these are accepted as accurate to the extent that "waste rock fines (minus 1-inch)" supposedly emphasize the grain-size effect in the mobility of metallic species. Grain size does affect the mobility, but to explain a concentration in one instance in which the grain size is 0.0059 "on a nugget effect" and then suggesting that a concentration from samples containing grains as much as 1-inch in diameter is a fairly exact number (lead EP TOX 3.87 versus 5.00 mg/l) does cause a logic problem.

3.1.3 WASTE-ROCK BATCH TESTING, PAGE 3-4

Report Text

Very little metal is leachable from the as-received rock, and both pH and Eh were essentially unaffected.

Response to Text

"Very little metal is leachable" is, relatively speaking, compared to the waste rock fines? The drinking water standard is exceeded by 390 μg/l. Even more interesting is the fact that the metal concentrations (with the exception of iron) increased in the 1-inch waste rock fine-batch-leach, but the pH was 1 unit higher than the as-received waste rock leach.

Report Text

When the coarse-grained fraction is crushed to minus 1-inch, metal leaching and Eh/pH behavior approach that observed for the fine-grained fraction.

Response to Text

Neither cadmium nor iron concentrations "approach those observed for the fine-grained fraction." Sulfate increases many times over that in either the as-received waste rock or the fines. This phenomena should be explained. Lead and manganese are essentially proportional to each other in the recomposited waste rock leach compared to the fines (ratio of 0.67 and 0.70, respectively) yet the zinc is only proportionately half the lead and manganese concentration (0.34)--why is that?

Mineralogy, surface area, and so forth are dominant depending on what metal or chemical parameter we wish to describe.

3.1.3 WASTE-ROCK BATCH TESTING, PAGE 3-6

Report Text

The data are best interpreted as indicating that the short-term behavior of waste rock is dominated by surface-area effects, since the scalped fines are a very great deal finer grained.

Response to Text

The behavior regarding the geochemistry of the scalped fines could also be explained by the fines having a different mineralogy (dominantly) than the bulk rock. This is logical based on the physical characteristics of the sulfide mineral enclosed in the brittle siliceous rock matrix. When the rock is broken, the sphalerite, galena, and so on (metal

sulfides) pop off of surfaces. Sphalerite typically shatters with this activity, whereas, the galena crystal sizes generally remain unchanged.

Report Text

It is considered that crushing the rock not only increased the total surface area, but also exposed fresh faces to redox reactions and subsequent dissolution of metals and sulfate.

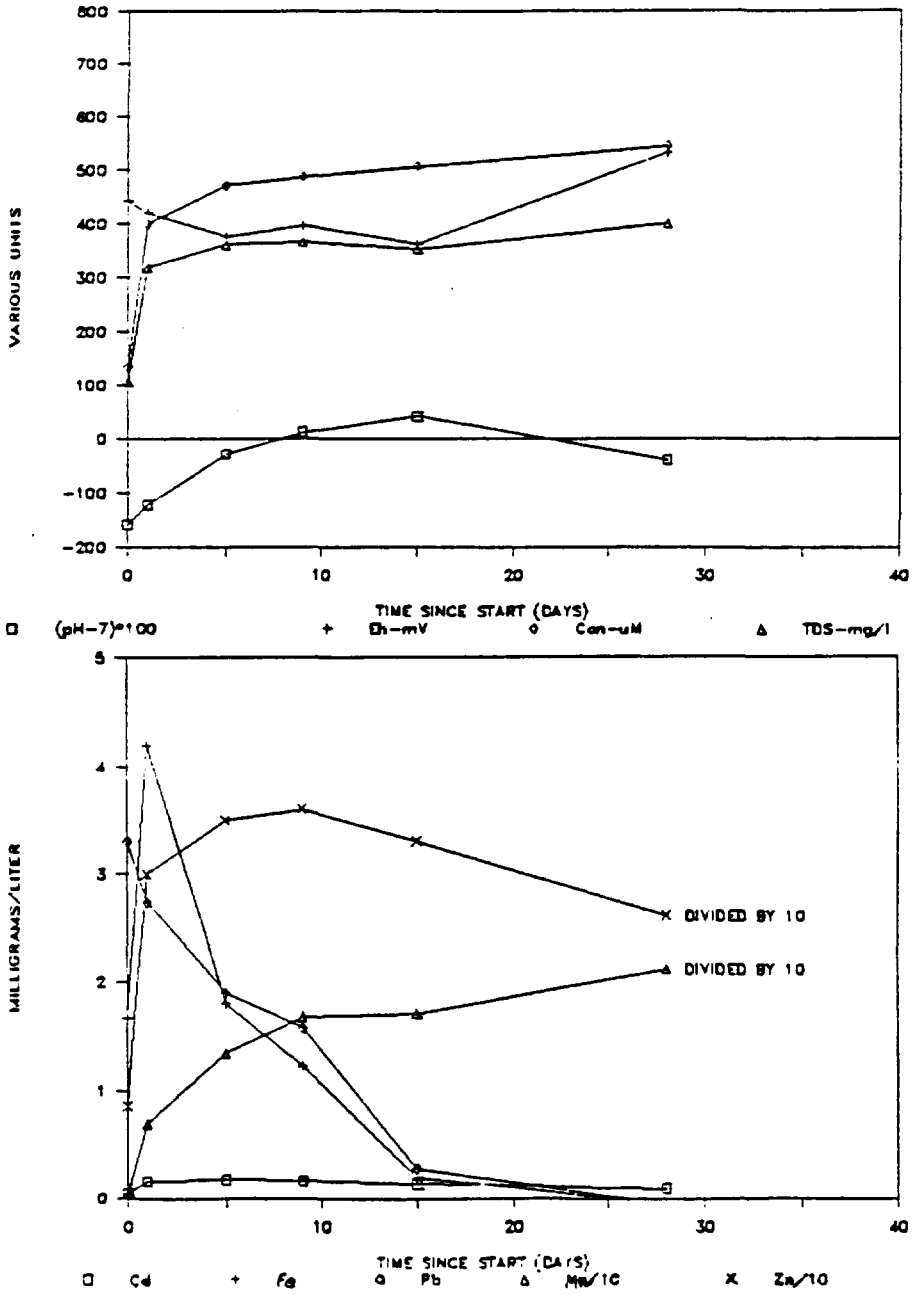
Response to Text

This sentence is certainly true. Mining and subsequent crushing of the rock increases the total surface area and exposes previously enclosed fresh crystal faces to chemical reactions that can lead to dissolution and transport of metals.

3.1.3 WASTE-ROCK BATCH TESTING, PAGES 3-7 and 3-8

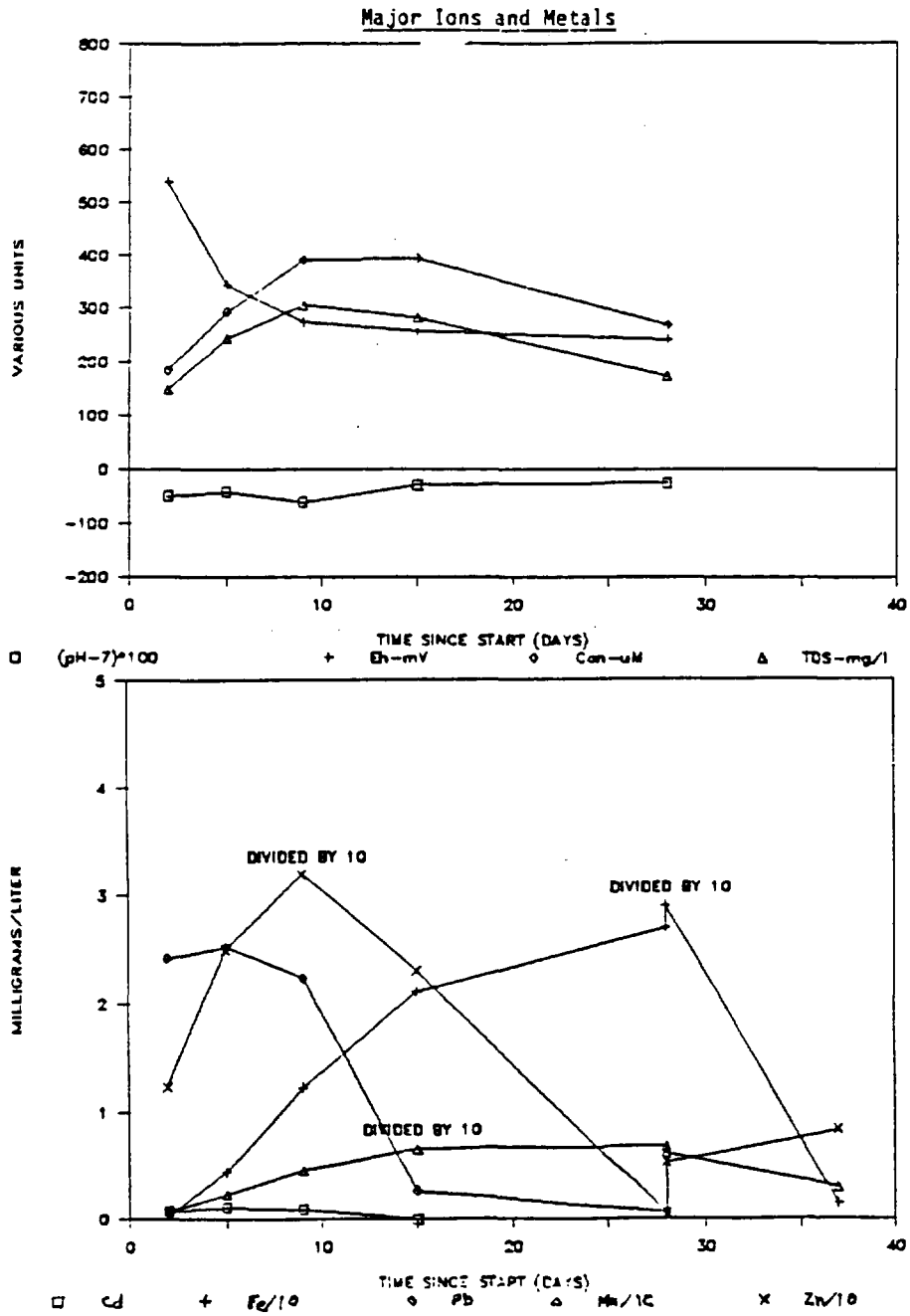
Report Figure--Figure 3.2--Waste Rock Column Leach Results (Method 1--Variably Saturated

Figure 3.2 - Waste Rock Column Leach Results (Method 1 - Variably Saturated)



Report Figure--Figure 3.3--Waste Rock Column Leach Results (Method 2--Fully Saturated

Figure 3.3 - Waste Rock Column Leach Results (Method 2 - Fully Saturated)



Response to Figures

Looking at what is plotted versus the available tables in the Appendix C, why are only the dissolved parameters plotted? There are total and dissolved metals data available. Are the other undesignated numbers in the table "dissolved" or "total?" Review comments assume dissolved.

Why are Day 37 results not on the graph? Text and graphs ignore the Day 37 results.

3.1.3 WASTE-ROCK BATCH TESTING, PAGES 3-9, 3-10, AND 3-11

Report Text (page 3-9)

In both the Method 1 and the Method 2 experiments, pH rises and Eh decreases during the test. (However, note that there is a small break in this trend at Day 28 in Method 1, the final day of the test.) For both cases, the pH initially rises from below 5.5 to above 6.5, which indicates that the rock has at least some capacity to offset any acid drainage that is generated. Furthermore, in both experiments the pH rises into the range at which solubility control for cadmium and lead is expected to be important and at which pH control on even zinc solubility is being approached. This is supported by the trends in the dissolved metals values, particularly in the case of the saturated experiment, which by Day 28 has cadmium, lead and zinc values at or below the relevant MCL values. Based on the pH, TDS and conductivity trends, it seems that the major-element chemistry does not differ significantly under these experimental conditions between the two sets of leaching conditions.

Response to Text

The pH initially rises from below 5.5 to above 6.5 in only Method 1, not both methods. Method 2 really does not change pH very much; 6.14 at day 28 to 6.70 at day 15. Method 2 needs to be replotted. The pH as shown does not agree with pH in the analytical report. The pH actually decreases with time. Neither test indicates a capacity to offset acid drainage.

Note: See above discussion for Page 2-11 regarding solubility control by pH. In both cases, the pH indicates decreasing pH near the end of the test.

"At or below relevant MCL values." Cadmium is at its MCL. Dissolved lead is below its MCL, but total lead exceeds the MCL (0.95 mg/l versus an MCL of 0.05 mg/l). If the "relevant MCL" value for zinc is the secondary drinking water standard of 5.0 mg/l, then both methods failed for both dissolved and total zinc.

Report Text (page 3-9)

In the Method 2 experiment, iron and manganese tend to increase steadily throughout the time period observed. When looked at in Eh-pH space, this can be readily explained. In both sets of experiments, the Eh-pH conditions are very close to the FeO(OH) - $Fe^{2+}_{(aq)}$ boundary. (See Section 2.5.2 concerning assumptions in use of activity diagrams.) The geochemical modeling presented in Appendix D shows that essentially all groundwaters (including mine waters) in the Galena subsite area are saturated with respect to FeO(OH). Thus, trends of decreasing Eh and increasing dissolved iron during leaching indicates that ferric (Fe^{3+}) iron is being released by dissolution of FeO(OH), and then reduced to ferrous (Fe^{2+}) iron, presumably through the oxidation of sulfide to sulfate, which increases during the tests, with subsequent dissolution of cadmium, lead and zinc. The difference in behavior of iron in Method 1 may be due to continued control on iron solubility by ferric oxyhydroxide.

The argument with respect to manganese is generally similar: the fluids remain in the $Mn^{2+}_{(aq)}$ field. Thus, it is expected that available manganese will remain in solution. As discussed in Hem (1985), the Eh-pH relations can be used to explain the similarities and differences (including the "lag" in solubility control as oxyhydroxides) between iron and manganese solution chemistry, and the trends seen in the waste-rock column tests are consistent with the expected behaviors of the two species, given the Eh-pH data for the waters.

For both experiments, the fresh-face phenomenon can be observed: the solution chemistry changes most rapidly for all parameters during the first 10 days, following which the rate of change in solution chemistry decreases for all parameters. These data, then, are consistent with the batch

test results in indicating that it is prudent to avoid crushing or otherwise increasing surface area and developing fresh mineral faces.

Response to Text

It is difficult to believe that the decreasing Eh from 273 (day 9) to 257 millivolts (day 15) as pH increases from 6.39 to 6.70 is responsible for iron increasing from 12.3 to 21.0 mg/l and for manganese from 4.57 to 6.60 mg/l. Eh/pH roughly balance one another. Did the author actually calculate the equilibrium iron and manganese changes?

Only the first two sentences say anything relevant to the discussion.

The rate of change in solution chemistry does not decrease for all parameters during the first 10 days. This is largely true for Method 1 (unsaturated) but certainly is not true for Method 2 (saturated), which clearly shows a greater rate of change in lead concentration after 10 days than before it. Furthermore, the data collection does not lend itself to definition of rate changes. It was designed to define the concentration changes with time. This is one of the few times that the author acknowledges the kinetics of these changes. The kinetics are related to chemical reactions of precipitation, dissolution, exchange, adsorption, and biological activity as discussed for review on page 2-11.

Report Text (page 3-10)

The major differences between the two experiments are in the absolute values of Eh and dissolved metals under the two protocols. While Eh falls in both experiments, it decreases substantially more in the Method 2 (saturated) tests. This, of course, is to be expected, as the Method 2 column was never reopened to atmosphere during the test cycle. As the Eh falls with increasing pH, the cadmium, lead and zinc values also fall toward and to MCL limits in Method 2.

Response to Text

Here is where a discussion of what happens when the Eh changes--between day 28 and day 37 results. It is true that the 28-day test revealed trends of decreasing metals in solutions. All tests indicated decreasing dissolved metal with time. However, a recharge event, such as rainfall, is simulated following a 9-day hiatus in saturation in Method 2. The chemistry indicates that the Eh increased from the lower 200 to the middle 400 millivolt range, pH decreased slightly and cadmium (0.02 mg/l) lead (1.68 mg/l) and zinc (8.4 mg/l) all exceeded MCLs again indicating a quasi-equilibrium established during saturation is really more of a cycle in which changing hydraulic conditions probably have more of an impact on metal mobility than time.

Report Text (page 3-11)

The trends in dissolved metals are not as dramatic in Method 1 tests. The three metals of principal concern, cadmium, lead, and zinc, decline in concentration after day 2, but do not drop to detection limits. Based on the rest of the solution chemistry, the difference in behavior of the metals appears to be related primarily to the significantly higher Eh of the solutions at late experimental times (approximately 400 mV in the Method 1 experiments compared to approximately 200 mV in the Method 2 experiments). note that the relationship probably is not a direct one: cadmium, lead and zinc occur in only one valence state, and therefore their concentration should not reflect direct redox effects. Rather, it is likely that the metals are responding to oxidation of sulfide to sulfate, as indicated by TDS (see also Appendix C), which rises sharply by Day 2 and then increases gradually until Day 28, coupled with the rising pH of the solutions, which would be expected to exert some significant control on the concentrations of the metals.

Response to Text

The effect of adsorption is probably more important in Method 1 than either equilibrium or oxidation. An Eh in the 100 millivolt range at near neutral pH causes the iron to

oxidize (FeOOH) and the lead to be adsorbed. Note the strong correspondence between iron and lead concentration. Zinc and cadmium are not as strongly adsorbed at this lower pH and manganese is slow to oxidize.

If oxidation of sulfide to sulfates is important in the increase in TDS, it is not reflected as expected in the pH. The pH increases for the first 15 days rather than decreasing if sulfuric acid was being formed as required by iron sulfide oxidation. Furthermore, both sulfate and TDS appear to approach a constant after about 5 days suggesting some form of equilibrium that is not a function of time. The mineralogy would have to be known to identify what is going on. Why does the normally conservative sulfate appear to decrease from 214 mg/l at day 5 to 180 mg/l at day 15?

Report Text (page 3-11)

With respect to evaluating the Additional Alternative, the most significant result of the two sets of column leaching tests is that placing of the waste rock below the water table can be expected to significantly decrease the leaching of metals when compared to the leach rate expected if it is located above the water table.

Response to Text

Placing the waste rock below the water table does not decrease the leaching of metals compared to emplacement above the water table. Emplacement above the water table releases much less iron, less lead, and probably less cadmium because the mine waters have a chance to achieve an equilibrium with repeated recharge events. Zinc and manganese will probably achieve some asymptotic concentration in a few months. The acidic groundwater will achieve some type of quasi-equilibrium releasing and mobilizing metals with each recharge event. The mine wastes below the water table will not be able to achieve equilibrium because of the

changing environmental conditions in the recharge water quality resulting from rainfall/snowmelt events. This cycle will continue until only the silica is left.

3.2.1 CHAT HEAD GRADE, PAGE 3-12

Report Text

Like the waste rock, the chat that was prepared for this project (Appendix C) was also towards the upper end of the metal content of chat samples that have been obtained over the life of the project. The composite chat sample analysis is presented in Table 3.4.

Table 3.4
TOTAL METALS CONCENTRATIONS IN THE COMPOSITE CHAT SAMPLE

Metal	Unit	Chat
Cadmium	mg/kg	10
Iron	mg/kg	7,649
Lead	mg/kg	486
Manganese	mg/kg	972
Zinc	mg/kg	2,999
Acid Potl	ton $CaCO_3$/kt	-0.3
Total S	Percent	0.16
Total S	ton $CaCO_3$/kt	5.0

Response to Text

The composite chat sample analysis should be compared with the chemistry range of comparable chats collected in the area if the text is going to make a comparison.

3.2.4 COLUMN LEACHING OF CHAT, PAGE 3-13

Report Text

As directed by the EPA Work Plan, Column leaching of chat was performed only using Method 1, that is for conditions simulating variably saturated flow above the water table. The results are presented in Figure 3.4, and are consistent with geochemical expectations. The leaching behavior is

dominated by the fine-grained nature of the chat and the relatively high and constant Eh, producing higher dissolved concentrations of cadmium and zinc, despite lower head grades and higher pH values than in the waste-rock tests. These trends can only be explained reasonably on the basis of a grain-size effect.

Response to Text

Surface area of the material is important. Grain size is one way of determining the surface area exposed to leaching.

PAGE 3-13, FIRST PARAGRAPH LAST LINE

"resonably" should be "reasonably."

Report Text

Two trends in the metals data are worth noting. First, there is a very close relationship between zinc and cadmium concentrations at all times, suggesting that they are being released from the same phase in the chat. Second, there is very little leachable lead in the chat samples under the experimental conditions, as would be expected for pH in the range of 6.6 to 7.6.

Response to Text

There is a relationship between cadmium and zinc--both are present in sphalerite and its oxidation products. However, the ratio of cadmium to zinc decreases with time. Cadmium continually decreases after day 2. Zinc, on the other hand, is erratic and actually analyzed as higher after 28 days than at the start.

Report Text

Note that there is a small change in the trends at Day 28 of the Method 1 leaching, showing small decrease in pH and increase in Eh and Zn. The best available explanation for this is that a "nugget" of reactive sphalertie was encountered during the Day 15 through Day 28 interval, temporarily releasing acid and Zn. However, it is worth noting that the other metals were unaffected.

Response to Report

There is no small decrease in pH in the data in Appendix C. In fact, the pH increases to an asymptotic level of between 7.55 (day 9) and 7.58 (day 28). With this exception, the erratic change in Eh, the sudden appearance of iron (0.17 mg/l) at day 15 increasing to day 28 (0.30 mg/l), and the changes in sulfate and zinc lead to an alternative hypothesis involving the potential of marcasite oxidation enhanced by biological activity resulting after an incubation period of about 2 weeks. The missing sulfate value for day 28 and no mineralogy makes this difficult to evaluate, but oxidation of a "nugget of reactive sphalerite" that the solution missed for 2 weeks is difficult to accept. Other metals are being affected.

3.3.1 HEAD GRADE OF MILLED TAILINGS, PAGE 3-14

Report Table

Table 3.5
TOTAL METALS CONCENTRATIONS IN THE MILL FEED

Metal	Unit	Chat
Cadmium	mg/kg	40
Iron	mg/kg	8,200
Lead	mg/kg	4,230
Manganese	mg/kg	140
Zinc	mg/kg	7,000

Response to Report

Assuming that the listed concentrations are the head grades and not the treated mill feed, it is difficult to compare the amount of total metals remaining in the sample that

could be leached with the chat and mine waste samples. We need the analysis after flotation.

3.3.2 EP TOXICITY, TABLE 3.6, PAGE 3-16

Table 3.6
EXTRACTION PROCEDURE TOXICITY TEST DATA FOR MILLED TAILINGS

Metal		Milled Tailings 100 Mesh	Standard 40 CFR 261 Table 1
Arsenic	mg/l	-0.5	-5.0
Barium	mg/l	0.4	100.0
Cadmium	mg/l	0.02	1.0
Chromium	mg/l	0.01	5.0
Lead	mg/l	12.3	5.0
Mercury	mg/l	-0.003	0.2
Selenium	mg/l	0.1	1.0
Silver	mg/l	-0.01	5.0

Note: "-" indicates less than detection limit shown.

Response to Table

The amount of lead released in the EP TOX test indicates a poor flotation cycle. Properly processed in the mill circuit, the lead should not fail the EP TOX test if the concentration of lead in the tailing is less than about 700 ppm. On this basis, the "Additional Alternative," is certainly not superior to the milling alternative.

3.3.3 BATCH LEACHING, PAGE 3-17

Report Text

The pH of the extract, 10.82, approaches the pH 12.5 value used by EPA to distinguish hazardous alkaline solutions, and the TDS and SO_4 concentrations of 646 mg/l and 238, respectively, are a factor of 20 greater than the comparable values for as-received waste rock and 5 to 10 times higher than the values for even the minus 1 inch as received fraction of the waste rock.

Response to Text

A pH of 10.82 does not "approach" a pH of 12.5. It is more than 10 times smaller.

What is missing here, however, is the fact that although TDS and sulfate are of interest, the metals are the focus of this work. Iron is higher for the batch leach of milled tailings but is a secondary drinking water standard. Lead, manganese, and zinc are all more than 10 times lower in the milled tailing leachate than the leachate of the waste rocks. Again, it is believed that the milled tailings used for this experiment are not representative of those produced by the EPA-developed milling method. Even though the tailings material is not representative, based on the above metal comparison, the water quality impacts of the milled rock is still much less than the as-received bulk waste rock.

Report Text (page 3-18)

While most of the metals are below detection limits at pH 11 and Eh 260 mV (probably controlled by carbonate solubilities, see Garrels and Christ (1965)), there is iron in solution, consistent with a simple Eh-pH diagram prediction that the solution would be in the stability field of $Fe(OH)_4^{1-}$ (aq) (Hem, 1985, Fig. 15).

Response to Text

It is unclear if the metals concentrations listed in Table 3.6 are dissolved or total. This is particularly important if one could compare the data with the reference listed. Why is the diagram not shown if it is pertinent to the discussion?

3.3.4 COLUMN LEACHING OF MILLED TAILINGS, PAGE 3-18

Report Text

The results of the column leaching test of the milled tailings are presented in Figure 3.5. It should be noted that the information on this figure is preliminary; the analysis of the test results is continuing at the time of writing of this report.

Response to Text

Even though the results of the day 28 tests were completed for the results of the column leaching tests by the time this text was prepared, the "longer-term behavior" of the milled tailings is discussed as though complete.

Report Text

In contrast to the apparent major-ion chemistry (represented by TDS and SO_4), iron, lead and zinc increase over the first 5 days and then level off or decline slightly with time.

Response to Text

These trends are not precisely the opposite of the trends seen in the other column experiments. They are different, but then they do not contain the sulfides present in the other three tests. The Eh and pH trends are similar to other trends. The TDS decreases rapidly as the reagents are flushed from the sample in the first few days. The TDS is the lowest of all four tests.

Report Text (page 3-19)

Unlike the waste-rock and chat samples, Mn concentration in the milled rock is firmly controlled by solubility of Mn oxides and oxyhydroxides (Hem, 1985, Fig. 16).

Response to Text

The manganese is controlled by the pH. At this pH, the manganese kinetics are very rapid and result in total adsorption as well as precipitation.

Report Text (page 3-19)

The relatively low Eh of the solutions is presumably related to the addition of organic reagents (principally xanthates).

Response to Text

The pH at or near 10 may also be reflecting an unbuffered reaction between very fine-grained silica and dilute distilled water. This reaction commonly results in an equilibrium pH between 9 and 10.

Report Text (page 3-21 and 3-22)

The low metals values during the batch test and the earliest portions of the column experiments reflect not only the Eh-pH conditions of the bulk solution, but attest to the efficiency of the flotation agents in coating the metal-bearing materials. However, as the soluble additives are flushed from the system (note decreasing TDS), the mineral surfaces become available for reaction. At elevated pH in oxidizing conditions, the solubility of most metals is no longer controlled by largely insoluble carbonates or sulfates, but rather by the much more soluble oxides or hydroxides (e.g., Hem, 1985, Figure 14; Garrels and Christ, 1965, Figure 7.26e), leading to soluble iron, lead and zinc, as seen in the test data.

Of these metals, the most troublesome is lead. The tailings lead values are comparable to the Method 1 values for lead in both waste-rock and chat experiments, and are much higher than the below-detection lead values observed in leachate from the comparable Method 2 experiments.

Note that relatively elevated concentrations of lead are expected whether the tailings were placed above or below the water table. If they were placed above the water table, the limited acidity of the incident infiltration (surely no more and probably less than that of the deionized water used in these experiments; see shallow groundwater data in Spruill, 1984) would not be expected to reduce pH into a range that

would provide solubility control by highly insoluble carbonates or sulfates. Alternatively, if the milled tailings were placed in flooded mine waters (e.g., Samples 30 through 34 in Spruill, 1984), then the long-term behavior might be expected to approach (or even exceed) the EP Toxicity behavior as the alkalinity of the solid phase is overwhelmed and the extremely fine-grained tailings release lead to solution.

Therefore, the column tests data support the conclusion derived from the EPA toxicity tests: the Additional Alternative (in this case, for placement below the water table) is distinctly superior to the EPA milling alternative in terms of leachability of lead, but less effective in terms of zinc.

Response to Text

The discussion on elevated pH in oxidizing conditions is interesting and correct for a closed system but not in the open dynamic surface and near surface conditions at the Galena Subsite. First, the soils pH ranges from 3 to 5 and infiltrating water will eventually decrease the pH. Until that time, the alkaline infiltrating water would react with the acid groundwater tending to neutralization, it thereby precipitates iron oxyhydroxides that can sorb additional dissolved metals (lead, cadmium, and zinc) thereby decreasing dissolved metals.

Notice that both dissolved cadmium and zinc have their lowest concentration in this column test compared to the other column tests, which is contrary to the concern about alkaline transport of metals.

In the second paragraph, the lead concentration is lower than either mine waste rock leach and should have been lower than the chat leach if the milled tailing product really had the amount of lead removed it should have had. Furthermore, by day 28, the lead concentration was below detection.

For the third paragraph, see the above discussion. Not only do the soils create an acid infiltrating recharge water, but the groundwater in the mine workings would benefit from the alkaline leachate given off by the milled tailing.

If the metals have been removed from the milled tailings before emplacement in either saturated or unsaturated conditions, it is difficult to understand where the increased metal load would come from. Removal of the metals-bearing mineralogy is source control and a permanent solution leading to reduction (to elimination) of the mobility, volume, and toxicity of the source(s).

The Additional Alternative is not "distinctly superior to the EPA milling alternative in terms of leachability of lead" or other metals, especially when the complete test results are considered.

3.4 SUMMARY OF RESULTS

3.4 SUMMARY OF RESULTS, PAGE 3-22

Report Text

The rate at which leaching occurs does not appear to be particularly sensitive to the concentration of metals in the solid phase in the material being leached.

Response to Text

If the data are plotted, the leaching rate is more proportional to the concentration of metal in the sample than to the grain size alone, but the unknown mineralogy will strongly affect the rate. Given a sample with an equal concentration of a metal (one mineralogy) in all grain sizes, the leaching rate is proportional to the grain size, with finer grain size having the higher leaching rate because

of its larger surface area on a volume or weight basis. It is difficult to understand the statement that the rate of leaching is not dependent on a solid phase if the solid phases (mineralogy) are not known.

Report Text

The rate at which leaching occurs reduces rapidly with time, apparently due to armouring of fresh oxidation faces by the products of oxidation. The corrollory of this is that the less disturbance that takes place of (particularly) the waste rock, the less the rate of metallic material dissolution.

Response to Text

Much is being ignored in this statement. Please see comments on text from the PRP report Page 2-11. The "armouring" is not demonstrated by mineralogy or anything other than supposition. Adsorption and precipitation could be summarized under this simplistic term. The permanence of the "armoring" depends on the permanence of the environmental conditions.

Report Text (page 3-23)

The rate of leaching is strongly influenced by whether the metal bearing material is above the water table (which enhances leaching) or below the water table (in which case the leaching effectively stops by day 28).

Response to Text

This testing does not evaluate the placement of the materials below the water table in the Galena subsite. It certainly does not indicate that leaching is enhanced in the unsaturated zone over that in the saturated zone. There is no mention of anticipated results of leaching if the groundwaters in the saturated zone were strongly acidic when the materials were backfilled.

Report Text (page 3-23)

After a period of time, it would appear that all the materials are similar in their behavior. Figure 3.6 presents the summation of all the column test results for zinc (upper) and lead ()lower). Despite the considerable scatter in the data, the trend is clear. Essentially all of the leachate metal concentrations appear to be heading rapidly towards a very small value, generally below the MCL for the substances. In simple terms, in the long-term, the waste rock and chat do not appear to provide significant dissolved metal loading to the Short Creek drainage.

Response to Text

The physical meaning of the tests is being ignored. There are differences in the behavior of the metals. For example, the unsaturated mine waste rock tests indicate that the lead, cadmium, and iron achieve reasonably low equilibrium concentrations. Zinc and manganese appear to be still achieving equilibrium with time. This can be compared with the saturated zone water rock placement that will continue to be leached as each rainfall event changes physiochemical conditions (day 37 results). Mine waste rock placement (particularly fine-grained metals bearing rock) should be placed in the unsaturated zone--not in the saturated zone.

Report Text (page 3-23)

Note, again, that the Day 37 data for Method 2 are not comparable to the earlier data, and thus, do not contradict the trend. The increase is only apparent, representing the final fraction that could ultimately be leached, (not the next increment of metal), in all cases a small fraction of the amount already leached by Day 28.

Response to Text

There are no data to support the supposition that the increase in metal release at day 37 represents the "final phase that could ultimately leached." First, the test really does not represent what would happen to waste rock emplaced

in the acid groundwater system and, second, the physiochemical conditions will be changing with the length of time between recharge events and the amount of recharge through the mine workings. These changing conditions will not allow the system to come to any significant equilibrium. Placement of the fine grained waste rock in the acid groundwater of the mine workings will result in the dissolution of additional heavy metals (to their solubility limits in some cases) until the sources are essentially removed by dissolution into the groundwater and surface water systems.

Section 4

WATER QUALITY IMPACTS

4.0 WATER QUALITY IMPACTS

4.0 WATER QUALITY IMPACTS, PAGE 4-1

Report Text

Under current conditions, infiltrating water may move through the waste rocks and chat on the surface, leaching some metals from those materials. Ultimately, those materials may exit the subsurface water system to the surface streams in the area, providing an incremental impact due to chemical loading of those streams. It would be expected that the net impact would be positive, producing a reduction in the mass flux of metals into the stream system.

Response to Text

This sentence appears to say that waste rock and chat (those materials) move through the groundwater system to the surface streams and increase the chemical loading of the streams. It this right? We should remove the fine-grained material if this is the case.

How can the increase in chemical loading of these streams be a net positive impact? This paper needs to demonstrate that putting the waste rock into the acidic groundwater system will result in a reduction in the mass flux of metals into the stream system.

4.1 FLOW CONCEPT, PAGE 4-2

Report Text

The water charges the near-surface aquifer which is made up of Mississippian age limestone and chert, which in the Galena area was also the ore body. This water-bearing unit is about 340 feet thick on average in the subsite area and is of moderate permeability except in the mineralized areas, where the permeability is high.

Response to Text

What is meant by "the mineralized area?" Does this include the mine workings, which are the major conduits of acidic groundwater containing dissolved metals to other parts of the groundwater system and to the surface water system? Please focus on the major problems.

Report Text

As the oxygenated water percolates into the mineralized areas, pH is reduced and contained metals are mobilized.

Response to Text

Dissolved oxygen alone is not necessary for oxidation to take place. There are electrochemical reactions and bacteriological reactions that can mobilize metals in the absence of oxygen.

Report Text

Water from the upper "aquifer" flows into the local streams, carrying with it any heavy metals and ionic species that have been mobilized.

Response to Text

What is meant by "upper aquifer" in this statement? Is this mine workings in the near-surface aquifer (component 2)?

4.2 THE ADDITIONAL ALTERNATIVE

4.2 THE ADDITIONAL ALTERNATIVE,
PAGE 4-2

Report Text

Based on all the testing done in this program, the lowest metallic concentrations are provided by the late-time data

of the Method 2 (submerged or saturated) leaching of waste rock. After only 28 days, it appears that the fresh faces provided by crushing the waste rock have ceased to contribute significant metals to the groundwater, and cadmium, lead, and zinc concentrations are all controlled at very low values (0.1 mg cadmium/l; 0.06 mg lead/l; 0.46 mg zinc/l).

Response to Text

If the Day 37 results are compared, the lowest metallic concentrations result from the milled tailing not the waste rock Method 2. The other parameters (sulfate, TDS, etc.) for the milled tailing are also lower than other leachates.

Report Text (page 4-3)

Given that these MCLs are drinking water standards which are protective of public health, disposal to mine voids below the water table is expected to reduce the metals source-term to levels that would also be protective of health.

Response to Text

Day 37 results do not meet MCLs and, therefore, is not protective of health.

Report Text

Note that even if all of the waste rock and chat cannot be disposed of below the water table in mine voids, the source-term will be reduced over the current source-term by whatever proportion can be disposed in fully saturated conditions, though as discussed below both are considered to be very small increments.

Response to Text

This sentence needs to be broken up and reworded or deleted. How is the current source term reduced? We disagree about putting bulk mine wastes in the saturated zone.

Report Text

As pointed out by Andes, there is more than enough space available in total mine voids (both saturated and

unsaturated) to accommodate the total volume of waste rock plus chat, and detailed engineering design should allow handling procedures that could be expected to reduce the metals concentrations in virtually all leachate to values that approach or meet drinking water standards.

Response to Text

How are more than half the current physical source-term reduced to "deminimus concentrations of potential leachate?" Is the leachate being reduced or is the concentration of metal ions? If metal ions, how are they being reduced? What handling procedures are "expected to reduce the metals concentrations in virtually all leachates?" What does "approach drinking water standards" mean?

4.3 EVALUATION OF FUTURE WATER QUALITY UNDER THE ADDITIONAL ALTERNATIVE, PAGE 4-5

Report Text (page 4-6)

Based on the totality of the leaching data and a review of the mine-water quality, it seems likely that the metals content of leachate from the waste rock and the chat some 75 to 100 years after the end of mining may resemble the as-received batch-test values more closely than the 1-month long Method 1 tests, given the crushing required by those tests. Apparently, then the contribution of metals to the water in the mine pools from surface waste rock and chat is probably a small proportion of the total metals in solution.

Response to Text

More discussion is needed to develop the logic and evidence that the "as received" batch test is more definitive that the Method 1 test to resemble the expected leachate after 75 to 100 years after mining.

Appendix D
Leach Test Results
Mine Waste Fines

-DRAFT-

Appendix D

LEACH TEST RESULTS

MINE WASTE FINES

INTRODUCTION

The PRP group at the Galena subsite have proposed an alternative to remediate the subsite of metals contamination contained in mine waste rock, chat, soils, and sediments. They propose that these mine waste materials would be characterized using a field portable XRF to determine relative metal content. Materials containing high metal content would be collected and then disposed of below the water table in existing mine voids. Material containing low metal content would be collected and then be disposed of above the water table in the mine voids. Material containing "the lowest metal content" would be placed as cover on the other materials. The surface would be shaped and contoured to the extent possible to reduce surface water infiltration.

The proposed PRP alternative was developed based on the results of the column leach tests conducted by the PRPs. The tests modelled surface water infiltration in flowthrough and saturated conditions on waste rock, chat, and some tailings produced to simulate milling of mine waste rock and the subsequent placement of the tailings into the existing mine voids. The water used in the PRP column leach testing was deionized water at a pH of approximately 5.5, which would represent surface water infiltration consisting mostly of rainwater. The purpose of EPA's leach tests was to obtain data on the potential leaching characteristics of waste rock fines in an environment that more closely simulates groundwater conditions at the Galena subsite than the test conditions used by the PRPs. Groundwater in the mine workings

typically have even lower pHs than the deionized water used in the column leach tests conducted by the PRP.

Water samples previously collected from mine shafts and mine voids during the Phase I sampling effort indicated a pH of about 3 to 5, as well as indicating definitive hydraulic gradients. With the PRP alternative, this low pH groundwater will be in contact with the backfilled mine waste materials containing high metal concentrations. Test data described within this technical memorandum indicates that these metals can be mobilized, thus increasing the metal loads in the groundwater and surface water.

TESTING PROCEDURE

To determine what level of increase in metal loading will occur, a series of jar leach tests were conducted using splits of mine waste rock samples collected during the June 1988 field sampling event. The mine waste rock was collected from seven separate piles in six zones. The test mine wastes are rock fines collected from a screening test operation to determine the amount of fine material (less than 1 inch) contained in the large waste rock piles and the approximate metals concentrations in the fine materials. The fine material used in the testing program consists of nominal minus 5/8-inch rock. This finer waste rock frequently contains the highest metal concentrations and these metals can be more rapidly leached than the coarse fraction when exposed to the acid groundwater due to the significant difference in available surface area per unit size.

The samples were dried and each moisture content determined. For this testwork, approximately 1000 grams of the dried

sample was split from the main sample using a Jones splitter to maintain representativeness.

To better determine size distribution in the feed for the tests, a portion of each sample was separated using a riffle splitter and wet screened at 80 mesh (180um) and 400 mesh (38um). These size fractions were chosen because they had been determined to represent the more chemically reactive portions of the material. Each size fraction (and the water used in screening, in case any metals had been solubilized) was analyzed for lead and zinc. The water used for screening was filtered through 0.45 micron filter paper to remove solids prior to analyzing for lead and zinc.

Table 1 shows the lead and zinc analyses by size fraction prior to the leaching tests. The results clearly show the metals concentration increases as the sample size distribution decreases.

Distilled water, with the pH adjusted to 4.5 using hydrochloric acid, was added in an equal weight to the split sample, making a mixture 50/50 in weight percent solids. HCI was used so that any sulfate produced from sulfide oxidation could be determined. Each sample mixture was weighed before the tests began, and at each sampling interval, to determine weight loss by evaporation.

The tests were conducted for 72 hours, with stirring and subsequent sampling intervals at 24, 48, and 72 hours. The samples were weighed, stirred for 1 minute, letting the solids settle prior to an aliquot being taken for analysis. Each aliquot was filtered at 0.45 micron to remove any suspended solids before analysis. The pH was not readjusted between each sampling. Each aliquot was analyzed for dissolved Zn, Pb, SO4, Mn, Fe, and Cd.

Table 1 shows the results from the jar tests. Results have been adjusted to reflect constant solution volume (to compensate for evaporation from the containers over the 72-hour test period).

RESULTS

The pH decreased during the first 24 hours of all seven tests. The rapid decrease in pH, from an initial pH of 4.5 to as low as 3.2 indicates the presence of a mineral phase or compound that controls the pH to less than 4.5. The range of measured pH is similar to pH measurements of groundwater in the mine shafts at the Galena subsite. This pH can actively leach metals from the waste rock fines.

Sulfate is a particularly important measurement in these tests. Sulfate forms by the oxidation and/or acid attack on metal-bearing sulfides in an oxidizing near-surface condition like the proposed emplacement of the waste rock into the groundwater system of the mined area. Lead, zinc, and cadmium all occur in and are dissolved from sulfide minerals present in the waste rock. Sulfate, with one exception, increases continuously in the jar tests through the 72-hour test period. This indicates that oxidation is continuing with the release of metals partially to solution and partially to the production of new intermediate mineralogic phases. The one exception is probably an analytical error because once sulfate is put into solution only a few things can reduce it. In reducing conditions, bacteria can convert it to hydrogen sulfide gas (rotten egg gas) but no odor was detected. High calcium concentrations (less than 2,000 ppm) could result in the precipitation of the mineral gypsum (calcium sulfate) but these sulfate concentrations are too low. An intermediate oxidation mineral product called

jarosite begins to form at these pH levels, but if this mineral controlled the sulfate, it would be evident in all the samples.

Dissolved zinc in the leach solutions reached concentrations in excess of 200 ppm within the first 24 hours of leaching for Samples P5-2, P8-1, P1-1, and P3-2. All zinc concentrations increased throughout the tests, some asymptotically approaching 300 ppm after 72 hours. Zinc concentrations in solution generally correlate well with the zinc concentrations in the head samples. Zinc, like sulfate, tends to remain in solution; therefore, the zinc concentration is a good indicator of metal dissolution from the mine waste rock.

Dissolved cadmium reached concentration in excess of 1 ppm in the leach solutions within the first 24 hours. Cadmium concentrations correlate well with zinc in the leach solutions, as would be expected from its association in the mineral sphalerite (zinc sulfide).

Lead and cadmium tend to achieve maximum dissolved concentrations within the first 24 hours. The mine wastes may contain mineralogical phases that release lead and cadmium to solution easily, but also only to some limiting concentration. The phases are not the same for both elements, but the near stability of concentrations above MCL's is of concern. As these test results indicate, if the finer-grained mine waste materials containing relatively high metal concentrations were to be placed in the mine voids and were able to contact the oxygenated acidic groundwater within these voids, additional dissolution of the lead, zinc, and cadmium would occur over what currently exists in the groundwater. Thus, fine-grained, high metal concentration waste

rock placed in the shallow groundwater system (if acidic) will further degrade water quality.

Work scope and other analytical results are shown following Table 1.

TABLE 1.

February 8, 1989
DEN67900.PM

CONSTANT
VOLUME ADJUSTED
SUMMARY RESULTS
JAR LEACHING TESTS
CHEROKEE COUNTY SITE

SAMPLE NO.	LEACH TIME HOURS	PPM - dissolved - Zn	Pb	SO4	Cd	ADJUSTMENT FACTOR	pH	Head Analysis,PPM Pb	Zn	SCREEN RESULTS Size	Pb ppm	Zn ppm
P4-2	24	24.58	3.50	1,092.36	0.18	0.9103	3.44	2350	1500	+80	2450	3300
	48	27.87	4.44	582.05	0.21	0.8819	3.56			-80+400	2050	1050
	72	30.77	4.39	563.27	0.22	0.8407	3.57			-400	6950	700
P5-2	24	195.09	0.22	1,386.90	0.61	0.9246	4.06	320	9600	+80	260	6750
	48	231.56	0.34	1,534.73	0.68	0.8975	4.21			-80+400	415	5550
	72	266.12	0.33	1,737.07	0.75	0.8557	4.3			-400	650	1600
P8-1	24	239.50	0.11	946.87	1.30	0.9283	3.59	860	15500	+80	320	14000
	48	261.49	0.14	1,000.89	1.44	0.9017	3.6			-80+400	780	6600
	72	278.01	0.36	1,079.25	1.54	0.8634	3.59			-400	3050	1700
P1-1	24	202.05	2.98	927.58	1.43	0.9184	3.52	5950	12500	+80	3500	19000
	48	220.47	3.00	937.23	1.57	0.8926	3.54			-80+400	6000	7150
	72	247.50	2.79	1,027.68	1.64	0.8564	3.55			-400	15000	4600
P3-2	24	268.49	0.29	1,420.75	1.52	1.1187	3.2	600	5150	+80	275	4950
	48	279.36	0.28	1,537.58	1.60	1.0828	3.2			-80+400	850	3250
	72	294.95	0.21	1,557.26	1.57	1.0313	3.19			-400	3450	1550
P4-1	24	17.20	4.60	609.01	0.13	0.8956	3.52	2900	1950	+80	1150	2700
	48	19.59	5.18	572.22	0.16	0.867	3.52			-80+400	3100	1100
	72	22.51	5.12	583.52	0.10	0.8336	3.52			-400	6800	700
P7-1	24	34.69	5.14	612.75	0.25	0.9011	3.78	7750	8350	+80	16500	10500
	48	40.19	5.28	784.53	0.28	0.8717	3.8			-80+400	13500	7150
	72	45.34	5.25	854.86	0.29	0.8381	3.83			-400	13500	2900

P4-2 means the second (2) mine waste rock pile sample collected from zone four (4)

Conditions

- o 50/50 weight percent solids
- o Mine waste rock, < 5/8 inch
- o Solution, distilled water, pH adjusted to 4.5 using HCl

SCOPE OF WORK

MINE WASTE SCREENING AND LEACHING TESTS

CHEROKEE COUNTY SITE

DECEMBER 1, 1988

Upon receiving the mine waste samples, please record the total as received weight and sample number. Dry the samples at approximately 105°C and record the final weight.

LEACH TESTS

From each dried sample, split and weigh using a riffle-type splitter, from the total, approximately 1,000 grams of materials. Place this material into a 2- or 4-liter beaker. Add to the beaker an equal weight of distilled water (equal to the mine waste material weight) in which the pH has been preadjusted to 4.5 using HCl. After the level has adjusted, mark the level of water in the beaker.

Allow the mixture to sit for 72 hours. At the end of each 24-hour period, note the water level and adjust the previous sampling level, then stir the mixture manually for 1 minute. Allow the solids to settle, then take an aliquot for analysis. Note the size of the aliquot taken along with the solution pH and conductivity. Don't re-adjust the pH during the test. Filter the aliquot through a 0.45-micron filter before analysis. Analyze each aliquot for the following parameters: Zn, Pb, SO4, Mn, Fe, and Cd.

SCREENING

Split the remainder of the sample into two equal parts using a riffle splitter. Save one part for future work, and wet screen the other at 80 mesh and 400 mesh. Save, dry, and weigh each size fraction, and also save the water used for screening. Analyze each size fraction for total lead (Pb) and zinc (Zn). Filter the water through a 0.45-micron filter and assay it for total Pb.

Report all results in the PPM.

SKYLINE LABS, INC.
SPECIALISTS IN EXPLORATION GEOCHEMISTRY
12090 WEST 50TH PLACE • WHEAT RIDGE, COLORADO 80033 • TEL.: (303) 424-7718

REPORT OF ANALYSIS

JOB NO. DKV 753
December 7, 1988
P.O. NO. 0889
PROJECT NO. 6905-03

Hazen Research, Inc.
Attn: Ron Rak
4601 Indiana Street
Golden, Colorado 80403

Analysis of 28 Pulps

ITEM	SAMPLE NUMBER	Pb (ppm)	Zn (ppm)
1	P1-1 +80	3500.	19000.
2	P1-1 -80+400	6000.	7150.
3	P1-1 -400	15000.	4600.
4	P3-2 +80	275.	4950.
5	P3-2 -80+400	850.	3250.
6	P3-2 -400	3450.	1550.
7	P4-1 +80	1150.	2700.
8	P4-1 -80+400	3100.	1100.
9	P4-1 -400	6800.	700.
10	P4-2 +80	2450.	3300.
11	P4-2 -80+400	2050.	1050.
12	P4-2 -400	6950.	700.
13	P5-2 +80	260.	6750.
14	P5-2 -80+400	415.	5550.
15	P5-2 -400	650.	1600.
16	P7-1 +80	16500.	10500.
17	P7-1 -80+400	13500.	7150.
18	P7-1 -400	13500.	2900.
19	P8-1 +80	320.	14000.
20	P8-1 -80+400	780.	6600.

ITEM	SAMPLE NUMBER	Pb (ppm)	Zn (ppm)
21	P8-1 -400	3050.	1700.
22	P1-1 HEAD A	5950.	12500.
23	P3-2 HEAD A	600.	5150.
24	P4-1 HEAD A	2900.	1950.
25	P4-2 HEAD A	2350.	1500.
26	P5-2 HEAD A	320.	9600.
27	P7-1 HEAD A	7750.	8350.
28	P8-1 HEAD A	860.	15500.

for Edwin V. Post

Gordon H. VanSickle
Manager

SKYLINE LABS, INC.
SPECIALISTS IN EXPLORATION GEOCHEMISTRY
12090 WEST 50TH PLACE • WHEAT RIDGE, COLORADO 80033 • TEL.: (303) 424-7718

REPORT OF ANALYSIS

JOB NO. DKV 752
December 7, 1988
P.O. NO. 0889
PROJECT NO. 6905-03

Hazen Research, Inc.
Attn: Ron Rak
4601 Indiana Street
Golden, Colorado 80403

Analysis of 7 Screening Water Samples

ITEM	SAMPLE NO.	Pb (mg/L)	Zn (mg/L)
1	P1-1	.7	65.0
2	P3-2	.1	55.0
3	P4-1	.2	4.9
4	P4-2	.3	6.9
5	P5-2	.4	69.0
6	P7-1	.8	16.0
7	P8-1	.2	59.0

Gordon H. VanSickle
Manager

Summary Results
Leaching Tests
Cherokee County Site Samples

Project 6905-03
Date: 12/6/88

ile HRI No.	Sample	Leach Time, hours	Zn	Pb	SO4	Mn	Fe	Cd	Con 1/	pH	Aliq. Liter	Weights, g 2/ Slurry	Solids	Sol'n
			Assays, ppm											
40847-1	P4-2	24	27.0	3.84	1,200	0.30	0.34	0.20	270	3.44	0.075	1,907.5	997.2	910.3
		48	31.6	5.04	660	0.30	0.26	0.24	309	3.56	0.075	1,804.1	997.2	806.9
		72	36.6	5.22	670	0.42	0.32	0.26	324	3.57	0.075	1,687.9	997.2	690.7
0847-2	P5-2	24	211	0.24	1,500	0.50	< 0.10	0.66	1200	4.06	0.075	1,920.2	995.6	924.6
		48	258	0.38	1,710	0.60	< 0.10	0.76	1380	4.21	0.075	1,818.1	995.6	822.5
		72	311	0.38	2,030	0.66	0.12	0.88	1020	4.30	0.075	1,701.3	995.6	705.7
0847-3	P8-1	24	258	0.12	1,020	0.40	0.26	1.40	721	3.59	0.075	1,921.0	992.8	928.3
		48	290	0.16	1,110	0.46	0.28	1.60	788	3.60	0.075	1,819.4	992.8	826.7
		72	322	0.42	1,250	0.50	0.36	1.78	839	3.59	0.075	1,706.1	992.8	713.4
40847-4	P1-1	24	220	3.24	1,010	0.20	0.80	1.56	700	3.52	0.075	1,914.9	996.5	918.4
		48	247	3.36	1,050	0.22	1.08	1.76	762	3.54	0.075	1,814.1	996.5	817.6
		72	289	3.26	1,200	0.20	1.24	1.92	824	3.55	0.075	1,702.9	996.5	706.4
40847-5	P3-2	24	240	0.26	1,270	0.56	42.0	1.36	1160	3.20	0.075	2,115.2	996.5	1,118.7
		48	258	0.26	1,420	0.62	47.8	1.48	1190	3.20	0.075	2,004.3	996.5	1,007.8
		72	286	0.20	1,510	0.64	50.0	1.52	1320	3.19	0.075	1,877.8	996.5	881.3
[illegible]7-6	P4-1	24	19.2	5.14	680	0.52	0.22	0.14	258	3.52	0.075	1,892.2	996.6	895.6
		48	22.6	5.38	660	0.62	0.20	0.18	278	3.52	0.075	1,788.6	996.6	792.0
		72	27.0	6.14	700	0.74	0.30	0.12	288	3.52	0.075	1,680.2	996.6	683.6
347-7	P7-1	24	38.5	5.70	680	0.16	0.10	0.28	227	3.78	0.075	1,899.4	998.3	901.1
		48	46.1	6.06	900	0.20	< 0.10	0.32	250	3.80	0.075	1,795.0	998.3	796.7
		72	54.1	6.27	1,020	0.22	< 0.10	0.35	268	3.83	0.075	1,686.4	998.3	688.1

1 Conductivity expressed as umhos / cm
2 Weight prior to sampling. Solids weights are final dried residue weights.

HAZEN RESEARCH, INC.

Sample HRI No.	Sample	Solids Data: Initial Weight, g	Percent Moisture	Calc'd Initial Dry 3/ Weight, g	Residue Weight, g	Final Filtrate Volume, L
40847-1	P4-2	1000	10.17	898.3	997.20	0.463
40847-2	P5-2	1000	0	1000	995.63	0.513
40847-3	P8-1	1000	21.43	785.7	992.75	0.512
40847-4	P1-1	1000	5.62	943.8	996.53	0.526
40847-5	P3-2	1000	7.41	925.9	996.46	0.740
40847-6	P4-1	1000	8.33	916.7	996.56	0.470
40847-7	P7-1	1000	4.92	950.8	998.29	0.490

3/ Calculated using 1000g initial weight and percent moisture data.

PRELIMINARY DATA
Hazen Research Inc.

Appendix E
Mass Load Analysis
Proposed Remedial Alternatives

DRAFT

Appendix E

MASS LOAD ANALYSIS--PROPOSED REMEDIAL ALTERNATIVES AT THE GALENA/CHEROKEE SUBSITE

BACKGROUND

Four remedial alternatives, in addition to the no-action alternative required by the NCP, are now being considered for the Galena subsite. Based on the results of the RI and subsequent Surface Water/Groundwater FS, it was determined that water quality in the Short Creek and its two principal tributaries, Owl Branch and "Tributary A" through Hells Half Acre, is dramatically impacted by runoff through mine wastes on the surface and through the subsurface mined workings. A water-quality model was developed in the FS that determined that approximately 26 percent of the mass loads of zinc, cadmium, and sulfate (on Short Creek below the confluence with Owl Branch) were due to the mine wastes (waste rock and chat) on the surface of the subsite and about 74 percent was due to the shallow groundwater system that had been substantially altered by the mining of the lead and zinc ores principally intercepted by Short Creek. The results of this model will be used as a basis of the no-action alternative to compare impacts on the mass load as a result of the various remedies under consideration.

In the following section, the alternatives are discussed in sufficient detail to support the reasoning used to estimate subsequent mass load changes as a result of the remedy. The alternatives are presented in more detail in the FS Technical Supplement. The estimates developed are order of magnitude and are suitable only for a relative comparison between alternatives.

of the alternative's effectiveness in improving water quality. Values used in estimating geochemical reactivity of various mine wastes are from work conducted by both EPA and the PRPs during the course of the Galena site work.

ALTERNATIVE 1
BASE CASE, NO-ACTION

The chemical parameters selected for consideration are sulfate, dissolved cadmium, and dissolved zinc, and can be considered as surrogates for other contaminants present in the surface and groundwater systems at the site. Once released, sulfate moves through the hydrological systems without major chemical reactions (precipitation or sorption losses). This behavior results in a conservative estimate for the system since mass is preserved. Sulfate serves as a good indicator of acid mine drainage (AMD) leachate produced by the oxidation of sulfides provided that metals in solution are less than at saturation and pH is above 3. Cadmium and zinc are also mobile and represent public health and environmental considerations analogous to unmobile constituents such as lead, which is typically attenuated in the sulfate-rich geohydrologic system. As sources of this AMD are removed or altered, subsequent mass loads of sulfate, cadmium, and zinc compared to the base case will change with a resulting improvement or excavation of the considered constituent. For the no-action alternative, the combined mass loads for the surrogates from the Owl Branch and Tributary A catchments, as determined in the OUFS, are as follows:

- Sulfate--4,190 lb/day
- Zinc--600 lb/day
- Cadmium--3.6 lb/day

ALTERNATIVE 2

MINE AND MILL ALL MINE WASTES

This alternative would remove all the waste rock and chat. The chat would be wet screened and then the chat fines and waste rock will be milled to recover lead and zinc concentrates. The mill tailings, which can pass an EP Toxicity test, would be backfilled to the available mine voids. Approximately 660,000 tons of waste rock and 1,000,000 tons of chat will be processed. As in all remedial alternatives, surface water channelization would be accomplished through Owl Branch and Tributary A-Hells Half Acre. Certain surface recontouring and revegetation would be conducted in conjunction with removal of the waste rock and defective deep wells would repaired.

For modeling considerations, the following two major surface water issues are involved:

1. Surface water runoff infiltrating and percolating through the surface mine wastes

2. Surface water recharge to the shallow groundwater system, primarily occurring as stream capture in mine voids

Rechannelization through the Owl Branch and Tributary A-Hells Half Acre, will divert an estimated 0.2 cfs and 0.8 cfs, respectively, from recharge to the groundwater system (EPA, Phase 1 RI, Sprull 1984). Based on the concentrations of existing groundwater quality estimated to be caused by recharge, the following mass load reductions due to rechannelization are estimated as the following:

The net reduction due to the removal of all mine waste rock and chat is then estimated to be as follows:

SO_4 = 582 lb/day
Zn = 71 lb/day
Cd = 0.29 lb/day

The new mass loads for the system reduced by the combined implementation of rechannelization and removal and processing of surface mine wastes is as follows:

lb/day SO_4: 4,190 - 576 - 582 = 3,032 lb/day; 27.7% reduction
lb/day Zn: 600 - 146 - 71 = 383 lb/day; 36.2% reduction
lb/day Cd: 3.6 - 0.85 - 0.29 = 2.46 lb/day; 31.7% reduction

ALTERNATIVE 3
MINE AND MILL ALL MINE WASTE
ROCK AND HALF OF THE CHAT

This alternative is the same as Alternative 2 except that about only one half the chat would be milled with the waste rock, assuming the other half passes the action level for lead. All other elements of the remedy such as channelization, deep-well remediation, and recontouring would be implemented. Since a portion of the chat would remain on the surface and still be subject to some leaching, the net reduction in mass load will be less than for Alternative 2 where all the chat and waste rock were removed and milled. To estimate the difference in mass load, it was assumed that leaching rate differences between the two types of wastes should be a function of the zinc concentrations and volumes of each. Zinc concentrations of mine waste rock was determined in the RI. The weighted average by volume of the waste rock was 5,530 ppm. Concentrations of zinc in chat from

more recent work (Appendix A) was averaged at 4,520 ppm. Based on about 660,000 tons of waste rock and 1,000,000 tons of chat, a weight-adjusted percentage of zinc is 44 percent waste rock, 56 percent chat. Assuming the zinc concentration is uniform over the chat volume, a 72 percent decrease [44+1/2(56)] in mass load would be expected for this alternative compared to Alternative 2, regarding removal of waste rock. Therefore, the difference would be as follows:

SO_4: 0.72 x 627 lb/day = 451 lb/day
Zn: 0.72 x 74 lb/day = 53 lb/day
Cd: 0.72 x 0.29 lb/day = 0.21 lb/day

The new mass loads for the system, reduced by the combined implementation of channelization and removal and processing of <u>only</u> mine waste rock, would be:

lb/day SO_4: 4,190 - 576 - 451 = 3,163 lb/day; 24.5% reduction
lb/day Zn: 600 - 146 - 53 = 401 lb/day; 33.2% reduction
lb/day Cd: 3.6 - 0.85 - 0.21 = 2.54 lb/day; 29.4% reduction

ALTERNATIVE 4
MINE AND DISPOSE OF ALL MINE WASTES IN AN ONSITE CONTAINMENT FACILITY

This alternative involves the removal of all mine waste rock and chat from the subsite. Instead of milling it to recover metals, it is placed in an onsite hazardous waste containment facility. Other remedy components such as channelization, recontouring, and deep well repair will be implemented.

From a mass load perspective, the results will be identical to Alternative 2. The loads will be as follows:

lb/day SO_4:	3,032 lb/day; 27.7% reduction
lb/day Zn:	383 lb/day; 36.2% reduction
lb/day Cd:	2.46 lb/day; 31.7% reduction

ALTERNATIVE 5
GEOCHEMICALLY CHARACTERIZE WASTE,
SEGREGATE BY PARTICLE SIZE,
SELECTIVELY BACKFILL, AND RECONTOUR

There are several ways to characterize mine wastes including metal concentration by waste type or size distribution. Say the XRF is used to characterize the mine wastes into high, low, and lowest metals concentrations. The highest concentrations will be backfilled into mine voids below the water table; the low-grade wastes will be backfilled above the high-grade wastes above the water table; the lowest concentrations will be used for cover material and recontoured to minimize infiltration. Other components including rechannelizing, recontouring, and deep well remediation will also be accomplished.

Assuming the surface materials meet action levels, the model assumes that the impact of surface water diversion and surface water cleanup will be similar to Alternative 2. There will be a net load reduction for these components as follows:

lb/day SO_4:	576 + 582 =	1,158 lb/day
lb/day Zn:	146 + 71 =	217 lb/day
lb/day Cd:	0.85 + .29 =	1.14 lb/day

However, placement of high metal containing materials in the acidic, oxygenated groundwater system of the mine voids could increase metal concentrations to thermodynamically saturated conditions. EPA leach tests on high-metal-concentration waste rock fines indicated there is significant parameter concentration differences between the relatively calcareous-rich Zone 4 (Blue Hole area) and the highly siliceous Zones 1, 5, and 7 (near Shoal Creek, Short Creek, and Hells Half acre, respectively). The siliceous material released about twice the amounts of zinc and sulfate as the calcareous material. Both waste types, however, released much higher concentrations of the metals than estimated for the groundwater in the mine voids as noted in the OUFS. The concentrations used for this modelling are the average concentrations of the minimum and maximums for each parameter. The Owl Branch contribution is estimated from the calcareous material leach tests typical of the Blue Hole area and the Tributary A zone, and were calculated in a similar manner for the siliceous materials. The concentration ranges and averages are shown as follows:

For carbonate-rich waste rock (per acid leach test):

SO_4: 550 - 580 mg/l - avg = 565 mg/l
Zn: 20 - 30 mg/l - avg = 25 mg/l
Cd: 0.1 - 0.2 mg/l - avg = 0.15 mg/l

For siliceous-rich waste rock (per acid leach test):

SO_4: 850 - 1,740 mg/l - avg = 1,295 mg/l
Zn: 45 - 270 mg/l - avg = 158 mg/l
Cd: 0.3 - 1/6 mg/l - avg = .95 mg/l

Per the OUFS model, the groundwater flow component in Owl Branch was estimated at 1.7 cfs. By removing the estimated

0.2 cfs of stream capture to be reduced by channelization and the 1.38 cfs from the undisturbed area, a groundwater flow component through these backfilled mine wastes of 0.12 cfs remains.

For Hells Half Acre, a 1.77 cfs flow rate was estimated minus the 0.8 cfs stream capture components (and 0.79 cfs from the undisturbed area) leaves a groundwater flow here of 0.17 cfs. If all buried, metal-rich mine wastes come in contact with these flows, the mass load increase to the system would be as follows:

For Owl Branch:

0.12 cfs x 565 mg/l SO_4 = 365 lb/day SO_4
0.12 cfs x 25 mg/l Zn = 16 lb/day Zn
0.12 cfs x 0.15 mg/l Cd = 0.1 lb/day Cd

For Hells Half Acre:

0.17 cfs x 1,295 mg/l SO_4 = 1,184 lb/day SO_4
0.17 cfs x 158 mg/l Zn = 145 lb/day Zn
0.17 cfs x 0.95 mg/l Cd = 0.87 lb/day Cd

Summing the net mass load change compared to base case results in the following:

SO_4: 4,190 lb/day - 1,158 + (365 + 1,184)
= 4,581 lb/day; a 9.3% net increase

Zn: 600 lb/day - 217 lb/day + (16 + 145)
= 544 lb/day; a 9.3% net decrease

Cd: 3.6 lb/day - 1.14 lb/day + (0.1 + 0.87)
= 3.43 lb/day; a 4.7% net decrease

Since overall mass load reductions were rather insignificant, another variation for backfilling would be to use the XRF to categorize the chat for lead and zinc and screen the waste rock into high and low geochemically active categories. Chat with higher lead and lower zinc concentrations will be mixed with coarse mine waste (+2 inch) that has been mined and screened. This material, which has low geochemical reactivity, will be placed into saturated mine voids to the high water table elevation. The geochemically more active and finer (-2 inch) mine waste material will be mixed with high zinc-containing chat and placed in the voids on top of the coarser material. The final cover will consist of chat meeting the action level for lead, which will be contoured and revegetated as required. Other remedy components, including surface water channelization, other recontouring, and deep well remediation, will also be implemented.

As in all the alternatives, there will be a net benefit and mass load reduction from surface water diversion. For the groundwater component, however, the factors that will govern mobilization of metals and sulfate from these coarser, less-reactive materials will again be permeability, flow rate, grain size, and mineralogy. There are no data available on the solubility difference between the mine waste grain size fractions for mine wastes at the Galena subsite, but the kinetic rate of metals released should be proportional to the square of the diameter. Since coarse-size mine waste (+2 inch) with low metal-bearing chat will be placed below the water table, most of the metal-bearing mineralogy will be encapsulated in coarser material and will leach only very slowly as compared to the metal-rich materials previously considered. Since this material may average 4 inches in average diameter, whereas the metal-rich bearing material may be 1/4-inch-average diameter, a significantly lower rate of dissolution would occur with coarse-grained waste at an equivalent available mineral content. To some degree, at

lower flow-through rates, mineral solubilities (Ksp) may have a greater influence than the kinetics affected by particle size.

In any case, it is conservatively estimated that the groundwater mass loads previously estimated would decrease by a factor of four. On that basis, the net mass load change for this alternative would be the following:

SO_4: 4,190 lb/day - 1,158 + 387 lb/day
= 3,419 lb/day; a 18.4% decrease

Zn: 600 lb/day - 217 lb/day + 40 lb/day
= 423 lb/day; a 29.5% decrease

Cd: 3.6 lb/day - 1.14 lb/day + 0.22 lb/day
= 2.68 lb/day; a 25.6% decrease

It would be worthwhile to perform pilot-scale flow-through leach tests to validate mass load reductions for this alternative.

SUMMARY BY ALTERNATIVE--MASS LOADS

Alternative Number	SO_4 Decrease (lb/day)	SO_4 Decrease (%)	Zn Decrease (lb/day)	Zn Decrease (%)	Cd Decrease (lb/day)	Cd Decrease (%)
1. No action	4,190	--	600	--	3.6	--
2. Mine/mill all waste	3,032	27.6	383	36.2	2.46	31.7
3. Mine/mill waste rock and 50 percent chat	3,163	24.5	401	33.2	2.54	29.4
4. Mine/dispose	3,032	27.6	383	36.2	2.46	31.7
5. Geochemically select/ backfill	3,419	18.4	423	29.5	2.68	25.6

Appendix F
Alternative Cost Estimates

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APPENDIX F.1
GENERAL ASSUMPTIONS/QUALIFICATIONS

These cost estimates have been prepared for guidance in project evaluation and implementation from the information available at the time of the estimate. The final costs of the project selected for implementation would depend on actual labor and material costs, actual site conditions, productivity, competitive market conditions, final project scope and schedule, the firm selected for final engineering design, and other variable factors. As a result, the final project costs could vary substantially from the estimate presented here. Because of these factors, funding needs must be carefully reviewed prior to making specific financial decisions or establishing final budgets. Presented below are specific assumptions and qualifications developed during alternative development and cost estimating.

Waste rock, as used in this report, is used to define the mine development rock (locally called bullrock). The assumed volume of waste rock includes material that have mass wasted from the pile (about 10 percent). Total volume = 547,605 cubic yards. Assuming 1.21 tons/cubic yard density, total tonnage = 662,602 tons.

Chat is defined as the pea gravel sized (about 1/4 X 0 inches) mine wastes that remain from early milling processes. The PRPs estimated the total chat volume to be 749,854 cubic yards. Assuming 1.35 tons/cubic yard density, total tonnage = 1,012,303 tons.

Sizing of treatment process systems and associated costs are dependent on waste types, quantities, and locations. All mine waste volume estimates and calculations were based on information provided by the PRPs since their mine waste volume data base contains more detail than EPA's. If actual waste types, tonnages, and locations are substantially different than currently estimated, major changes in costs

are possible.

For all alternatives, it has been assumed that title to mine wastes (both waste rock and chat) and access for removal, transport, and/or disposal can be obtained by EPA. No costs were allocated for these items.

The cost estimates assume that up to 600 acres of disturbed area will be revegetated following implementation of the remedy. EPA Region VII has assumed that revegetation can be completed at a unit cost of $1,000 per acre. This would include top soil additions, soil amendments, and re-seeding where appropriate.

Mine waste tonnages and the remedy areas consist of those within the eight defined EPA zones. Other areas of the subsite were excluded from the study since these areas had insignificant disturbances compared to the disturbances in the EPA defined zones.

All remedial construction has been presumed to be conducted at Health and Safety Level D.

All costs were calculated in current 1989 dollars. Federal pricing guidelines for contracts were considered.

APPENDIX F.2
REMEDIAL ACTION ASSUMPTIONS

Table F-1 presents a list of the remedial actions that comprise various elements of the remedial alternatives selected for detailed analysis. These remedial actions were used in a computer spreadsheet format to develop a cost estimate for each remedial alternative (see Appendix F.3). For each action, assumptions and qualifications are presented below with the unit cost.

1.0 ACTIONS TO SUPPORT MILLING/FLOTATION (ALTERNATIVES 2 AND 3)

1.1 Mining

This action includes mining and loading the waste rock and/or chat for transport to the milling/flotation plant.

Waste Rock. Unit costs for this operation were obtained from Means Heavy Construction Data-1988 for three scenarios - easy, difficult, and very difficult mining/loading. These unit costs range from $1.46 to $3.26 per ton. Easy mining assumes a track mounted bucket loader can be used and more difficult mining assumes a backhoe and a loader are required. Very difficult mining assumes a dragline and loader would be required. Engineering judgement and knowledge of the site was used to estimate a weighted unit cost of $2.22 per ton.

Chat. Unit costs for this operation were obtained from Means Heavy Construction Data-1988. It is assumed that all chat can be mined relatively easily and loaded with large wheeled loaders at a unit cost of $1.25 per ton.

Remedial Action Components	Alternative No. 1 No Action	Alternative No. 2 Mine And Mill All Mine Wastes	Alternative No. 3 Mine And Mill All Waste Rock And Half Of the Chat	Alternative No. 4 Mine And Dispose Of All Mine Wastes In Onsite Containment Facility	Alternative No. 5 Geochemically Characterize Wastes, Segregate, Selectively Backfill, And Recontour
1.0 Actions To Support Milling/Flotation					
1.1 Mining		X	X		
1.2 Haul To Mill		X	X		
2.0 Actions To Support Mine Waste Backfill					
2.1 Mine/Dispose (IntraZone Materials)					X
2.2 Mine/Dispose (Interzone Materials)					X
2.3 Chat Cover Material					X
3.0 Actions To Support Onsite Containment					
3.1 Mining				X	
3.2 Haul To Containment Unit				X	
3.3 Containment Unit Filling				X	
4.0 Misc. Additional Site Work		X	X	X	X
5.0 Facility Capital Costs					
5.1 Mine Waste Crushing Plant		X	X		
5.2 Chat Wet Screening Plant		X	X		
5.3 Mill/Flotation Plant		X	X		
5.4 Mine Waste Screening Plant					X
5.5 Containment Unit				X	
6.0 Facility Operating Costs					
6.1 Mine Waste Crushing Plant		X	X		
6.2 Chat Wet Screening Plant		X	X		
6.3 Mill/Flotation Plant		X	X		
6.4 Mine Waste Screening Plant					X
7.0 Support Fieldwork Costs					
7.1 Chat Characterization			X		X
7.2 Cut/Fill Design					
8.0 Revegetation Costs		X	X	X	X
9.0 Rechannelization					
9.1 Tributary A		X	X	X	X
9.2 Owl Branch		X	X	X	X
10.0 Remediate Deep Wells					
10.1 Rehabilitation		X	X	X	X
10.2 Plugging		X	X	X	X
11.0 Monitor Water Quality					
11.1 Well Construction		X	X	X	X
11.2 Monitoring Program		X	X	X	X

Table F-1
Remedial Action Components
For Selected Remedial Alternatives

1.2 Haul to mill

This action assumes 10 cubic yard highway trucks would be used to haul the waste rock and/or chat to the milling/flotation plant. The cost estimates assumes that the plant would be located adjacent to the Hell's Half

Acre area north of Galena. Assumed average unit cost is $0.35 per ton-mile. This cost is based on discussions with local contractors, who indicated a long-term haulage contract would probably run $0.20 to $0.50 per ton-mile.

2.0 ACTIONS TO SUPPORT MINE WASTE BACKFILL (ALTERNATIVE 5)

2.1 Mine/Dispose (Intrazone material)

Waste Rock. This action would include mining the waste rock followed by direct backfill disposal, including compaction, in nearby mine voids. Three scenarios (easy, difficult, and very difficult) were developed and unit costs were obtained from Means Heavy Construction Data-1988. These unit costs range from $2.25 to $5.50 per ton. Easy mining/disposal assumes a scraper and dozer combination can be used and more difficult mining/disposal assumes a backhoe would also be required. Very difficult mining assumes a dragline would be required for disposal. Engineering judgement and knowledge of the site was used to estimate a weighted unit cost of $3.65 per ton.

Chat. This action would include mining the chat followed by direct backfill disposal, including compaction, in nearby mine voids. Three scenarios (easy, difficult, and very difficult) were developed and unit costs were obtained from Means Heavy Construction Data-1988. These unit costs range from $1.53 to $3.85 per ton. Easy mining/disposal assumes a scraper (excavation and haul) and dozer combination can be used and more difficult mining/disposal assumes an additional push dozer would also be required. Very difficult mining assumes a dragline would be required for disposal. Engineering judgement and knowledge of the site was used to estimate a weighted unit cost of $2.42 per ton.

2.2 Mine/Dispose (Interzone material)

This action assumes the cut/fill ratio of mine waste to available nearby void space is not in balance and that the material must be hauled between zones to be properly back filled.

Waste Rock. This action includes mining/loading as described for Actions 1.1 and 1.2. Three disposal

scenarios, as described in Action 2.1, were then added to the mining/loading estimates. Unit costs ranged from $2.50 to $6.00 per ton. Engineering judgement and knowledge of the site was used to estimate a weighted unit cost of $4.03 per ton.

Chat. This action includes mining/loading as described for Actions 1.1 and 1.2. Three disposal scenarios, as described in Action 2.1, were then added to the mining/loading estimates. Unit costs ranged from $1.65 to $4.25 per ton. Engineering judgement and knowledge of the site was used to estimate a weighted unit cost of $2.63 per ton.

2.3 Chat cover material

The scope of this action would include using a scraper to excavate and haul chat (with lead concentrations less than the action level) for disposal as a cover for the mine wastes disposed of below grade. A dozer would be used to spread and compact the chat. The unit cost for this action is estimated at $2.00 per ton.

Long-term O&M associated with this action includes maintaining the cover. Erosion and subsidence areas would be annually repaired using a dozer. The estimated annual cost of this action is assumed to be 1% of the initial cost.

3.0 ACTIONS TO SUPPORT ON-SITE CONTAINMENT (ALTERNATIVE 4)

3.1 Mining

See Action 1.1.

3.2 Haul to containment unit

See Action 1.2.

3.3 Containment unit filling

Mine wastes from the disturbed areas would be dumped from the haul trucks in the containment unit. A dozer would be used to spread and compact the mine wastes. The estimated unit cost for this action is $1.34 per cubic yard of mine waste.

4.0 MISCELLANEOUS ADDITIONAL SITE WORK

This action includes primarily dust control and miscellaneous site preparation activities during mining, haulage, and backfilling. Specific activities may include: dust covers for trucks, watering the haul roads, work site debris removal, and access road construction and maintenance. The estimated cost of these activities is estimated to total 5 percent of the total costs of earth work (i.e. Actions 1 through 3) required for each specific remedial alternative.

5.0 FACILITY CAPITAL COSTS

5.1 Mine waste crushing plant

The mine waste crushing plant would process a nominal 100 TPH of 24 by 0 inch waste rock to a product size of 100 percent minus 3/8 inch. The mine waste crushing plant capital cost estimate is based on the installed cost of the list of equipment in Table F-2.

5.2 Chat wet screening plant

The chat screening plant would wet screen chat at 35 mesh at a nominal 100 TPH. The minus 35 mesh fraction would be stored in agitated storage tanks for eventual feed to the milling unit. The larger size fraction would be dewatered and stockpiled. The chat wet screening plant capital cost estimate is based on the installed cost of the list of equipment in Table F-3.

5.3 Mill/flotation plant

The milling/flotation plant would have a design capacity of 700 TPD. The milling/flotation plant capital cost estimate is based on the installed cost of the list of equipment in Table F-4.

5.4 Mine waste screening plant

The mine waste screening plant would physically screen the waste rock (nominally 2 inch) after mining and prior to backfilling. The facility would have a design capacity of 500 TPH. To reduce the waste rock haulage requirements, the screening facility must be portable for relocating to the various waste rock areas. The capital cost estimate is based on the installed cost of the list of equipment in Table F-5.

5.5 Containment unit

The cost estimated for the containment unit assumes that RCRA criteria for a double-lined, double leachate collection unit would apply (Table F-6). To contain all of the waste rock and the chat about 20 acres at a fill depth of 30 feet would be required.

Long-term O&M associated with the containment unit involves maintaining the cover, collection and disposal of leachate, and monitoring. The estimated annual cost of this action is assumed to be 1% of the installation cost.

Table F-2
MINE WASTE CRUSHING PLANT
CAPITAL COST ESTIMATE
(for Alternatives 2 and 3)

ITEM	ESTIMATED INSTALLED COST
Site Preparation	$15,000
Grizzley and Hopper	$5,000
Picking belt	$5,000
Magnet/Trash Bin	$8,000
3 inch Scalping screen with chutes	$25,000
Jaw crusher - 150 hp	$100,000 *
Conveyors (4 at 50 ft, 1 at 100 ft)	$75,000
Short head cone - 100hp	$100,000 *
3/8 in. Screen/hopper/pan/chutes	$25,000
Dust collection systems	$50,000
Structural steel/lift beam/hoist	$35,000
Heat/light/ventilation	$10,000
Fine ore bins	$40,000
Foundation concrete/excavation	$40,000
Power distribution/starters Siding/insulation	$100,000
	$20,000
Wash down water/sumps	$5,000
Subtotal	$658,000
Engineering and construction management-20%	$131,600
TOTAL CAPITAL COST ESTIMATE	$789,600

* Assumes used equipment would be available.

Table F-3
CHAT WET SCREENING PLANT
CAPITAL COST ESTIMATE
(for Alternatives 2 and 3)

ITEM	ESTIMATED INSTALLED COST
Site Preparation	$10,000
2 Feeders/hoppers	$15,000
Conveyor (150 ft)	$37,500
Double deck wet screen (6X16), and Dewater screen (4X16)	$80,000
Conveyor (75 ft)	$19,000
Sump/pump/piping/drains	$15,000
Thickener/pump	$100,000 *
Agitated storage tanks (2) w/ pumps	$50,000
Structural steel - 10T	$10,000
Siding	$2,000
Heat/light/ventilation	$5,000
Foundation concrete/excavation	$20,000
Power distribution/starters	$37,500
Subtotal	$401,000
Engineering and construction management-20%	$80,200
TOTAL CAPITAL COST ESTIMATE	$481,200

* Assumes used equipment would be available.

Table F-4
MILLING/FLOTATION PLANT
CAPITAL COST ESTIMATE
(for Alternatives 2 and 3)

ITEM	ESTIMATED INSTALLED COST
Weigh feeder/belt to mill	$5,000
Ball mill/foundation (8X9 ft)	$150,000 *
Ball mill sump/pump/cyclone/piping	$10,000
Foundations/structural steel/hoist - 15T	$15,000
Lead/zinc sulfide and oxide circuits	$130,000
Thickener/foundation/underflow pump	$125,000 *
Head tank	$10,000
Tailing piping	$75,000
Structural steel/lifts/hoists - 25T	$25,000
Siding/insulation	$20,000
Heat/light/ventilation	$25,000
Foundation/concrete/excavation	$30,000
Sumps/pumps	$15,000
Water supply	$65,000
Power distribution/starters	$75,000
Substation facility	$100,000
Change house/administration facility	$30,000
Subtotal	$905,000
Engineering and construction management-20%	$181,000
TOTAL CAPITAL COST ESTIMATE	$1,086,000

* Assumes used equipment would be available.

Table F-5
MINE WASTE SCREENING PLANT
CAPITAL COST ESTIMATE
(for Alternative 5)

ITEM	ESTIMATED INSTALLED COST
Hopper/feeder	$10,000
Conveyor - 150 ft	$45,000
Screen/hoppers	$30,000
Auxiliary power/dust control	$75,000
Subtotal	$160,000
Engineering and construction management-20%	$32,000
TOTAL CAPITAL COST ESTIMATE	$192,000

Table F-6

CONTAINMENT UNIT CAPITAL COST ESTIMATE

(for Alternative 4)

ITEM	UNIT COST PER ACRE	ESTIMATED INSTALLED COST
Containment Unit Liner		
Clay	$106,500	
Synthetic lining (HDPE)	$38,700	
Sand	$29,000	
Synthetic lining (HDPE)	$38,700	
Liner cover material	$38,700	
Subtotal	$251,600	
		$5,032,000
Containment Unit Cover		
Synthetic lining	$38,700	
Liner cover material	$19,400	
Clay	$71,000	
Synthetic lining	$38,700	
Sand	$29,000	
Topsoil	$35,500	
Hydro-seed	$3,000	
Subtotal	$235,300	
		$4,706,000
Other items (70% of cover/liner cost)		
Leachate detection systems		
Leachate collection systems		
Leachate treatment/disposal		
Monitoring wells		
Access roads		
Security fence		
Operations facility		
Quality control testing		
Subtotal		$6,816,600
Subtotal		$16,554,600
Engineering and construction management-20%		$3,310,920
TOTAL CAPITAL COST ESTIMATE		$19,865,520

6.0 FACILITY OPERATING COSTS

6.1 Mine waste crushing plant

The mine waste rock crushing facility would be operated one shift per day for five days per week. About 662,000 tons of waste rock would be processed at a nominal 100 TPH for the life of the project.

Labor

Labor for this operation is included in the labor estimate for the milling/flotation unit (see Action 8.3).

Equipment

Leased - lump sum $50,000	$0.08/ton

Maintenance

Annual estimate 2% of capital costs	$0.07/ton

Utilities

Electricity - 4 kwh/ton
$0.20/ton

TOTAL	$0.35/ton

6.2 Chat wet screening plant

The chat wet screening plant would be operated for two shifts per day, five days per week at a nominal 100 TPH rate. About 1,000,000 tons of chat would be processed through this facility during the life of the project for Alternative 2 or about 500,000 tons processed for Alternative 3.

Labor	
2 operators per shift @ $19.50/hour	$0.40/ton
Equipment	
Leased - lump sum $50,000	$0.05/ton
Maintenance	
Annual estimate 2% of capital costs	$0.02/ton
Utilities	
Electricity - negligible	
TOTAL	$0.47/ton

6.3 Mill/flotation plant

The milling/flotation plant would be operated 3 shifts per day, seven days per week for the life of the project. The waste rock and/or chat would be processed at about 220,000 tons per year.

Labor	
16 operators - mill	
5 supervisors	
2 operators - crusher	
4 maintenance mechanics	
4 E&I mechanics	
1 chemist	
1 general superintendent	
33 @ an average $19.50 per hour	$6.08/ton
Maintenance	
Annual estimate 2% of capital costs	$0.10/ton
Utilities	
Electricity - 20 kwh/ton	$1.00/ton
Process Supplies	
Reagents (conditioners/frothers)	
Balls	
Liners	
Flocculants	
Subtotal	
$4.75/ton	
TOTAL	$11.93/ton

6.4 Mine waste screening plant

The mine waste rock screening facility would be operated one shift per day, 5 days per week. About 662,000 tons of waste rock would be processed at a nominal 500 TPH for the life of the project.

Labor	
3 operators per shift @ $19.50/hour	$0.19/ton
Equipment	
Leased - lump sum $50,000	$0.08/ton
Maintenance	
Annual estimate 2% of capital costs	$0.01/ton
Utilities	
Electricity - negligible	
TOTAL	$0.28/ton

7.0 SUPPORTING FIELD WORK COSTS

7.1 Chat characterization

The chat characterization task involves sampling the chat using surface, depth, and auger samples to characterize the lead and zinc concentrations. A portable XRF unit will be used for estimating concentrations in the field. This activity will be conducted concurrently with chat removal operations (about a 3 year time frame for alternative 3 and a 1 year time frame for Alternative 5).

Labor	
2 operators @ $19.50 per hour	$243,400
Equipment	
Sample truck with drill rig	$100,000
XRF portable units (2)	$ 50,000
TOTAL	$393,400

7.2 Cut/fill design

This action would support the alternatives that would dispose of the mine waste rock and chat into adjacent mine working subsidences and shafts. For each area containing mine wastes and disposal space, a cut/fill engineering plan would be developed. The cut/fill plan would verify the volume of the waste and the disposal space and determine the most efficient disposal operation. Conducting this study for each general area will minimize the amount of waste that may have to be transferred to other disposal areas. In addition, the cut/fill plan will help ensure that the waste removal operations assist in providing a recontour that will promote good drainage. The estimate for these engineering services is $197,200

8.0 RECONTOUR/REVEGETATION COSTS

This task provides for limited recontouring and revegetation remedial actions. The amount of revegetation and recontouring is dependent upon the other components of a particular alternative. For those alternatives that support removal of all the surface mine wastes, recontouring is performed during the waste removal actions. However, for Alternative 3, half the volume of chat is left on the surface after the removal action is complete. These remaining chat piles will be recontoured in order to support the revegetation actions. The estimated cost for this recontouring is $0.35 per ton of chat.

Revegetation includes addition of top soil and/or soil amendments to the ground cover so as to support vegetative growth. These areas will then be seeded. EPA Region VII unit cost estimate for these activities is $1,000 per acre.

The amount of acres requiring revegetation is dependent on the particular alternative. For Alternatives 2 and 4, it assumed that about half of the 710 disturbed acres can be revegetated. This is the estimated disturbed area not including the open subsidences and pits (say 360 acres). For Alternative 3, it is assumed that slightly more acres (say 400) will be available for revegetation because the remaining chat can be used to fill adjacent subsidence areas. Alternative 7 will provide the most acres for revegetation because of the extensive backfill of subsidences. This cost estimate assumes about 80% of the 710 disturbed acres (568) will be revegetated.

9.0 RECHANNELIZATION

9.1 Tributary A

The Tributary A diversion would establish a lined channel through the Hells Half Acre area. This channel would be constructed in an existing, unlined channel that has deteriorated. The new channel will consist of 3,800 feet of concrete-lined channel with a 10 year storm capacity (300 cfs). About 1,600 feet of the 3,800 feet is estimated to required additional earthwork to reconstruct the existing channel around subsidence areas.

Unit costs

Concrete	$70/foot of channel (3,800 feet)
Earthwork	$10/foot of channel (3,800 feet)
Additional earthwork	$80/foot of channel (1,600 feet)
8 four-foot drop structures	$10,000 each

TOTAL COST $512,000

The long-term annual O&M associated with this action involves maintaining the channel and drop structures. The annual cost is estimated to be 0.5% of the installation cost

9.2 Owl Branch Lined Channel

The Owl Branch diversion would establish a lined channel through the Blue Hole subsidence area. This channel would consist of 1,800 feet of concrete-lined channel with a 10 year storm capacity (320 cfs). About 1,600 feet of the 3,800 feet is estimated to required additional earthwork to reconstruct the existing channel around subsidence areas.

Unit costs

Concrete	$70/foot of channel (1,800 feet)
Earthwork	$10/foot of channel (1,800 feet)
4 four-foot drop structures	$10,000 each

TOTAL COST $184,000

The long-term annual O&M associated with this action involves maintaining the channel and drop structures. The annual cost is estimated to be 0.5% of the installation cost

10.0 REMEDIATE DEEP WELLS

This action assumes one deep aquifer well on the subsite requires plugging (Eagle Picher) and three deep aquifer wells require rehabilitation (City of Galena wells 1 and 2 and the Jayhawk well).

10.1 Rehabilitation

Well rehabilitation costs assume that the well is 16 inch in diameter, 1200 feet deep, pump set at 500 feet with an 8 inch diameter column, and that the pump column will require complete replacement. This action consists of the following items (per well):

Pull pump	$ 3,500
Televiewer inspection	$ 1,200
Cement well	$ 8,500
Drill out cement	$ 5,000
Replace pump column	$31,500
Install pump	$ 3,500
Total	$53,200

10.2 Plugging

Deep aquifer well plugging will be implemented on the Eagle Picher well which has essentially been abandoned. The cost estimate assumes that the well is 1200 feet deep, with no obstructions, and requires complete plugging with cement.

Mobilization/expenses	$ 1,800
Televiewer inspection	$ 1,200
Labor for plug	$ 3,800
Cement	$ 9,200
Total	$16,000

11.0 MONITORING WATER QUALITY

The water monitoring action would establish 10 groundwater monitoring locations and 4 surface water locations. The quality of the water would be monitored twice a year from each location, from the time implementation of the remedy begins for five years. Information obtained from the program would be used to determine initial water quality, monitor the potential impacts to water quality during remedy implementation, and then evaluate the effectiveness of the remedy after implementation is complete. After five years, the cost of the monitoring program will be included in the periodic five-year review required by the NCP.

11.1 Well construction

Eight new wells would be constructed for the monitoring program. Two of the monitoring wells are (or will be) existing - the City of Galena deep aquifer well and the new deep aquifer well that will be constructed for the new RWD.

Well construction costs	
8 wells, 75 feet depth @ $100/foot	$60,000

11.2 Monitoring program

Cost per event (one event every 6 months)	
Sampling	$ 4,000
Analyses	$ 7,000
Subtotal	$11,000
TOTAL COST FOR 5-YEAR MONITORING PROGRAM	$170,000

APPENDIX F.3
ALTERNATIVE COST SUMMARY TABLES

ALTERNATIVE 2 COST SUMMARY TABLE MINE AND MILL ALL MINE WASTES
GALENA SUBSITE, CHEROKEE COUNTY, KS

TASK NO.	TASK:	PROJECT CAPITAL COSTS QTY	UNIT COST	COST	TASK SUBTOTAL	CAP. RPLMNT. COSTS 15 YEAR	30 YEAR	LONG-TERM ANNUAL OPERATING COSTS	SUBT
1.0	ACTIONS TO SUPPORT MILLING/FLOTATION ALTERNATIVES								
1.1	Mining								
	Mine/load waste rock	662,603 TONS	$2.22 $/TON	$1,470,979					
	Mine/load chat	1,012,303 TONS	$1.25 $/TON	$1,265,379					
					$2,736,357				
1.2	Haul to mill								
	Waste rock	927,542 TON-MILES	$0.35 $/TON-MIL	$324,640					
	Chat	1,242,472 TON-MILES	$0.35 $/TON-MIL	$434,865					
					$759,505				
2.0	ACTIONS TO SUPPORT MINE WASTE BACKFILL								
2.1	Mine/Dispose (Intrazone material)								
	Waste rock	0 TONS MW	$3.65 $/TON	$0					
	Chat	0 TONS CHAT	$2.42 $/TON	$0					
					$0				
2.2	Mine/Dispose (Interzone material)								
	Waste rock	0 TONS MW	$4.03 $/TON	$0					
	Chat	0 TONS CHAT	$2.63 $/TON	$0					
					$0				
2.3	Chat cover material	0 TONS CHAT	$2.00 $/TON	$0				Cover maintenance	1.00%
					$0				
3.0	ACTIONS TO SUPPORT ON-SITE CONTAINMENT								
3.1	Mining								
	Mine/load waste rock	0 TONS	$2.22 $/TON	$0					
	Mine/load chat	0 TONS	$1.25 $/TON	$0					
					$0				
3.2	Haul to containment unit								
	Waste rock	0 TON-MILES	$0.35 $/TON-MIL	$0					
	Chat	0 TON-MILES	$0.35 $/TON-MIL	$0					
					$0				
3.3	Containment unit filling								
	Waste Rock/Chat	0 cubic yds	$1.34 $/CU YD	$0					
					$0				
4.0	MISCELLANEOUS ADDITIONAL SITE WORK								
	5% of total earthwork cost			$174,793					
					$174,793				
5.0	FACILITY CAPITAL COSTS								
5.1	Mine waste crushing plant	$789,600		$789,600					
5.2	Chat wet screening plant	$481,200		$481,200					
5.3	Mill/flotation plant	$1,086,000		$1,086,000					
5.4	Mine waste screening plant			$0					
5.5	Containment unit							Containment maintenance	1.00%
					$2,356,800				
6.0	FACILITY OPERATING COSTS								
6.1	Mine waste crushing plant	662,603 TONS MW	$0.35 $/TON	$231,911					
6.2	Chat wet screening plant	1,012,303 TONS CHAT	$0.47 $/TON	$475,782					
6.3	Mill/flotation plant	662,603 TONS MW	$11.93 $/TON	$7,904,854					
		202,461 TONS CHAT	$11.93 $/TON	$2,415,355					
6.4	Mine waste screening plant	TONS MW	$0.28 $/TON	$0					
					$11,027,902				
7.0	SUPPORTING FIELDWORK COSTS								
7.1	Chat characterization	0		$0					
7.2	Cut/fill engineering	0		$0					
					$0				
8.0	RECONTOUR/REVEGETATION COSTS								
		360 ACRES	$1,000 $/ACRE	$360,000					
		0 TONS CHAT	$0.35 $/TON	$0					
					$360,000				

ALTERNATIVE 2 COST SUMMARY TABLE MINE AND MILL ALL MINE WASTES
GALENA SUBSITE, CHEROKEE COUNTY, KS

TASK NO.	TASK:	PROJECT CAPITAL COSTS QTY	UNIT COST	COST	TASK SUBTOTAL	CAP. RPLMNT. COSTS 15 YEAR	30 YEAR	LONG-TERM ANNUAL OPERATING COSTS		SUBTOTAL
9.0	RECHANNELIZATION									
9.1	Tributary A	512,000		$512,000			$512,000	Channel maintenance	0.50%	$2,560
9.2	Owl Branch Lined Channel	184,000		$184,000			$184,000	Channel maintenance	0.50%	$920
					$696,000					
10.0	REMEDIATE DEEP WELLS									
10.1	Rehabilitate well	3 WELLS	$53,200 $/WELL	$159,600			$159,600			
10.2	Plug well	1 WELLS	$16,000 $/WELL	$16,000			$16,000			
					$175,600					
11.0	MONITORING WATER QUALITY									
11.1	Well construction	$60,000		$60,000						
11.2	Monitoring program	$110,000		$110,000						
					$170,000					
	PROJECT COSTS SUBTOTAL				$18,456,958	$0	$871,600	ANNUAL COSTS SUBTOTAL		$3,480
	BID and SCOPE CONTINGENCIES	10%			$1,845,696	$0	$87,160			$348
	PROJECT COSTS TOTAL				$20,302,653	$0	$958,760			$3,828

ALTERNATIVE 3 COST SUMMARY TABLE MINE AND MILL ALL WASTE ROCK AND HALF OF CHAT
GALENA SUBSITE, CHEROKEE COUNTY, KS

TASK NO.	TASK:	PROJECT CAPITAL COSTS QTY	UNIT COST	COST	TASK SUBTOTAL	CAP. RPLMNT. COSTS 15 YEAR	CAP. RPLMNT. COSTS 30 YEAR	LONG-TERM ANNUAL OPERATING COSTS		SUBTOTAL
1.0	ACTIONS TO SUPPORT MILLING/FLOTATION ALTERNATIVES									
1.1	Mining									
	Mine/load waste rock	662,603 TONS	$2.22 $/TON	$1,470,979						
	Mine/load chat	506,152 TONS	$1.25 $/TON	$632,689						
					$2,103,668					
1.2	Haul to mill									
	Waste rock	927,542 TON-MILES	$0.35 $/TON-M	$324,640						
	Chat	621,236 TON-MILES	$0.35 $/TON-M	$217,433						
					$542,072					
2.0	ACTIONS TO SUPPORT MINE WASTE BACKFILL									
2.1	Mine/Dispose (Intrazone material)									
	Waste rock	0 TONS MW	$3.65 $/TON	$0						
	Chat	0 TONS CHAT	$2.42 $/TON	$0						
					$0					
2.2	Mine/Dispose (Interzone material)									
	Waste rock	0 TONS MW	$4.03 $/TON	$0						
	Chat	0 TONS CHAT	$2.63 $/TON	$0						
					$0					
2.3	Chat cover material	0 TONS CHAT	$2.00 $/TON	$0				Cover maintenance	1.00%	$0
					$0					
3.0	ACTIONS TO SUPPORT ON-SITE CONTAINMENT									
3.1	Mining									
	Mine/load waste rock	0 TONS	$2.22 $/TON	$0						
	Mine/load chat	0 TONS	$1.25 $/TON	$0						
					$0					
3.2	Haul to containment unit									
	Waste rock	0 TON-MILES	$0.35 $/TON-M	$0						
	Chat	0 TON-MILES	$0.35 $/TON-M	$0						
					$0					
3.3	Containment unit filling									
	Waste Rock/Chat	0 cubic yds	$1.34 $/CU YD	$0						
					$0					
4.0	MISCELLANEOUS ADDITIONAL SITE WORK									
	5% of total earthwork cost			$132,287						
					$132,287					
5.0	FACILITY CAPITAL COSTS									
5.1	Mine waste crushing plant	$789,600		$789,600						
5.2	Chat wet screening plant	$481,200		$481,200						
5.3	Mill/flotation plant	$1,086,000		$1,086,000						
5.4	Mine waste screening plant	$0		$0						
5.5	Containment unit							Containment maintenance	1.00%	$0
					$2,356,800					
6.0	FACILITY OPERATING COSTS									
6.1	Mine waste crushing plant	662,603 TONS MW	$0.35 $/TON	$231,911						
6.2	Chat wet screening plant	506,152 TONS CHAT	$0.47 $/TON	$237,891						
6.3	Mill/flotation plant	662,603 TONS MW	$11.93 $/TON	$7,904,854						
		101,230 TONS CHAT	$11.93 $/TON	$1,207,677						
6.4	Mine waste screening plant	TONS MW	$0.28 $/TON	$0						
					$9,582,334					
7.0	SUPPORTING FIELDWORK COSTS									
7.1	Chat characterization	393,400		$393,400						
7.2	Cut/fill engineering	0		$0						
					$393,400					
8.0	RECONTOUR/REVEGETATION COSTS									
		400 ACRES	$1,000 $/ACRE	$400,000						
		500,000 TONS CHAT	$0.35 $/TON	$175,000						
					$575,000					

ALTERNATIVE 3 COST SUMMARY TABLE MINE AND MILL ALL WASTE ROCK AND HALF OF CHAT
GALENA SUBSITE, CHEROKEE COUNTY, KS

TASK NO.	TASK:	PROJECT CAPITAL COSTS QTY	UNIT COST	COST	TASK SUBTOTAL	CAP. RPLMNT. COSTS 15 YEAR	30 YEAR	LONG-TERM ANNUAL OPERATING COSTS		SUBTOTAL
9.0	RECHANNELIZATION									
9.1	Tributary A	512,000		$512,000			$512,000	Channel maintenance	0.50%	$2,560
9.2	Owl Branch Lined Channel	184,000		$184,000			$184,000	Channel maintenance	0.50%	$920
					$696,000					
10.0	REMEDIATE DEEP WELLS									
10.1	Rehabilitate well	3 WELLS	$53,200 $/WELL	$159,600			$159,600			
10.2	Plug well	1 WELLS	$16,000 $/WELL	$16,000			$16,000			
					$175,600					
11.0	MONITORING WATER QUALITY									
11.1	Well construction	$60,000		$60,000						
11.2	Monitoring program	$110,000		$110,000						
					$170,000					
	PROJECT COSTS SUBTOTAL				$16,727,161	$0	$871,600	ANNUAL COSTS SUBTOTAL		$3,480
	BID and SCOPE CONTINGENCIES	10%			$1,672,716	$0	$87,160			$348
	PROJECT COSTS TOTAL				$18,399,877	$0	$958,760			$3,828

ALTERNATIVE COST SUMMARY TABLE MINE AND DISPOSE OF ALL MINE WASTES IN ON-SITE CONTAINMENT FACILITY
GALENA SUBSITE, CHEROKEE COUNTY, KS

TASK NO.	TASK:	PROJECT CAPITAL COSTS QTY	UNIT COST	COST	TASK SUBTOTAL	CAP. RPLMNT. COSTS 15 YEAR 30 YEAR	LONG-TERM ANNUAL OPERATING COSTS		SUBTOTAL
1.0	ACTIONS TO SUPPORT MILLING/FLOTATION ALTERNATIVES								
1.1	Mining								
	Mine/load waste rock	0 TONS	$2.22 $/TON	$0					
	Mine/load chat	0 TONS	$1.25 $/TON	$0					
					$0				
1.2	Haul to mill								
	Waste rock	0 TON-MILES	$0.35 $/TON-	$0					
	Chat	0 TON-MILES	$0.35 $/TON-	$0					
					$0				
2.0	ACTIONS TO SUPPORT MINE WASTE BACKFILL								
2.1	Mine/Dispose (Intrazone material)								
	Waste rock	0 TONS MW	$3.65 $/TON	$0					
	Chat	0 TONS CHAT	$2.42 $/TON	$0					
					$0				
2.2	Mine/Dispose (Interzone material)								
	Waste rock	0 TONS MW	$4.03 $/TON	$0					
	Chat	0 TONS CHAT	$2.63 $/TON	$0					
					$0				
2.3	Chat cover material	0 TONS CHAT	$2.00 $/TON	$0			Cover maintenance	1.00%	$0
					$0				
3.0	ACTIONS TO SUPPORT ON-SITE CONTAINMENT								
3.1	Mining								
	Mine/load waste rock	662,603 TONS	$2.22 $/TON	$1,470,979					
	Mine/load chat	1,012,304 TONS	$1.25 $/TON	$1,265,380					
					$2,736,359				
3.2	Haul to containment unit								
	Waste rock	772,167 TON-MILES	$0.35 $/TON-	$270,258					
	Chat	1,237,618 TON-MILES	$0.35 $/TON-	$433,166					
					$703,425				
3.3	Containment unit filling								
	Waste Rock/Chat	1,300,000 cubic yds	$1.34 $/CU Y	$1,742,000					
					$1,742,000				
4.0	MISCELLANEOUS ADDITIONAL SITE WORK								
	5% of total earthwork cost			$259,089					
					$259,089				
5.0	FACILITY CAPITAL COSTS								
5.1	Mine waste crushing plant	$0		$0					
5.2	Chat wet screening plant	$0		$0					
5.3	Mill/flotation plant	$0		$0					
5.4	Mine waste screening plant	$0		$0					
5.5	Containment unit	$19,865,520		$19,865,520			Containment maintenance	1.00%	$198,655
					$19,865,520				
6.0	FACILITY OPERATING COSTS								
6.1	Mine waste crushing plant	0 TONS MW	$0.35 $/TON	$0					
6.2	Chat wet screening plant	0 TONS CHAT	$0.47 $/TON	$0					
6.3	Mill/flotation plant	0 TONS MW	$11.93 $/TON	$0					
		0 TONS CHAT	$11.93 $/TON	$0					
6.4	Mine waste screening plant	0 TONS MW	$0.28 $/TON	$0					
					$0				
7.0	SUPPORTING FIELDWORK COSTS								
7.1	Chat characterization	$0		$0					
7.2	Cut/fill engineering	$0		$0					
					$0				
8.0	RECONTOUR/REVEGETATION COSTS								
		360 ACRES	$1,000 $/ACRE	$360,000					
		0 TONS CHAT	$0.35 $/TON	$0					
					$360,000				

ALTERNATIVE 4 COST SUMMARY TABLE MINE AND DISPOSE OF ALL MINE WASTES IN ON-SITE CONTAINMENT FACILITY
GALENA SUBSITE, CHEROKEE COUNTY, KS

TASK NO.	TASK:	PROJECT CAPITAL COSTS QTY	UNIT COST	COST	TASK SUBTOTAL	CAP. RPLMNT. COSTS 15 YEAR	30 YEAR	LONG-TERM ANNUAL OPERATING COSTS		SUBTOTAL
9.0	RECHANNELIZATION									
9.1	Tributary A	512,000		$512,000			$512,000	Channel maintenance	0.50%	$2,560
9.2	Owl Branch Lined Channel	184,000		$184,000			$184,000	Channel maintenance	0.50%	$920
					$696,000					
10.0	REMEDIATE DEEP WELLS									
10.1	Rehabilitate well	3 WELLS	$53,200 $/WELL	$159,600			$159,600			
10.2	Plug well	1 WELLS	$16,000 $/WELL	$16,000			$16,000			
					$175,600					
11.0	MONITORING WATER QUALITY									
11.1	Well construction	$60,000		$60,000						
11.2	Monitoring program	$110,000		$110,000						
					$170,000					
	PROJECT COSTS SUBTOTAL				$26,707,993	$0	$871,600	ANNUAL COSTS SUBTOTAL		$202,135
	BID and SCOPE CONTINGENCIES	10%			$2,670,799	$0	$87,160			$20,214
	PROJECT COSTS TOTAL				$29,378,792	$0	$958,760			$222,349

ALTERNATIVE 5 COST SUMMARY TABLE GEOCHEMICALLY CHARACTERIZE WASTES, SEGREGATE, SELECTIVELY BACKFILL AND RECONTOUR
GALENA SUBSITE, CHEROKEE COUNTY, KS

TASK NO.	TASK:	PROJECT CAPITAL COSTS QTY	UNIT COST	COST	TASK SUBTOTAL	CAP. RPLMNT. COSTS 15 YEAR	30 YEAR	LONG-TERM ANNUAL OPERATING COSTS		SUBTOTAL
1.0	ACTIONS TO SUPPORT MILLING/FLOTATION ALTERNATIVES									
1.1	Mining									
	Mine/load waste rock	0 TONS	$2.22 $/TON	$0						
	Mine/load chat	0 TONS	$1.25 $/TON	$0						
					$0					
1.2	Haul to mill									
	Waste rock	0 TON-MILES	$0.35 $/TON-	$0						
	Chat	0 TON-MILES	$0.35 $/TON-	$0						
					$0					
2.0	ACTIONS TO SUPPORT MINE WASTE BACKFILL									
2.1	Mine/Dispose (Intrazone material)									
	Waste rock	511,117 TONS MW	$3.65 $/TON	$1,865,577						
	Chat	440,589 TONS CHAT	$2.42 $/TON	$1,066,225						
					$2,931,802					
2.2	Mine/Dispose (Interzone material)									
	Waste rock	151,487 TONS MW	$4.03 $/TON	$610,493						
	Chat	65,562 TONS CHAT	$2.63 $/TON	$172,428						
					$782,921					
2.3	Chat cover material	506,151 TONS CHAT	$2.00 $/TON	$1,012,302				Cover maintenance	1.00%	$10,123
					$1,012,302					
3.0	ACTIONS TO SUPPORT ON-SITE CONTAINMENT									
3.1	Mining									
	Mine/load waste rock	0 TONS	$2.22 $/TON	$0						
	Mine/load chat	0 TONS	$1.25 $/TON	$0						
					$0					
3.2	Haul to containment unit									
	Waste rock	0 TON-MILES	$0.35 $/TON-	$0						
	Chat	0 TON-MILES	$0.35 $/TON-	$0						
					$0					
3.3	Containment unit filling									
	Waste Rock/Chat	0 cubic yds	$1.34 $/CU Y	$0						
					$0					
4.0	MISCELLANEOUS ADDITIONAL SITE WORK									
	5% of total earthwork cost			$236,351						
					$236,351					
5.0	FACILITY CAPITAL COSTS									
5.1	Mine waste crushing plant	$0		$0						
5.2	Chat wet screening plant	$0		$0						
5.3	Mill/flotation plant	$0		$0						
5.4	Mine waste screening plant	$192,000		$192,000						
5.5	Containment unit							Containment maintenance	1.00%	$0
					$192,000					
6.0	FACILITY OPERATING COSTS									
6.1	Mine waste crushing plant	0 TONS MW	$0.35 $/TON	$0						
6.2	Chat wet screening plant	TONS CHAT	$0.47 $/TON	$0						
6.3	Mill/flotation plant	0 TONS MW	$11.93 $/TON	$0						
		TONS CHAT	$11.93 $/TON	$0						
6.4	Mine waste screening plant	662,604 TONS MW	$0.28 $/TON	$185,529						
					$185,529					
7.0	SUPPORTING FIELDWORK COSTS									
7.1	Chat characterization	$393,400		$393,400						
7.2	Cut/fill engineering	$197,200		$197,200						
					$590,600					
8.0	RECONTOUR/REVEGETATION COSTS									
		568 ACRES	$1,000 $/ACRE	$568,000						
		0 TONS CHAT	$0.35 $/TON	$0						
					$568,000					

ALTERNATIVE 5 COST SUMMARY TABLE GEOCHEMICALLY CHARACTERIZE WASTES, SEGREGATE, SELECTIVELY BACKFILL AND RECOUNTOUR
GALENA SUBSITE, CHEROKEE COUNTY, KS

TASK NO.	TASK:	PROJECT CAPITAL COSTS QTY	UNIT COST	COST	TASK SUBTOTAL	CAP. RPLMNT. COSTS 15 YEAR	30 YEAR	LONG-TERM ANNUAL OPERATING COSTS		SUBTOTAL
9.0	RECHANNELIZATION									
9.1	Tributary A	512,000		$512,000			$512,000	Channel maintenance	0.50%	$2,560
9.2	Owl Branch Lined Channel	184,000		$184,000			$184,000	Channel maintenance	0.50%	$920
					$696,000					
10.0	REMEDIATE DEEP WELLS									
10.1	Rehabilitate well	3 WELLS	$53,200 $/WELL	$159,600			$159,600			
10.2	Plug well	1 WELLS	$16,000 $/WELL	$16,000			$16,000			
					$175,600					
11.0	MONITORING WATER QUALITY									
11.1	Well construction	$60,000		$60,000						
11.2	Monitoring program	$110,000		$110,000						
					$170,000					
	PROJECT COSTS SUBTOTAL				$7,541,105	$0	$871,600	ANNUAL COSTS SUBTOTAL		$13,603
	BID and SCOPE CONTINGENCIES	10%			$754,111	$0	$87,160			$1,360
	PROJECT COSTS TOTAL				$8,295,216	$0	$958,760			$14,963

APPENDIX F.4
ALTERNATIVE PRESENT WORTH CALCULATIONS

ALTERNATIVE 2 PRESENT WORTH ANALYSIS
GALENA SUBSITE, CHEROKEE COUNTY, KS

Assumptions:
- o Present worths calculated with 5 percent discount rate for a 30-year period.
- o All project capital costs (all costs associated with alternative implementation) assumed spent in year zero
- o All costs are in 1989 dollars.

	1989 COSTS	PRESENT WORTH VALUE
PROJECT CAPITAL COSTS	$20,302,653	$20,302,653
15 YEAR CAPITAL REPLACEMENT COSTS	$0	$0
30 YEAR CAPITAL REPLACEMENT COSTS	$958,760	$221,838
ANNUAL LONG-TERM O&M COSTS	$3,828	$58,846
5-YEAR PERIODIC REVIEW	$10,000	$153,725
TOTAL PRESENT WORTH VALUE:		$20,737,062

ALTERNATIVE 3 PRESENT WORTH ANALYSIS
GALENA SUBSITE, CHEROKEE COUNTY, KS

Assumptions:
- o Present worths calculated with 5 percent discount rate for a 30-year period.
- o All project capital costs (all costs associated with alternative implementation) assumed spent in year zero.
- o All costs are in 1989 dollars.

	1989 COSTS	PRESENT WORTH VALUE
PROJECT CAPITAL COSTS	$18,399,877	$18,399,877
15 YEAR CAPITAL REPLACEMENT COSTS	$0	$0
30 YEAR CAPITAL REPLACEMENT COSTS	$958,760	$221,838
ANNUAL LONG-TERM O&M COSTS	$3,828	$58,846
5-YEAR PERIODIC REVIEW	$10,000	$153,725
TOTAL PRESENT WORTH VALUE:		$18,834,285

ALTERNATIVE 4 PRESENT WORTH ANALYSIS
GALENA SUBSITE, CHEROKEE COUNTY, KS

Assumptions:
- o Present worths calculated with 5 percent discount rate for a 30-year period.
- o All project capital costs (all costs associated with alternative implementation) assumed spent in year z
- o All costs are in 1989 dollars.

	1989 COSTS	PRESENT WORTH VALUE
PROJECT CAPITAL COSTS	$29,378,792	$29,378,792
15 YEAR CAPITAL REPLACEMENT COSTS	$0	$0
30 YEAR CAPITAL REPLACEMENT COSTS	$958,760	$221,838
ANNUAL LONG-TERM O&M COSTS	$222,349	$3,418,045
5-YEAR PERIODIC REVIEW	$10,000	$153,725
TOTAL PRESENT WORTH VALUE:		$33,172,399

ALTERNATIVE 5 PRESENT WORTH ANALYSIS
GALENA SUBSITE, CHEROKEE COUNTY, KS

Assumptions:
- o Present worths calculated with 5 percent discount rate for a 30-year period.
- o All project capital costs (all costs associated with alternative implementation) assumed spent in year zero.
- o All costs are in 1989 dollars.

	1989 COSTS	PRESENT WORTH VALUE
PROJECT CAPITAL COSTS	$8,295,216	$8,295,216
15 YEAR CAPITAL REPLACEMENT COSTS	$0	$0
30 YEAR CAPITAL REPLACEMENT COSTS	$958,760	$221,838
ANNUAL LONG-TERM O&M COSTS	$14,963	$230,023
5-YEAR PERIODIC REVIEW	$10,000	$153,725
TOTAL PRESENT WORTH VALUE:		$8,900,801