EPA 530-R-94-031
NTIS PB94-200979

TECHNICAL RESOURCE DOCUMENT

EXTRACTION AND BENEFICIATION OF ORES AND MINERALS

VOLUME 4

COPPER

August 1994

U.S. Environmental Protection Agency
Office of Solid Waste
Special Waste Branch
401 M Street, SW
Washington, DC 20460

DISCLAIMER AND ACKNOWLEDGEMENTS

This document was prepared by the U.S. Environmental Protection Agency (EPA). The mention of company or product names is not to be considered an endorsement by the U.S. Government or by the EPA.

This Technical Resource Document consists of four sections. The first is EPA's Profile of the copper industry; the remaining sections are Reports from several site visits conducted by EPA. The Profile Section was distributed for review to the U.S. Department of the Interior's (DOI's) Bureau of Mines and Bureau of Land Management; the States of Arizona, New Mexico, and Utah; the American Mining Congress (AMC), and environmental organizations. Summaries of the comments and EPA's responses are presented as Appendix 1-A to the Profile Section. The Site Visit Reports were reviewed by individual company, State, and Federal representatives who participated in the site visit. Comments and EPA responses are included as appendices to the specific Site Visit Section. EPA is grateful to all individuals who took the time to review sections of this Technical Resource Document.

The use of the terms "extraction," "beneficiation," and "mineral processing" in the Profile section of this document is not intended to classify any waste streams for the purposes of regulatory interpretation or application. Rather, these terms are used in the context of common industry terminology.

TABLE OF CONTENTS

APPENDICES

LIST OF TABLES

Page

LIST OF FIGURES

Page

1.0 MINING INDUSTRY PROFILE: COPPER

1.1 INTRODUCTION

This Industry Profile presents the results of U.S. Environmental Protection Agency (EPA) research into the domestic copper mining industry and is one of a series of profiles of major mining sectors. Additional profiles describe lode gold, placer gold, lead/zinc, iron, and several industrial mineral sectors, as presented in the current literature. EPA has prepared these profiles to enhance and update its understanding of the mining industry and to support mining program development by the states. EPA believes the profiles represent current environmental management practices as described in the literature.

Each profile addresses the extraction and beneficiation of ore. The scope of the Resource Conservation and Recovery Act (RCRA) as it applies to mining waste was amended in 1980 when Congress passed the Bevill Amendment, Section 3001(b)(3)(A). The Bevill Amendment states that "solid waste from extraction, beneficiation, and processing of ores and minerals" is excluded from the definition of hazardous waste under Subtitle C of RCRA (40 CFR 261.4(b)(7)). The exemption was conditional on EPA's completion of studies required by RCRA Sections 8002(f) and (p) on the environmental and health consequences of the disposal and use of these wastes. EPA submitted the initial results of these studies in the 1985 *Report to Congress: Wastes from the Extraction and Beneficiation of Metallic Ores, Phosphate Rock, Asbestos, Overburden From Uranium Mining, and Oil Shale* (U.S. EPA 1985a). In July 1986, EPA made a regulatory determination that regulation of extraction and beneficiation wastes under Subtitle C of RCRA was not appropriate (51 FR 24496; July 3, 1986). EPA concluded that Subtitle C controls were unwarranted and found that a wide variety of existing Federal and State programs already addressed many of the risks posed by extraction and beneficiation wastes. Instead of regulating extraction and beneficiation wastes as hazardous wastes under Subtitle C, EPA indicated that these wastes should be controlled under Subtitle D of RCRA.

EPA reported their initial findings on mineral processing wastes from the studies required by the Bevill Amendment in the 1990 *Report to Congress: Special Wastes From Mineral Processing* (U.S. EPA 1990a). This report covered 20 specific mineral processing wastes; 3 of the 20 involved copper processing wastes. In June 1991, EPA issued a regulatory determination (56 FR 27300) stating that regulation of these 20 mineral processing wastes as hazardous wastes under RCRA Subtitle C is inappropriate or infeasible. These 20 wastes, including slag from primary copper processing, calcium sulfate wastewater treatment plant sludge from primary copper processing, and slag tailings from primary copper processing, are subject to applicable state requirements. Any mineral processing wastes not specifically included in this list of 20 wastes no longer qualifies for the exclusion
(54 FR 36592). Due to the timing of this decision and the limited numbers of copper industry wastes at issue, copper processing wastes are not addressed in this profile.

In addition to preparing profiles, EPA has undertaken a variety of activities to support State mining programs. These activities include visiting a number of mine sites; compiling data from State regulatory agencies on waste characteristics, releases, and environmental effects; preparing summaries of mining-related sites on the National Priorities List (NPL); and examining specific waste management practices and technologies. EPA has also conducted studies of State mining-related regulatory programs and their implementation.

The purpose of this profile is to provide additional information on the domestic copper mining industry. The report describes copper extraction and beneficiation operations with specific reference to the wastes associated with these operations. The report is based on literature reviews and on comments received on earlier drafts. This report complements, but was developed independently of, other Agency activities, including those described above.

This profile briefly characterizes the economics of the industry and the geology of copper ores. Following this discussion is a review of copper extraction and beneficiation methods; this section provides the context for descriptions of wastes and materials managed by the industry, as well as a discussion of the potential environmental effects that may result from copper mining. Appendix 1-B of this profile presents case studies of extraction and beneficiation methods at nine large copper mines in the United States in 1990. The profile concludes with a description of the current regulatory programs that apply to the copper mining industry as implemented by EPA, Federal land management agencies, and the State of Arizona.

1.2 ECONOMIC CHARACTERIZATION OF THE INDUSTRY

The physical properties of copper, including malleability and workability, corrosion resistance and durability, high electrical and thermal conductivity, and ability to alloy with other metals, have made it an important metal to a number of diverse industries. Copper was an historically important resource for the production of tools, utensils, vessels, weapons, and objects of art. According to the Bureau of Mines, in 1992, copper production was used for building construction (41 percent), electrical and electronic products (24 percent), industrial machinery and equipment (13 percent), transportation (12 percent), and consumer products (10 percent) (U.S. DOI, Bureau of Mines 1993a).

The United States is the second largest copper producer in the world. Next to Chile, the United States had the second largest reserves (45 million metric tons) and reserve base (90 million metric tons) of contained copper in 1992. Also, in 1992, United States' copper operations produced about 1.7 million metric tons. In 1991, 1.63 million metric tons were produced. The total value of copper produced in 1992 ($4.1 billion) is slightly more than 1991's value ($3.9 billion). Arizona led production in 1992, followed by New Mexico, Utah, Michigan, and Montana. In the same year, copper was also recovered from mines in seven other States (U.S. DOI, Bureau of Mines 1993a, 1993b).

In 1991, the top 25 copper producers in the United States generated more than 95 percent of the United States' domestic copper production. These producers are listed in Table 1-1

Table 1-1. Leading United States Copper Producers in 1991, by Output

Rank	Mine	County and State	Operator
1	Morenci/Metcalf	Greenlee, AZ	Phelps Dodge Corporation
2	Bingham Canyon	Salt Lake, UT	Kennecott, Utah Copper Corporation
3	San Manuel	Pinal, AZ	Magma Copper Company
4	Chino	Grant, NM	Phelps Dodge Corporation
5	Tyrone	Grant, NM	Phelps Dodge Corporation, Burro Chief Copper Company
6	Sierrita	Pima, AZ	Cyprus Sierrita Corporation
7	Ray Complex	Pinal, AZ	ASARCO Incorporated
8	Bagdad	Yavapai, AZ	Cyprus Bagdad Copper Company
9	Pinto Valley	Gila, AZ	Pinto Valley Copper Corporation
10	Mission Complex	Pima, AZ	ASARCO Incorporated
11	Inspiration	Gila, AZ	Cyprus Miami Mining Corporation
12	White Pine	Ontonagon, MI	Copper Range Company
13	Continental	Silver Bow, MT	Montana Resources, Inc.
14	Twin Buttes	Pima, AZ	Cyprus Sierrita Corporation
15	Troy	Lincoln, MT	ASARCO Incorporated
16	San Xavier	Pima, AZ	ASARCO Incorporated
17	Superior (Magma)	Pinal, AZ	Magma Copper Company
18	Miami	Gila, AZ	Pinto Valley Copper Corporation
19	Casteel	Iron, MO	The Doe Run Company
20	Silver Bell	Pima, AZ	ASARCO Incorporated
21	Lakeshore	Pinal, AZ	Cyprus Casa Grande Corporation
22	Johnson	Cochise, AZ	Arimetco Incorporated
23	Oracle Ridge	Pinal, AZ	South Atlantic Ventures Ltd.
24	Yerington	Lyon, NV	Arimetco Incorporated
25	Mineral Park	Mohave, AZ	Cyprus Mineral Park

(Source: U.S. DOI, Bureau of Mines 1993b)

. By the end of 1991, 8 primary and 5 secondary smelters, 10 electrolytic and 6 fire refineries, and 14 electrowinning plants were in operation in the United States (U.S. DOI, Bureau of Mines 1993b).

In 1991, the consumption of copper and brass materials in the U.S. decreased by 4 percent from 1990 levels. Refined copper was used at approximately 20 wire-rod mills, 41 brass mills, and 750 foundries, chemical plants, and other manufacturers. The Bureau of Mines estimates that by year end 1992, United States, consumption of copper exceeded 2.1 million tons (U.S. DOI, Bureau of Mines 1993a).

Historically, the United States is one of the largest holders of refined copper reserves; it currently holds 16 percent of the world's reserves. More than 90 percent of the United States copper reserves are located in the top five copper-producing States. Copper reserves are defined by the U.S. Geological Survey (USGS) as that part of the resource base thought to be economically recoverable from operating or developing sites with existing technology. Copper reserves reported at operating or developing sites are anticipated to be sufficient to meet the projected cumulative demand of nearly 130 million tons of primary copper through the year 2000. In addition, some of the material already identified in the reserve base, once determined to be infeasible to mine, may become feasible with improved technology or higher copper prices.

The number of operating copper mines has decreased from 68 mines in 1989 to 65 mines in 1992. Of the 65 mines actively producing copper in the United States, 33 list copper as the primary product. The remaining 32 mines produce copper either as a byproduct or co-product of gold, lead, zinc, or silver (U.S. DOI, Bureau of Mines 1993b). Thirteen of the 33 active mines that primarily produce copper are located in Arizona; the remaining mines are located throughout New Mexico, Utah, Michigan, and Montana (U.S. DOI, Bureau of Mines, Unpublished).

In 1988, there were 17 copper mills using leaching methods in the United States, with total production of approximately 227,000 metric tons of electrowon copper (U.S. EPA 1989e; U.S. DOI, Bureau of Mines 1993b). According to the U.S. Bureau of Mines, in 1991 15.7 million metric tons of copper ore were beneficiated using leaching methods to recover 441,000 metric tons of copper (an increase of 194% in three years) (U.S. DOI, Bureau of Mines 1993b). While solution operations are conducted throughout the southwestern United States, almost 75 percent of the facilities (14) are located in Arizona. There are two facilities in New Mexico, one in Utah, and one in Nevada. An inventory and description of the 17 facilities that conduct leaching operations are provided in Table 1-2.

Use of the dump-leach method is common at the majority of solution operations, although an increasing number of facilities are now using underground leach methods. As an alternative to conventional surface or underground extraction techniques, *in situ* leach operations are becoming more commonplace in copper production operations. The majority of these techniques are used in old stopes or block-cave rubble where the ore deposit is disturbed. Another method, similar to underground leaching in existing mine workings is *in situ* leaching of undisturbed ore deposits. The difference being that the ore is leached in place. Such operations are considered experimental by the Bureau of Mines. Recent developments in copper solution mining technologies [e.g., *in situ* leaching, Solvent Extraction (SX), ion extraction, and Electrowinning (EW)] have significantly increased copper production from leaching operations. Many major copper mines have installed improved leach circuits, increasing their copper production by as much as 30 percent.

In 1989, approximately 50 percent of the solution operations used Solvent Extraction/Electrowinning (SX/EW) recovery methods; other mines employed cementation-type recovery units (U.S. DOI, Bureau of Mines, Unpublished). The growth in copper production that occurred in 1990 is largely the result of increases from SX/EW recovery. The SX/EW recovery of copper was 312,000 metric tons in 1989. It increased to 393,000 metric tons in 1990 and to 441,000 metric tons in 1991 (U.S. DOI, Bureau of Mines 1992, 1993b).

Table 1-2. Inventory of Active Copper Solution Mining Operations in the United States (1988)

Operation	Location	Leaching Method						Recovery Method		Capacity (Metric Tons)
		Underground Leach	In-situ*	Dump	Heap	Vat	Agitation	Precipitation	SX/EW	
ASARCO, Inc.										
Ray	Hayden, Arizona									15,000/ 29,000
Silver Bell	Marana, Arizona									6,000
Battle Mountain Gold Co.										
Battle Mountain	Battle Mountain, Nevada									5,000
Cyprus Casa Grande Corp.										
Casa Grande	Casa Grande, Arizona									10,000
Cyprus Minerals Co.										
Bagdad	Bagdad, Arizona									6,800
Mineral Park	Kingman, Arizona									3,500
Sierrita/Esperanza	Sahuarita, Arizona									6,000
Cyprus Miami	Claypool, Arizona									42,500
Kennecott										
Bingham Canyon	Bingham Canyon, Utah									36,000
Kocide Chemical										
Van Dyke	Casa Grande, Arizona									NA[**]
Leaching Technology, Inc.										
Nacimiento	Cuba, New Mexico									NA[**]
Magma Copper Co.										
Miami Leach	Miami, Arizona									5,000
Pinto Valley	Miami, Arizona									16,000
San Manuel	San Manuel, Arizona									25,000
Phelps Dodge Corp.										
Chino	Hurley, New Mexico									52,000- 53,000
Copper Queen	Bisbee, Arizona									2,500
Morenci/Metcalf	Morenci, Arizona									155,000

[*] Experimental only
[**] NA - Not available

(Source: U.S. EPA 1989e)

1.3 ORE CHARACTERIZATION

Copper is an element that occurs in minor amounts in the Earth's crust. Estimates of average crustal prevalence are on the order of 0.0058 percent by weight (U.S. DOI, Geological Survey 1973). Deposits considered to be economically recoverable at current market prices may contain as little as 0.5 percent of copper or less, depending on the mining method, total reserves, and the geologic setting of the deposit.

Copper deposits are found in a variety of geologic environments, depending on the rock-forming processes that occurred. In general, copper deposits are formed by hydrothermal processes (i.e., the minerals are precipitated as sulfides from heated waters associated with igneous intrusions or areas of otherwise abnormal lithospheric heating). Plate tectonic theory has provided a new framework for understanding the global distribution of ore deposits, since areas of plate divergences and convergences are where most hydrothermal activity occurs. These deposits can be grouped in the following broad classes: porphyry and related copper deposits, sediment-hosted copper deposits, volcanic-hosted massive sulfide deposits, veins and replacement bodies associated with metamorphic rocks, and deposits associated with ultramafic, mafic, ultrabasic, and carbonatite rocks. Each of these deposit classes is discussed further, below.

1.3.1 Porphyry Copper and Associated Deposits

The most commonly mined type of copper deposit, porphyry copper, occurs mainly in magmatic, volcanic arc, and back-arc tectonic regions of plates along modern or ancient subduction zones (plate-convergence boundaries). As a consequence, it is found predominantly in areas such as along the western continental edges of North and South America (U.S. DOI, Bureau of Mines 1992). Major copper porphyry deposits are also located in the southwestern United States, associated with large granitic intrusions.

Copper porphyry deposits are a type of disseminated mineral deposit, found dispersed throughout small fractures in porphyritic felsic intrusives (granitic rocks with large feldspar or quartz crystals in a finer matrix). By an unknown process, intrusive granitic plutons are fractured into pieces and the tiny veins and pore fillings are filled with the hydrothermal solutions, recementing the rock with the mineral-laden deposits. When the host rock is limestone, the resulting deposits are called "skarn" deposits. Because the copper is so physically dispersed, these ore deposits are considered low-grade, requiring large-scale mining methods (e.g., open pit) (Press and Siever 1978).

Porphyry copper deposits and their associated skarn, hydrothermal veins, and replacement breccia deposits were the predominant class of deposits mined in 1989 (59 percent of total world mining). These deposits made up 93 percent of the United States copper mine capacity in 1990 (U.S. DOI, Bureau of Mines 1992). The largest mines of this type in the United States are the Morenci Mine, Arizona, and the Bingham Canyon Mine, Utah.

1.3.2 Sedimentary and Metasedimentary Deposits

Sedimentary copper deposits generally occur in rocks formed in passive continental margin and interior environments and along intracontinental rift systems. These types of copper ores are chemical precipitates formed from copper-bearing hydrothermal brines that percolate through the sediments, or minerals that are redistributed by later metamorphic activity. Stratiform sedimentary and metasedimentary deposits are an important source of copper, making up 24 percent of the worldwide copper mining activity. Mining of this class of deposits in the United States represented 6 percent of the copper mining capacity in 1990 (U.S. DOI, Bureau of Mines 1992).

1.3.3 Volcanogenic Massive Sulfide Deposits

Copper deposits found in ultramafic sequences were probably generated at ocean plate spreading centers. Because copper-bearing massive sulfides are associated with the submarine volcanic activity in these tectonic settings, deposits are commonly found in ophiolite rock formations. While volcanogenic and vein copper deposits are more numerous than porphyry- and sedimentary-type deposits, they are generally smaller in both capacity and reserves. Volcanogenic deposits made up 7 percent of the worldwide copper mining activity in 1989 (U.S. DOI, Bureau of Mines 1992).

1.3.4 Veins and Replacement Deposits

Copper found in veins was deposited along rock joints, fractures, faults, bedding planes, or other zones of structural weakness through which the mineral-bearing hydrothermal solutions were able to percolate. Vein deposit morphology is typically tabular, with varying degrees of uniformity in thickness. Replacement deposits result when relatively low temperature ore-depositing fluids dissolve the mineral in place and an equal volume of new crystal is formed (Press and Siever 1978). Vein and replacement-type copper deposits made up 7 percent of the worldwide copper mining activity in 1989 (U.S. DOI, Bureau of Mines 1992).

1.3.5 Ultrabasic and Carbonatite Deposits

Copper-bearing alkaline ultrabasic rocks and carbonatites intrude stable continental cratons, presumably related to mantle-derived magmas associated with intracontinental hotspots. Ultrabasic and carbonatite copper deposits made up 4 percent of the worldwide copper mining activity in 1989 (U.S. DOI, Bureau of Mines 1992).

1.3.6 Mineral Assemblages

Copper occurs in about 250 minerals; however, only a few of these are commercially important. The mineral assemblage of a copper deposit is the result of reactions between hydrothermal solutions and the host rock, influenced by wall rock chemistry, solution chemistry, temperature, and pressure. Most copper ores contain some amount of sulfur-bearing minerals. The weathering environment affecting the ore body following deposition is determined mainly by the availability of oxygen. Ores exposed to air tend to be oxidized, while those in oxygen-poor environments remain as sulfides.

The most common sulfide minerals are chalcopyrite ($CuFeS_2$), covellite (CuS), chalcocite (Cu_2S), bornite (Cu_5FeS_4), enargite (Cu_3AsS_4), and tetrahedrite ($(CuFe)_{12}Sb_4S_{13}$). Predominant oxide minerals are chrysocolla ($CuSiO_3$), malachite (Cu_2CO_3), azurite ($Cu_3(CO_3)_2(OH)_2$), and cuprite (Cu_2O). Chalcopyrite is the most common mineral found in porphyry-type deposits. Chalcocite occurs predominantly in hydrothermal veins (U.S. DOI, Geological Survey 1973).

1.4 COPPER EXTRACTION AND BENEFICIATION PRACTICES

1.4.1 Extraction Operations

1.4.1.1 Typical Mining Operations

Extraction is the operation of physically removing ore from deposits in the earth. There are three basic methods of extracting copper ore: surface, underground, and solution mining. Surface and underground mines usually operate independently of each other, although underground techniques are sometimes used before and/or after surface methods. Some open-pit surface operations extract massive sulfide deposits and intersect abandoned underground workings that were closed due to the low grade (or lack) of oxide and sulfide ore.

Open-pit mining is the predominant method used today by the copper mining industry. This is due primarily to inherently high production rates, relative safety, low costs, and flexibility in extraction. According to the U.S. Bureau of Mines (1993b) open-pit mines represent 83 percent of domestic mining capacity. The remaining 17 percent of the active copper mines use various types of high-tonnage underground operations. Underground mining operations are used to mine deeper, and richer ore bodies. Factors influencing the choice of mining method include the size, shape, dip, continuity, depth, and grade of the ore body; topography; tonnage; ore reserves; and geographic location.

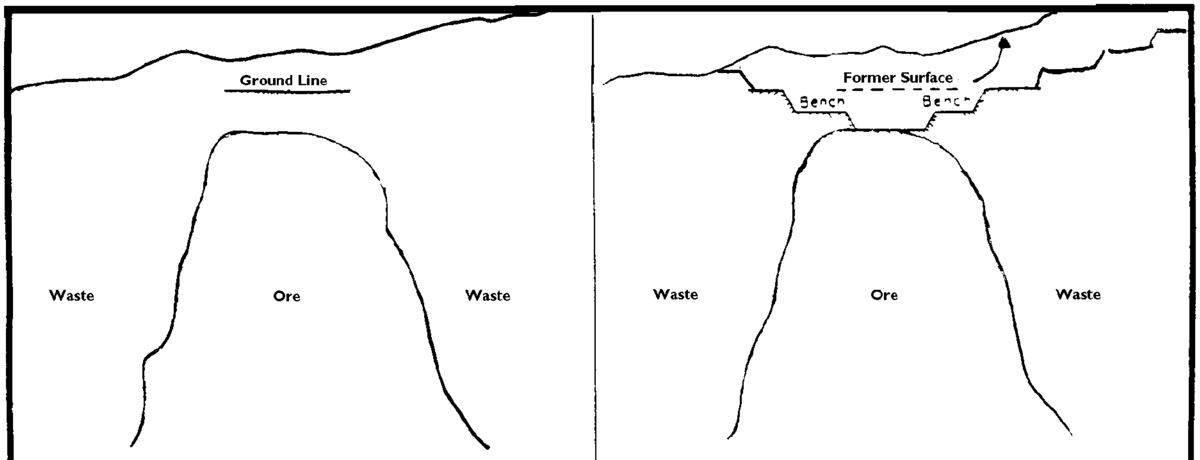
Solution mining of copper oxide and sulfide ores has increased since 1975. In this method, dilute sulfuric acid is percolated through ore contained in dumps, on leach pads, or underground leaching of broken rubble in or around formerly active stopes. Experimental work on *in situ* leaching, where the ore is leached in place, is also being conducted. The copper-bearing pregnant leach solution (PLS) is collected, and copper is recovered by SX/EW or precipitation methods. Solution mining has enabled facilities to beneficiate lower-grade sulfide and oxide ores.

1.4.1.2 Surface Mining Methods

As indicated above, most copper is produced by surface mining methods. Surface mining involves the excavation of ore from the surface by removing overburden (nonmineralized soil and rock that cover an ore body) and waste rock (poorly mineralized or very low-grade soil and rock that are within the ore body or surrounding it) to expose higher-grade minerals. In general, overburden is removed as efficiently and rapidly as possible, usually with little comminution. Overburden piles compose the largest volume of wastes generated by surface extraction activities (Beard 1990).

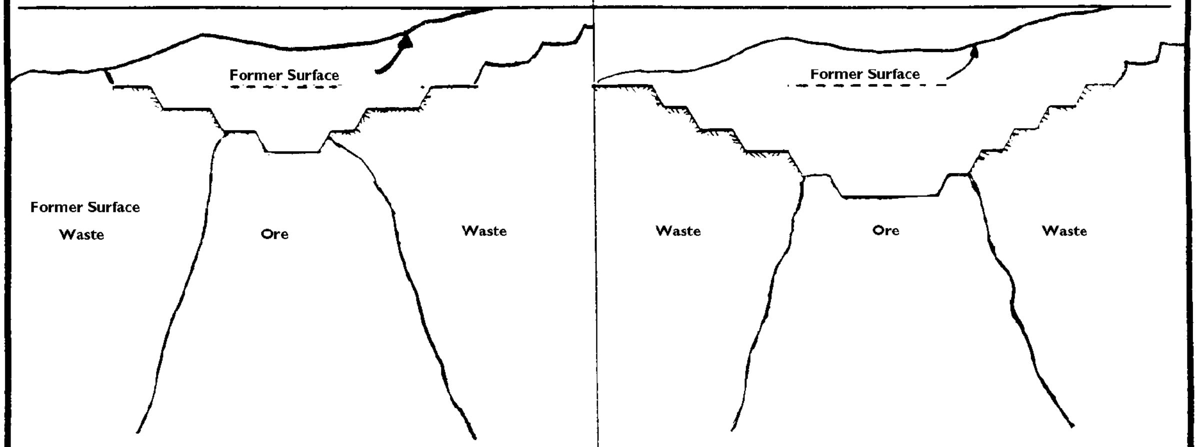
Advantages of surface mining operations, as compared to underground operations, include flexibility in production rates without deterioration of workings, relative safety for workers, ability to practice selective mining and grade control, and low cost per ton of ore recovered. Surface mining also has lower development and maintenance costs than underground mining because it requires fewer specialized systems. During expanded development, however, some surface mines with large amounts of prestripping waste could have higher costs than established underground mines.

Open-pit mining is most common in the copper mining industry because the ore body being mined is large and the overburden depth is usually limited. Open-pit mine designs are based on the configuration of the ore body, the competence of the rock, and other factors. The mine shape is formed by a series of benches or terraces arranged in a spiral or in levels with interconnecting ramps. Open-pit mines may reach several thousand feet below the surface. The different stages of open-pit mining are depicted in Figure 1-1



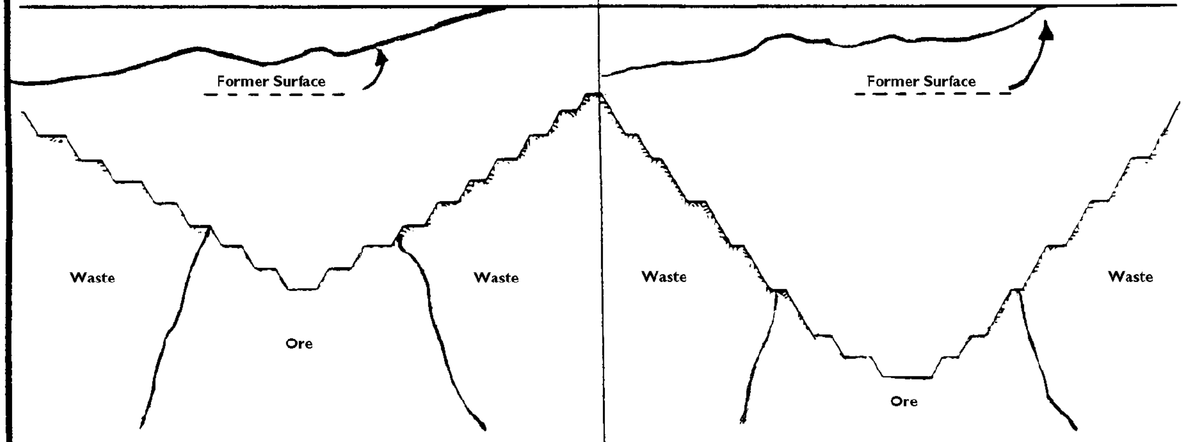
A. Section through an ore body before open-pit operations start

B. Section through ore body just as ore is exposed



C. Section through ore body as ore production commences

D. Section through ore body as waste stripping and ore production proceed



E. Section through ore body as mining continues

F. Section through ore body an economic limit of pit operations is reached

Figure 1-1. States of Open-Pit Mining

(Source: Stout 1980)

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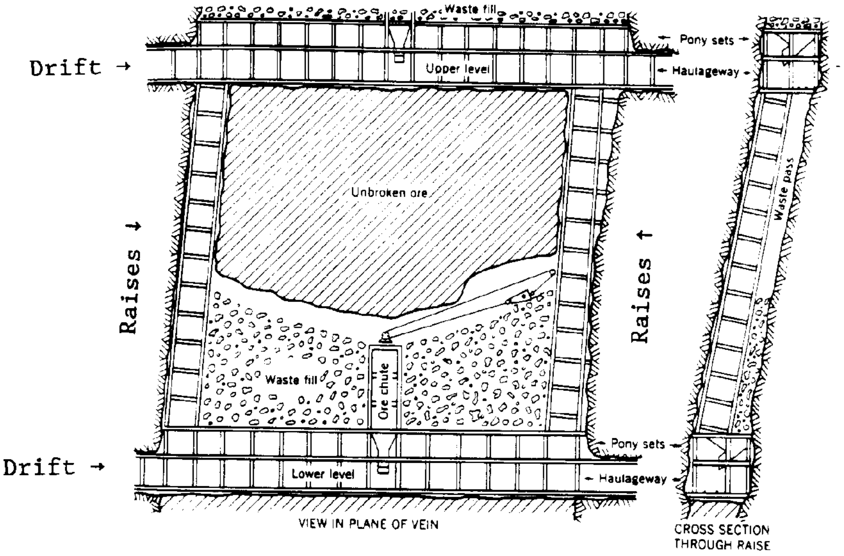
In the development stage, overburden is stripped off to expose the ore. The waste and ore are excavated by drilling rows of 6- to 12-inch (diameter) blast holes. Samples from the blast holes are analyzed to determine the grade. The blast holes are filled with a mixture of ammonium nitrate and fuel oil (ANFO) type explosive. Most mining operations use nonelectric caps and delays to control the blasting sequence. Usually, an entire segment of a bench is "shot" at one time. Subsequently, large electric or diesel shovels or front-end loaders transport the ore onto trucks, trains, or conveyor belts for removal to milling or leaching facilities, depending on the type of ore (sulfide or oxide) and grade. A pneumatic or hydraulic impact hammer, similar to a jackhammer, is used to break up waste and ore too large to handle in the pit or in subsequent crushing operations.

1.4.1.3 Underground Mining Methods

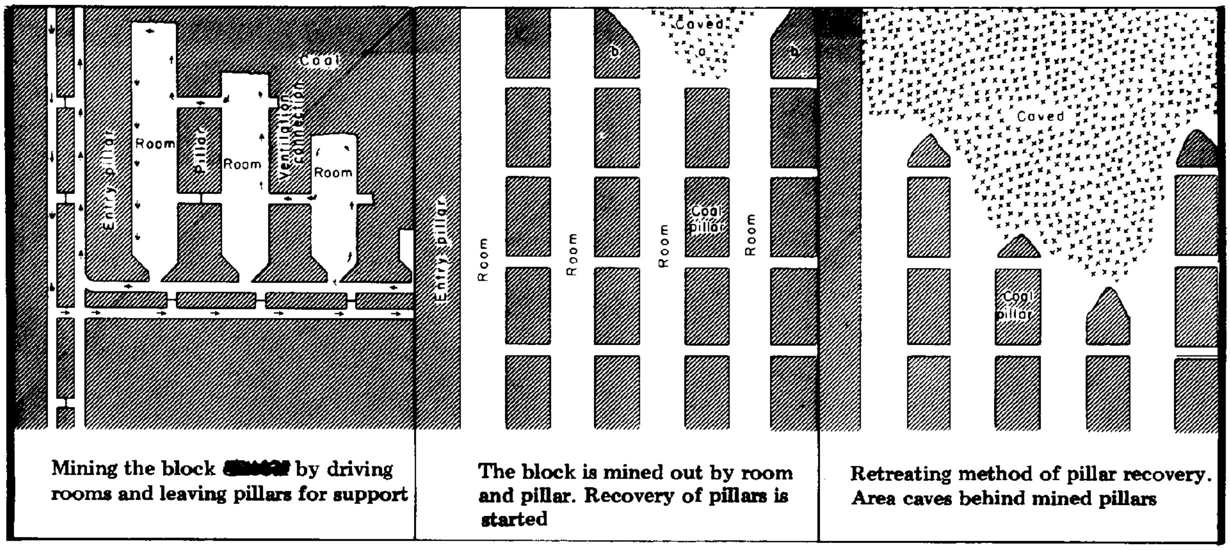
Underground mining methods are usually employed to mine richer, deeper, and smaller ore bodies where open-pit methods would be impractical. Underground mining operations are complex combinations of tunneling, rock support, ventilation, electrical systems, water control, and hoists for the transportation of people, ore, and materials. The three main underground mining methods used to mine copper ore are stoping, room-and-pillar, and block caving. All of these methods can be used in several variations, depending on the characteristics of the ore body.

Common stoping methods include cut-and-fill (see Figure 1-2a), square-set (timbered) stoping, open stoping, shrinkage stoping, sublevel stoping, and other variations. In general, all these underground operations involve sinking a vertical shaft or driving a horizontal adit, both of which provide access to the ore body. This type of extraction technique is best adapted to steeply dipping vein-type deposits. Today, underground operations using stoping methods are usually byproduct producers of copper and have relatively low copper tonnages (Stout 1980).

Most underground stope mines are designed with two or more shafts and a series of parallel drifts, known as levels, which intersect the main shaft. Ore mining occurs in areas between adjacent levels in irregular cavities called stopes (see Figure 1-2a). The stopes are connected to the levels by tow raises (one on each side of the block of ore to be mined), manways (to provide access), and chutes (to remove the ore). The ore is drilled and blasted at the face of the stope, then raked (or mucked) down



a. Cut-and-Fill Operation



Mining the block [redacted] by driving rooms and leaving pillars for support

The block is mined out by room and pillar. Recovery of pillars is started

Retreating method of pillar recovery. Area caves behind mined pillars

b. Room-and-Pillar Operation

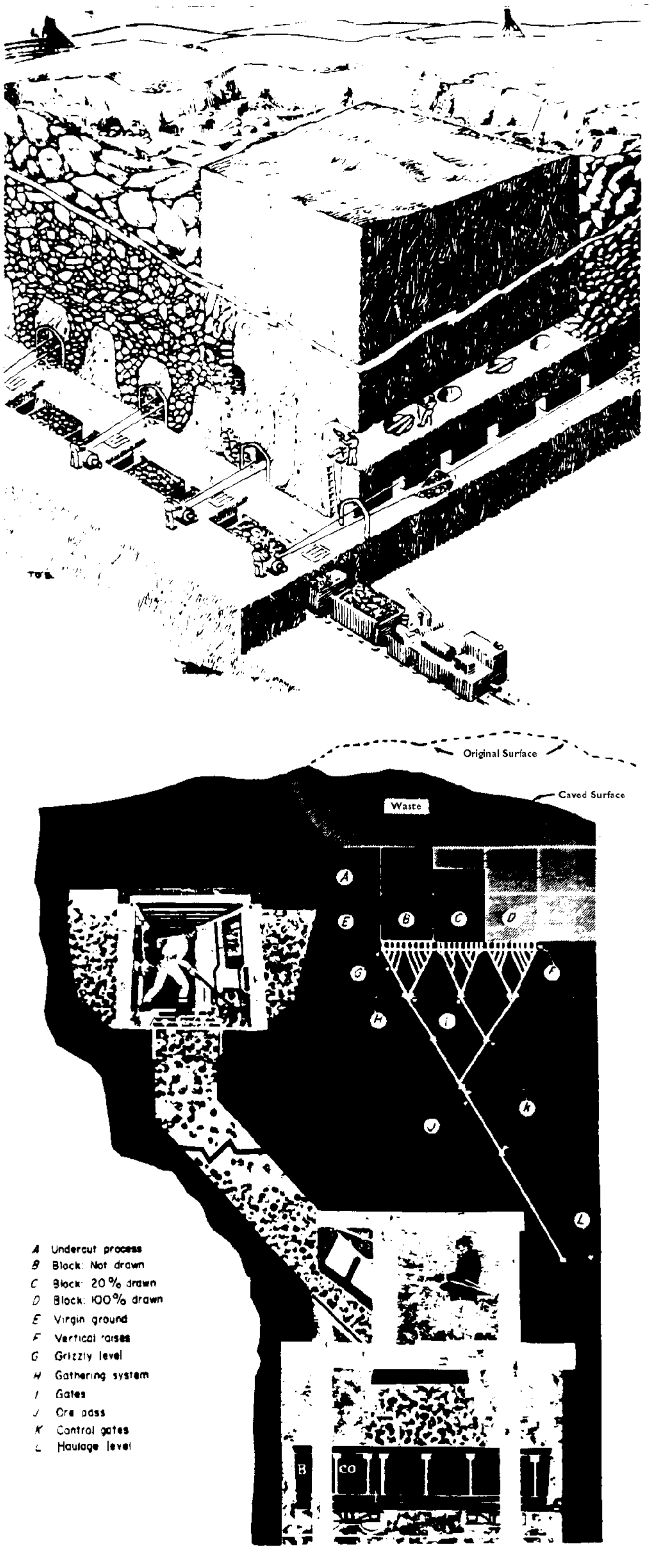
Figure1-2. Cut-and-Fill and Room-and-Pillar Underground Mining Methods

(Source: U.S. DOI, Bureau of Mines 1965a; Stout 1980) a chute. The chutes are located above the main haulage drifts and intercept them. The ore is loaded onto rail or rubber tire ore cars that haul it to the shaft. It is then dumped into another chute that feeds the ore into buckets that are hoisted to the surface. Waste rock, known as mine development rock (material removed to access the ore body), is handled the same way, except that it is hauled to an adjacent stope. There, it is dumped into a raise that feeds into a stope where it is backfilled to provide a working area to drill out the next ore cut (Stout 1980).

Room-and-pillar mining operations produce more tonnage than any other type of mine operation. Room-and-pillar operations are best adapted to mining large, flat deposits or massive deposits where sequential slices or levels may be mined. Mining is conducted in a nearly horizontal or horizontal altitude. Depending on the access design for the deposit, vertical shafts or relatively horizontal inclines or declines may be used. A double entry system is designed to provide ventilation, men and materials access, and ore transport (Stout 1980).

A typical room-and-pillar operation is illustrated in Figure 1-2b. Usually, ore is mined in two phases, the first phase involves driving large horizontal drifts (called rooms) parallel to each other and smaller drifts perpendicular to the rooms. The area between the intersection of the rooms and drifts forms the pillars, which support the roof. Rooms vary in size from 6 to 60 feet high and 10 to 100 feet wide. The size of each room and pillar is dependent on the quality of the rock. Between 30 and 60 percent of the ore remains unmined in the pillars. Once the mine reaches the end of the ore body, the second phase of operations may begin to recover the ore left behind in the pillars. Starting from the back of the mine and working forward, the pillars are mined out one at a time, a technique called "pillar robbing." Timbers are used to temporarily support the roof. Once a pillar is mined out, the timbers are removed and the ground is allowed to collapse. This procedure is called "retreating" and produces ore at a relatively low cost per ton (Stout 1980).

Block caving (depicted in Figure 1-3



(Source: Stout 1980)

Figure 1-3. Block-Caving Methods

(Source: Stout 1980)

) is a third large-tonnage underground mining method used to mine copper. This method includes undercut block and sublevel block caving. The block-caving method of mine development utilizes the natural forces of gravity to cause the ore to break on its own accord without being drilled and blasted. A typical block-caving mine is developed by first driving a series of parallel haulage drifts below the ore body. From the haulage drifts, a series of raises are driven at a 45-degree angle forming the grizzly level. A second set of finger raises are driven perpendicular to the inclined grizzlies. The grizzlies and finger raises are spaced at suitable intervals to produce effective caving. The ends of the finger raises are star drilled or ring drilled with a series of drillholes radiating out from the raise and blasted together. This creates the cavities that start the caving process (Stout 1980).

The caving action is caused by the ore caving under its own weight into a large cavity in the stopes. Because the broken ore takes up more volume than the solid ore, the stope fills up; this, in turn, stops the caving process. As the broken ore is mucked or slushed (i.e., pulled) from the back of the stope by drawing ore from the raises, a cavity is created which restarts the caving process. The more rapid the withdrawal rate, the more rapid the caving action. Consequently, raises must be "pulled or mucked" evenly to ensure uniform caving (Stout 1980).

1.4.2 Beneficiation Operations

Beneficiation of ores and minerals is defined in 40 CFR 261.4 as including the following activities: crushing; grinding; washing; filtration; sorting; sizing; gravity concentration; flotation; ion exchange; solvent extraction; electrowinning; precipitation; amalgamation; roasting; autoclaving; chlorination; and heap, dump, tank, and *in situ* leaching. The beneficiation method(s) selected varies with mining operations and depends on ore characteristics and economic considerations.

1.4.2.1 Conventional Milling/Flotation

This section describes the typical stages in the conventional milling/flotation of sulfide ores. A flowsheet illustrating this process is presented in Figure 1-4.

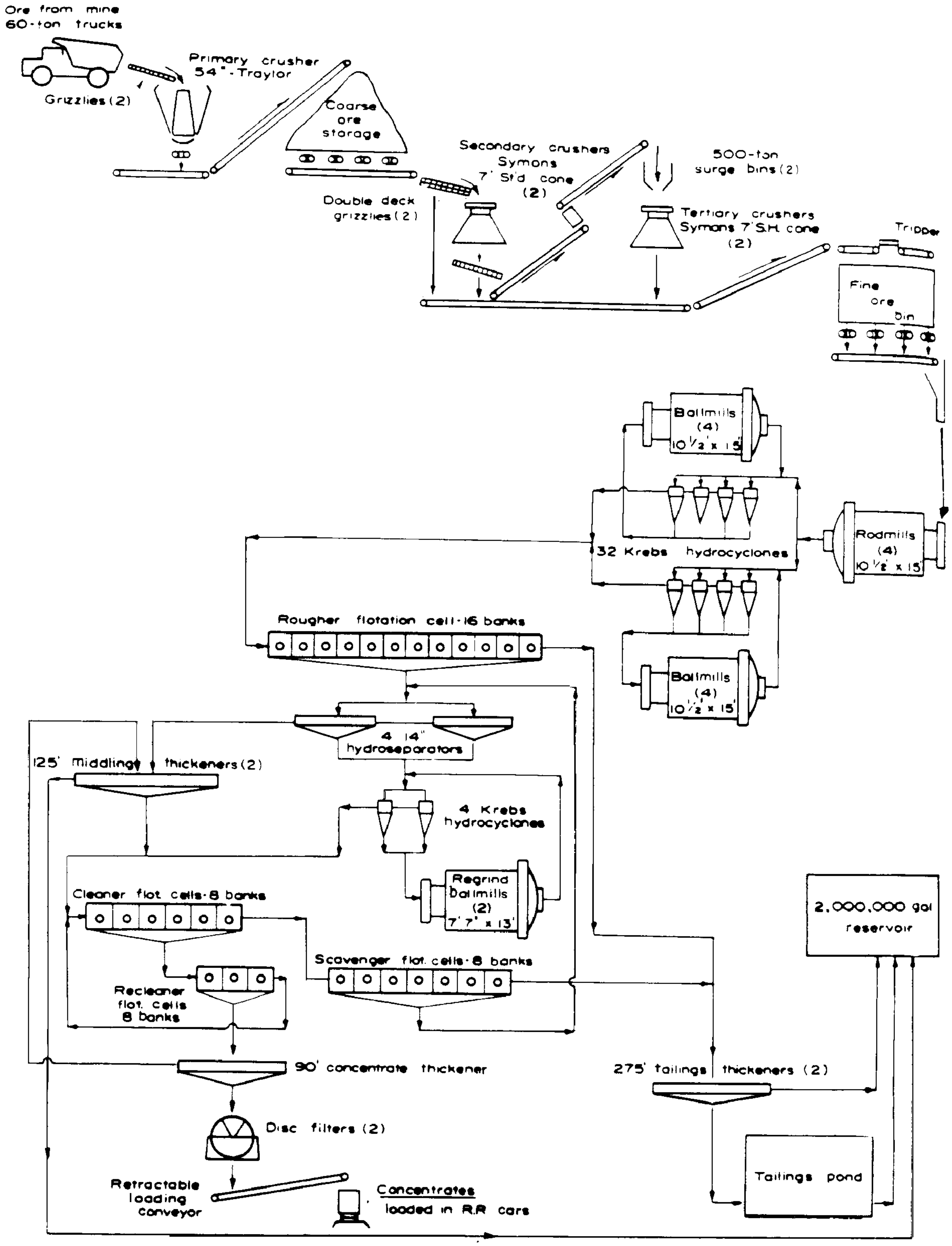
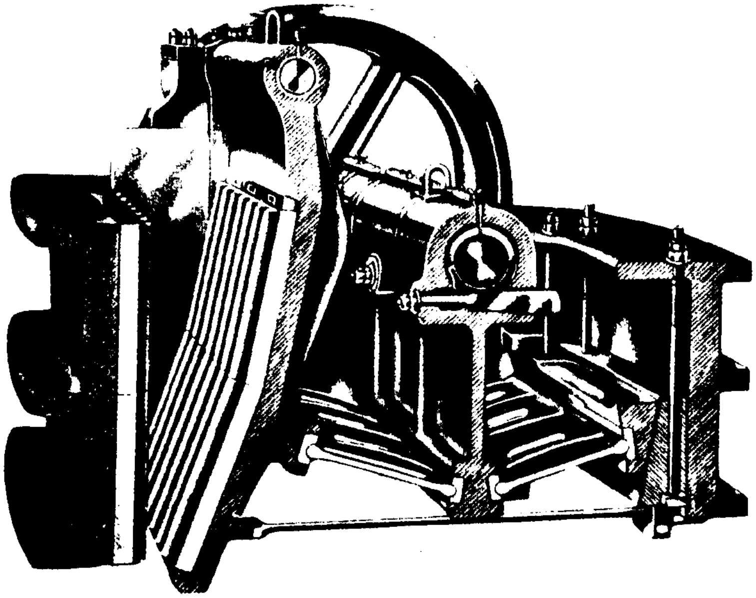


Figure 1-4. Flowsheet for Sulfide Ore Beneficiation

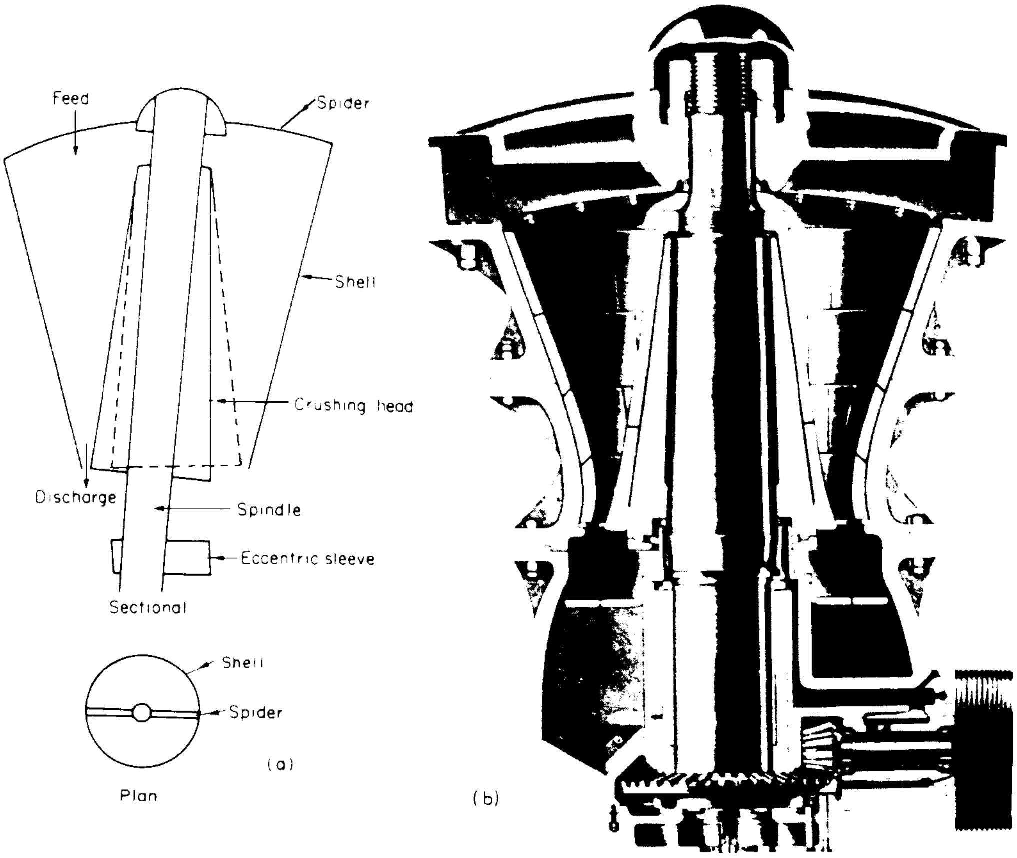
(Source: U.S. DOI, Bureau of Mines 1965a)

Crushing and Grinding (Comminution)

The first step in beneficiation is comminution. Typically, this is accomplished by sequential size reduction operations—commonly referred to as crushing and grinding. Crushing may be performed in two or three stages. Primary crushing systems consist of crushers, feeders, dust control systems, and conveyors used to transport ore to coarse ore storage. Primary crushing is often accomplished by a jaw or gyratory crusher, since these units can handle larger rocks. Figure 1-5a



a. Cross-Section of a Jaw-Type Crusher



b. Functional Diagram and Cross-Section of a Cone-Type Crusher

Figure 1-5. Typical Crushers

(Source: Wills 1981)

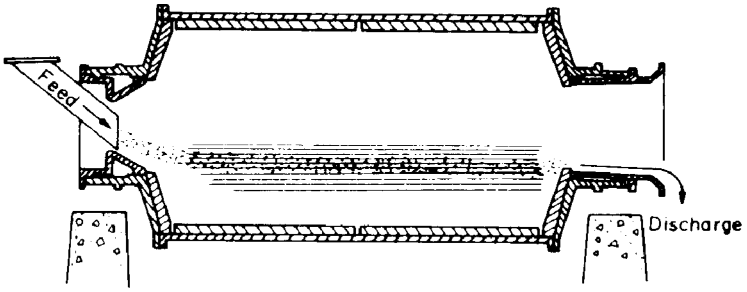
shows a typical jaw-type crusher. Cone crushers, shown in Figure 1-5b, work best at large, high-capacity operations because they can handle larger tonnages of material. The feed to primary crushing is generally run-of-mine ore, which is reduced from large pieces (2 to 4 feet in dimension) to smaller pieces (8 to 10 inches in dimension). Primary crushing systems are typically located near or in the pit at surface mines or below the surface in underground mines. Crushed ore is then transferred to secondary crushers, usually located near the next step in beneficiation. The ore may be temporarily stored in piles at the site.

Secondary and tertiary crushing usually are performed in surface facilities in cone crushers, although roll crushing or hammer mills are sometimes used. In these reduction stages, ore must be reduced to about 0.75 inches before being transported (usually on conveyer belts) to a grinding mill (U.S. Congress, Office of Technology Assessment 1988; Taggart 1945; Wills 1981).

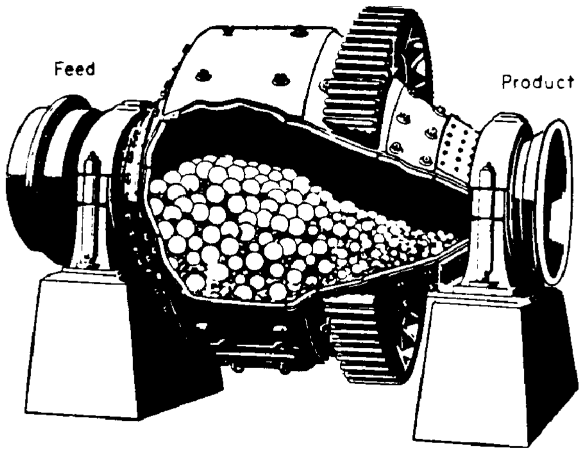
Size separators (such as grizzlies and screens) control the size of the feed material between the crushing and grinding stages. Grizzlies are typically used for very coarse material. Screens mechanically separate ore sizes using a slotted or mesh surface that acts as a "go/no go" gauge. Vibrating and shaker screens are the most commonly used types of separators. There are many different types of vibrating screens, designed to handle material between 25 centimeters (cm) and 5 mm. After the final screening, water is added to the crushed ore to form a slurry (U.S. Congress, Office of Technology Assessment 1988; Taggart 1945; Wills 1981).

Grinding is the last stage in comminution. In this operation, ore particles are reduced and classified (typically in a hydrocyclone) into a uniformly sorted material between 20 and 200 mesh. Most copper facilities use a combination of rod and ball mills to grind sulfide ore (Figures 1-6a and b). Rod mills use free steel rods in the rotating drum to grind the ore. A ball mill works by tumbling the ore against free steel balls and the lining of the mill. Rod and ball mills are constructed with replaceable liners composed of high-strength chrome-molybdenum steel bolted onto the mill shell. The grinding face of the liner is ribbed to promote mixing. The liners require extensive maintenance and must be replaced regularly. To replace the liner, the mill must be taken out of production. A shutdown of a mill requires additional milling capacity to prevent overall mill shutdowns during maintenance (U.S. Congress, Office of Technology Assessment 1988; Taggart 1945; Wills 1981). In some cases, ore and water are fed into an autogenous mill (where the grinding media are the hard ores themselves), or a semiautogenous mill (where the grinding media are the ore supplemented by large steel balls).

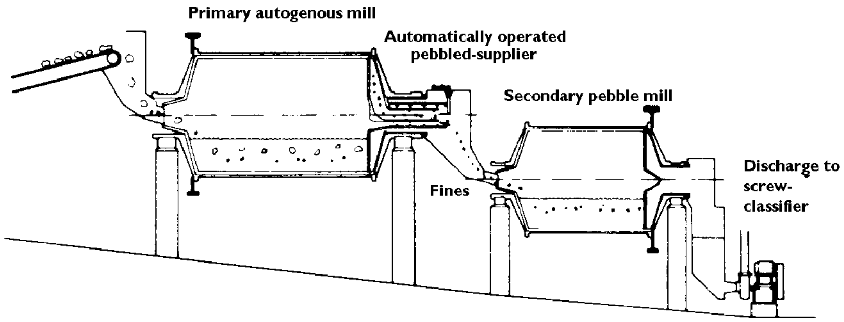
Typically, grinding circuits are organized in series configuration as shown in Figure 1-6c.



a. Rod Mill



b. Ball Mill



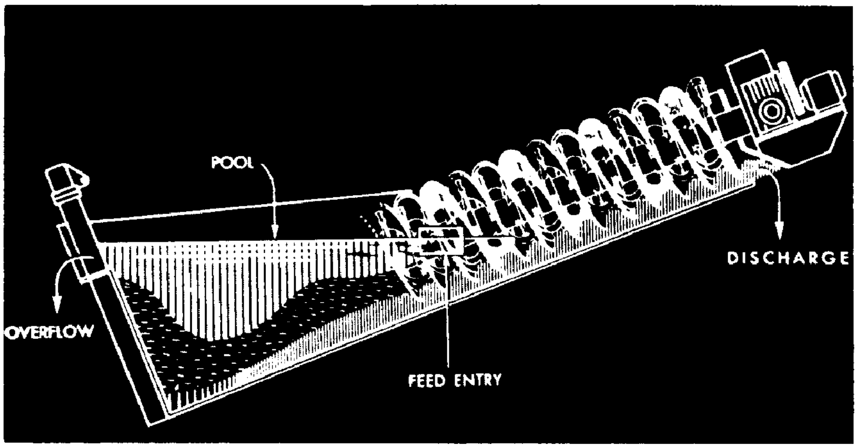
c. Grinding Circuit

Figure 1-6. Typical Milling Units

(Source: Wills 1981)

Each unit in the series produces successively smaller material. Typically, crushed ore and water enter the rod mill. When the material is reduced to a certain particle size, it becomes suspended in the slurry (because of its size and specific gravity and the motion of the mill). The fine material then floats out in the overflow from the mill (U.S. Congress, Office of Technology Assessment 1988; Taggart 1945; Wills 1981). At this point, the ore slurry is classified according to particle size in a hydrocyclone or similar device. Oversize material passes to the ball mill for additional grinding. Undersize material moves to the next phase of beneficiation.

After grinding, ore is pumped to a classifier designed to separate fine-grained material (less than 5 mm) from coarse-grained material requiring further grinding. This method is used to control both under and over milling or grinding. Classification is based on differences in the size, shape, density, and settling rate of particles in a liquid medium (i.e., water). Various kinds of hydraulic classifiers are used. These generally fall into two categories: horizontal, and vertical current classifiers. Mechanical classifiers (shown in Figure 1-7a)



a. Mechanical Classifier (Old Technology)

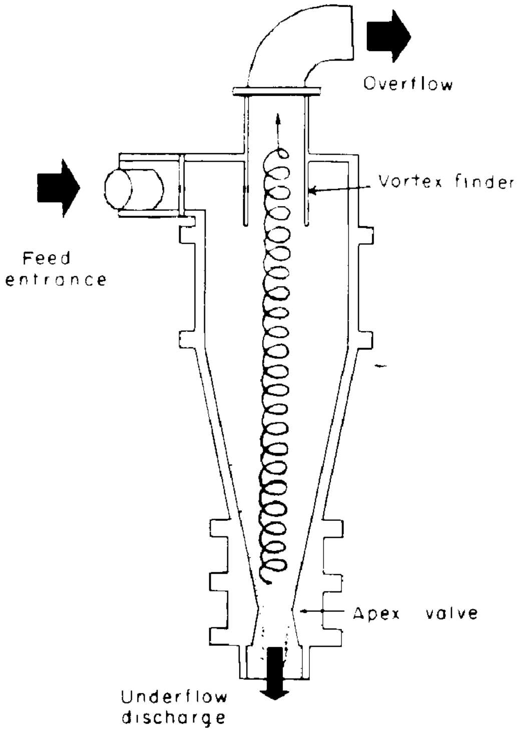


Figure 1-7. Classifiers

(Source: Wills 1981)

are horizontal current classifiers, which are no longer in wide use. The hydrocyclone (see Figure 1-7b) is the standard technology for vertical classifiers in use today (U.S. Congress, Office of Technology Assessment 1988; Taggart 1945; Wills 1981).

Flotation

The second step in the beneficiation of sulfide ore is concentration. The purpose of concentration is to separate the valuable mineral (or "values") from nonvaluable minerals (referred to as "gangue"). There are a variety of concentration methods. Selection of a method (or methods) to use for a particular ore is based on the ore mineralogy and mineral liberation size. Froth flotation is the standard method of concentration used in the copper industry. About 75 percent of all copper is produced by this method. The most significant technological development in flotation in recent years is the column flotation cell, which is being installed at most concentrators (Berkeley Study 1985).

One of the advantages to the flotation method is that it makes the recovery of molybdenum [as molybdenite (MoS_2)] by selective flotation viable at some properties. The recovery of molybdenite, when the molybdenum price is adequate, can provide a significant portion of a mine's revenue. In addition to the byproduct of molybdenum, most of the precious metals in the copper concentrate are recovered in anode slimes during subsequent electrorefining steps. As of 1985, there were eight copper and seven copper-molybdenite froth-flotation-type concentrators in the United States (Berkeley Study 1985; U.S. DOI, Bureau of Mines 1987a; U.S. Congress, Office of Technology Assessment 1988).

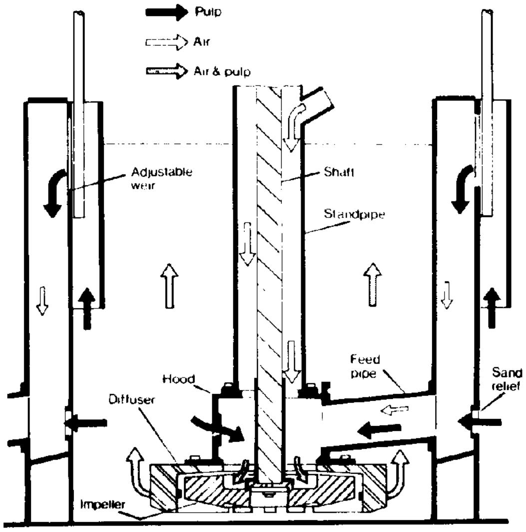
Currently, there are 11 copper flotation concentrators in operation in Arizona and New Mexico. ASARCO operates four (Ray, Mission, and two newly opened facilities), Cyprus operates two (Bagdad and Sierrita); Magma operates three (San Manuel, Pinto Valley, and Superior); and Phelps Dodge operates two (Tyrone and Chino).   Three other concentrators are on stand-by: ASARCO's Silver Bell facility, Cyprus' Esperanza facilities, and Phelps Dodge's Ajo facility.

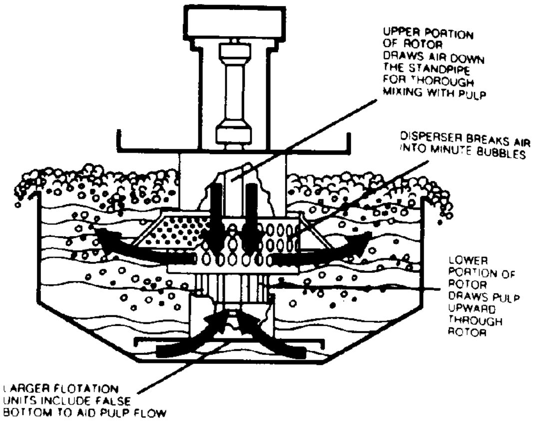
b. Hydrocyclone (New Technology)

Typical flotation cells are depicted in Figure 1-8

Denver D-R Flotation Cells

Denver Sub-Aeration Cell





Wemco Fagergen Cell

Figure 1-8. Types of Flotation Cells

(Source: Wills 1981; U.S. Congress, Office of Technology Assessment 1988)

. In general, they resemble a large washing machine that keeps the particles in suspension through agitation. The ore is first conditioned with chemicals to make the copper minerals water-repellent (i.e., hydrophobic) without affecting the other minerals. Air is then pumped through the agitated slurry to produce a bubbly froth. The hydrophobic copper minerals are aerophillic, that is they are attracted to air bubbles, to which they attach themselves, and then float to the top of the cell. As they reach the surface, the bubbles form a froth that overflows into a trough for collection. The minerals that sink to the bottom of the cell and are removed for disposal (Wills 1981).

The simplest froth flotation operation is the separation of sulfide minerals from gangue minerals (such as limestone or quartz). The separation of different types of sulfide minerals, such as chalcopyrite, from pyrite is more complex, because the surfaces of the minerals have to be modified so that the reagents attach only to the mineral to be floated. In practice, each ore is unique; consequently, there is no standard flotation procedure. Once the unit is operational, continued monitoring of the ore feed mineralogy is critical to fine-tune the flotation units when changes occur. These changes occur because ore bodies are not homogeneous; variations in feed and mineralogy are normal and may require circuit modifications (Biswas and Davenport 1976; Taggart 1945; Wills 1981).

Conventional flotation is carried out in stages. The purpose of each stage depends on the types of minerals in the ore. Selective flotation of chalcocite-bearing sulfide ores and the rejection of pyrite utilizes three types of flotation cells: roughers, cleaners, and scavengers. Roughers use a moderate separating force to float incoming ore and to produce a medium-grade concentrate. Cleaners use a low separating force to upgrade the rougher concentrate by removing additional pyrite and gangue waste material to produce a high-grade concentrate. Scavengers provide a final, strong flotation treatment for the rougher tailings by using a strong concentration of reagents and vigorous flotation to recover as much of the remaining sulfide minerals as possible. The float from the scavenger flotation is often recycled through a regrinding mill and sent back to the rougher flotation cells. Throughout the operation, the pyrite is depressed by employing a modifying agent, such as lime, for pH control (U.S. Congress, Office of Technology Assessment 1988; Biswas and Davenport 1976; Taggart 1945; Wills 1981).

Because flotation is partially dependent on ore particle size, regrinding of the particles between the rougher and cleaner flotation cells may be needed. Tailings from the cleaner flotation may be sent back to the flotation circuit for additional recovery (Biswas and Davenport 1976; U.S. DOI, Bureau of Mines 1965a; Wills 1981).

For more complex ores, the first stage of flotation is often a bulk float. This is similar to the rougher stage, in which much of the waste and some of the byproduct metals are eliminated. The bulk concentrate goes to roughers (which float specific types of sulfides) and then to cleaners. Again, a regrinding circuit may be needed between rougher cells (Biswas and Davenport 1976; U.S. DOI, Bureau of Mines 1965a; Wills 1981).

Froth flotation is carried out using reagents that, when dissolved in water, create hydrophobic forces that cause the values to float. Reagents can be added prior to entering the initial rougher flotation stage and/or during subsequent steps in the flotation operation. The reagents used in flotation concentrators are called collectors, depressants, activators, frothers, flocculants, filtering aides, and pH regulators. A complete list of the reagents typically used in a copper flotation circuit is presented in Table 1-3 (Biswas and Davenport 1976; U.S. DOI, Bureau of Mines 1987a).

Table 1-3. Reagent Consumption at U.S. Copper Sulfide Concentrators in 1985

Copper Froth Flotation		**Molybdenum Copper**	
	Amount*		**Amount***
Flotation Reagents		**Flotation Reagents**	
Collectors		Collectors	
Ethylxanthate	632	Ethylxanthate	417
Amylxanthate	307	Amylxanthate	261
Isopropylxanthate	154	Isopropylxanthate	70
Alkyl dithiophosphate	629	Isobutylxanthate	123
Thionocarbamate	146	Unspecified xanthates	224
Mixtures of thio reagents	338	Alkyl dithiophosphate	405
Unspecified thio reagents	26	Unspecified dithiophosphate	62
TOTAL	**2,232**	Xanthogen formate	48
Depressants		Thionocarbamate	709
Cyanide salt	5	Unspecified sulfide collector	765
Frothers		Fuel oil	2,207
Aliphatic alcohol	1,044	Kerosene	55
Pine oil	271	**TOTAL**	**5,346**
Polyglycol ether	20	Depressants	
Unspecified polyol	1,566	Phosphorous pentasulfide	1,926
TOTAL	**2,901**	Cyanide salt	3,652
Flocculants		Sulfide salt	19,649
Aluminum salts	155	Sodium silicate	102
Anionic polyacrylamide	74	**TOTAL**	**25,329**
Nonionic polyacrylamide	111	Activators	
Unspecified polymer	113	Sodium sulfide or hydrosulfide	**14,613**
TOTAL	**453**	pH Regulators	
		Sulfuric acid	2,203
		Caustic soda (NaOH)	3
		TOTAL	**2,206**
		Frothers	
		Aliphatic alcohol	2,936
		Pine oil	227
		Phenol	777
		Polyglycol ether	219
		Unspecified polyol	587
		TOTAL	**4,746**
		Flocculants	
		Anionic polyacrylamide	157
		Nonionic polyacrylamide	52
		Polyacrylate	374
		Unspecified polymer	66
		TOTAL	**649**
		Dispersants	
		Sodium silicate	51
		Polyphosphate	273
		TOTAL	**324**
TOTAL	**5,591**	**TOTAL**	**53,213**
pH Regulators		**pH Regulators**	
Lime	224,268	Lime	357,129

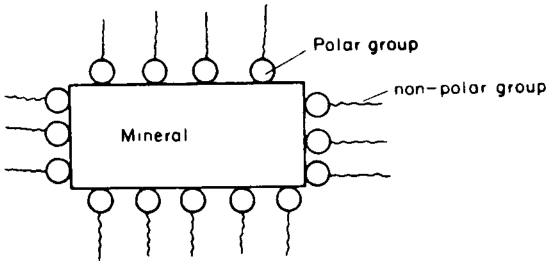
*Quantity in thousands of pounds

(Source: U.S. DOI, Bureau of Mines 1987a)

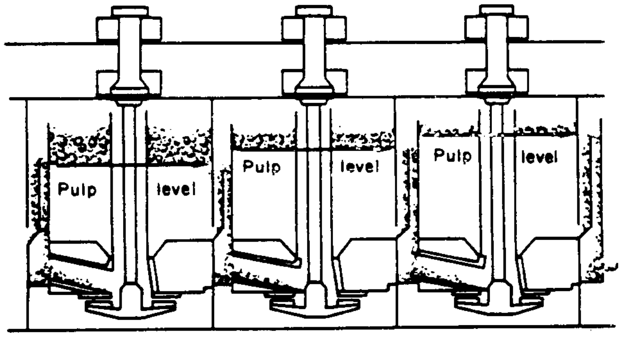
Ionic collectors are added to the ore slurry to create the hydrophobic surfaces on sulfide minerals. The best-known sulfide collectors are potassium and sodium xanthates. Other types of collectors are thionocarbomates, dithiophosphates, and thiocarbanilides. Kerosene and fuel oil are used as molybdenite collectors. The longer carbon-chain potassium amyl xanthate typically is used as a collector in scavenger flotation cells to promote flotation of difficult-to-float, partially oxidized sulfate-filmed copper minerals (Biswas and Davenport 1976; U.S. DOI, Bureau of Mines 1987a).

A copper collector typically is composed of a complex heteropolar molecule, which has a charged (i.e., negative) sulfur-bearing polar group end and a noncharged nonpolar group end. The nonpolar radical is a hydrocarbon that has pronounced water-repellant properties, whereas the polar group reacts with water and the copper mineral surface (see Figure 1-9a). The reaction between sulfide minerals and sulfide collectors (such as xanthates) results in insoluble metal xanthates that are strongly hydrophobic. The copper sulfide mineral becomes a surface covered with air-avid hydrocarbon nonpolar ends seeking an air bubble attachment (Wills 1981).

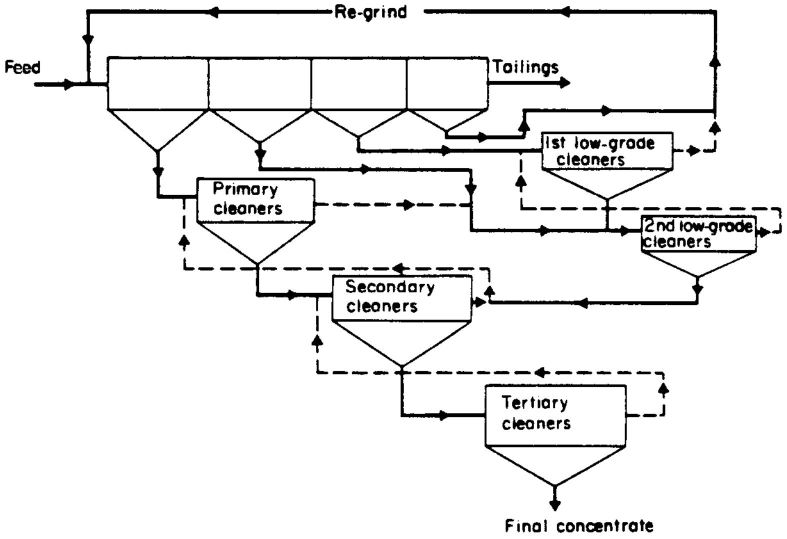
Mechanical flotation cells (see Figure 1-9b



a. Collector Adsorption on Mineral Surface



b. Bank of Flotation Cells



c. Complex Flotation Circuit

Figure 1-9. Flotation

(Source: Wills 1981)

) introduce air into the slurry, creating dispersed bubbles to which the hydrophobic complexes attach (and on which they then float to the surface). Frothers are chemically similar to ionic collectors; they absorb on the air-water interface and reduce the surface tension, thus stabilizing the bubbles. The resultant froth must be short-lived and self-deteriorating or the flotation units would be enveloped in foam. Standard frothing agents used in copper and copper-molybdenite concentrators include alcohols, pine oil, and polyglycol ethers (Biswas and Davenport 1976; U.S. DOI, Bureau of Mines 1987a).

Differential flotation for complex ores that contain sulfides (other than copper sulfides) requires the use of reagents that modify the action of the collector either by intensifying or reducing its water-repellant effect on the valuable mineral surface (Figure 1-9c). These reagents are known as modifiers or regulators or, in copper-molybdenite concentrators, as depressants and activators. The most common modifier is the OH (hydroxyl) ion. Lime or sodium carbonate is used to raise the pH of the slurry and regulate the pulp alkalinity. The second most common modifier in copper flotation is the cyanide ion derived from sodium cyanide. It is normally used to depress pyrite while floating chalcopyrite or chalcocite in rougher flotation. Standard activators used in the copper and copper-molybdenite flotation circuit for oxidized copper mineral surfaces are sodium sulfide and sodium hydrosulfide (Biswas and Davenport 1976; U.S. DOI, Bureau of Mines 1987a).

Copper mineral concentrate, the product of flotation, is then sent to a smelter for processing. The waste material or tailings from this operation is sent to a tailings pond for disposal. Copper concentrates exiting the flotation circuit contain 60 to 80 percent water. Dewatering the concentrate in a thickener, then in disc or drum filters for final dewatering, produces a relatively dry product ready for further shipping and processing. The collected water is usually recycled to the milling circuit. The settling of solids in the thickeners is enhanced by chemical reagents known as flocculants. Filter cake moisture is regulated by reagents known as filtering agents. Typical flocculants and filtering agents used are polymers, nonionic surfactants, polyacrylate, and anionic and nonionic polyacrylamides (ASARCO 1991).

At most facilities, thickening of tailings is a common step prior to pumping the thickened slurry to the tailings pond and ultimately disposing of the thickened slurry. Thickening minimizes the amount of water placed in the pond and the pond size. The thickened tailings retain sufficient water to allow them to flow in the tailings pipeline without undue wear on the transport system (Arizona BADCT Draft).

Thickening is usually accomplished by settling in large tanks, known as thickeners. The settling of solids in tailings thickeners is also enhanced with flocculants. Gravity causes the flocculated solids to settle to the bottom of the thickener, where they are scraped to a discharge outlet by a slowly rotating rake. Collected water from this process is generally recycled back to the mill to be used in beneficiation activities (U.S. Congress, Office of Technology Assessment 1988; Biswas and Davenport 1976; U.S. DOI, Bureau of Mines 1987a; Wills 1981).

1.4.2.2 Leach Operations (*In Situ*, Dump, Heap, and Vat)

Copper is increasingly recovered by solution, or hydrometallurgical, methods. These include dump, heap, and vat leaching techniques, as well as underground (or *in situ*) leaching methods. Each of these methods results in a pregnant leach solution (PLS). Copper is recovered from the PLS through precipitation or by solvent extraction/electrowinning (SX/EW) (U.S. Congress, Office of Technology Assessment 1988).

Characteristics of typical leaching operations are presented in Tables 1-4 and 1-5. Recent developments in SX/EW technology have made solution mining a major factor in copper production. Currently, solution copper mining techniques account for approximately 30 percent of domestic copper production. Two-thirds of all United States copper mines employ various types of solution operations. Solution mining generally has lower day-to-day operating costs than other mining methods (Weiss 1985).

Table 1-4. Characteristics of Copper Leaching Methods

	Vat Leaching	Heap Leaching	Dump Leaching	Underground and In situ Leaching
Ore grade	Moderate to high	Moderate to high	Low	Low to high (dependent upon mine conditions and layout)
Types of ore	Oxides, silicates, and some sulfides	Oxides, silicates, and some sulfides	Sulfides, silicates, and oxides	Oxides, silicates, and some sulfides
Ore preparation	May be crushed to optimize copper recovery	May be crushed to optimize copper recovery	Blasting	None
Container or pad	Large impervious vat	Impervious barrier of clay, synthetic material, or both	None for existing dumps; new dumps intended to be leached would be graded, and covered with an impermeable polyethylene membrane, or bedrock, protected by a layer of select fill	None
Solution	Sulfuric acid for oxides; acid cure and acid-ferric cure provide oxidant needed for mixed oxide/sulfide ores	Sulfuric acid for oxides; acid cure and acid-ferric cure provide oxidant needed for mixed oxide/sulfide ores	Acid ferric-sulfate solutions with good air circulation and bacterial activity for sulfides	Sulfuric acid, acid cure, acid-ferric cure, or acid ferric-sulfate, depending on the ore type
Length of leach cycle	Days to months	Days to months	Months to years	Months
Solution application method	Spraying, flooding, and circulation	Spraying or sprinkling	Ponding/flooding, spraying, sprinkling, and trickle systems	Injection holes, recovery holes
Metal recovery method	SX/EW for oxides and mixed oxide/sulfide ores; iron precipitation for mixed ores	SX/EW for oxides and mixed oxide/sulfide ores; iron precipitation for mixed ores	SX/EW for oxides and mixed oxide/sulfide ores; iron precipitation for mixed ores	SX/EW for oxides and mixed oxide/sulfide ores; iron precipitation for mixed ores

(Source: U.S. Congress, Office of Technology Assessment 1988)

Most ores occur as mineral compounds that are insoluble in water; leaching involves chemical reactions that convert copper into a water-soluble form followed by dissolution. The leaching reagent used by each operation is dependent on the mineralogical composition of the ore material. Several types of reagents are used to produce these chemical reactions, including acids and bacterium (Weiss 1985).

Acid leaching of ores and concentrates is the most common method of hydrometallurgical extraction. Its use is confined to acid-soluble, oxide-type ores that are not associated with acid-consuming rock types containing high concentrations of calcite (such as limestone and dolomite). Some ores require a form of concentration and/or pretreatment, such as roasting or calcification, before leaching. Typical acidic leaching agents include hydrochloric acid (HCL), sulfuric acid (H_2SO_4), and iron sulfate ($Fe_2(SO_4)$). Sulfuric and hydrochloric acid leaching at atmospheric pressure is the most common type of copper leaching. Copper minerals such as azurite, malachite, tenorite, and chrysocolla, are completely soluble in sulfuric acid at room temperature. Other, less oxidized, cuprite and sulfide ores, such as chalcocite, bornite, covelite, and chalcopyrite, require the addition of ferric sulfate and oxygen (as oxidants) to accomplish leaching. Leaching ores containing bornite and chalcopyrite with ferric sulfate is very slow, even at elevated temperatures (Weiss 1985).

Table 1-5. Background on Copper Leaching Methods

Leaching Method	Mineralization	Percent Copper in Ore	Sulfuric Acid Concentration in Leachate (kg/m^3)	Copper Concentration in Pregnant Solution (kg/m^3)	Leach Cycle	Representative Size of Operation	Copper Leached (metric tpd)
Dump	Sulfide or mixed oxide/sulfide	0.05+	1-5	1-2	3-30 years	5 x 10^6 metric tons of ore	100
Heap	Oxide	0.5-1	2-10	2-5	1+ months	3 x 10^5 metric tons of ore	20
Under-ground and *In Situ*	Oxide (with some sulfide)	0.5-1	3-50	3-10	1-25 years	4 x 10^6 metric tons of ore	20
Vat	Oxide	1-2	50-100	30-40	5-10 days	6-12 vats	10-120
Agitation	Oxide	0.05+	50-100	30-50	2-5 hours	45 leach tanks 47 thickeners	350

(Source: U.S. EPA 1989e)

For certain minerals, alkaline (or basic) leaching is an effective means of extracting copper. Alkaline leaching is more selective than acid leaching and particularly appropriate for ores with large amounts of acid-consuming carbonate rocks. This selectivity often results in lower recovery if the metals are not fully liberated in the comminution stage. Silica- and silicate-rich ores can be treated using alkaline leaching agents at raised temperatures. The principal reagents used in alkaline leaching are the hydroxides and carbonates of sodium and ammonia, but potassium hydroxide, calcium hydroxide, and sodium sulfide also are used. When leaching with ammonia (NH_3), ammonium carbonate [$NH_3/(NH_4)_2CO_3$] or ammonium sulfate $(NH_3)_2SO_4$ systems are often used. Those metals, which can form amines of copper, cobalt, and nickel, can be dissolved in ammoniacal ammonium carbonate or ammoniacal ammonium sulfate solutions at atmospheric pressure. Native copper can be leached in hydrochloric acid or by ammonia/ammoniacal ammonium sulfate agents (Weiss 1985).

Microbial (or bacterial) leaching is appropriate for low-grade sulfide ores at dump, and heap leach, underground or possibly *in situ* leaching operations. This type of leaching is much slower than typical acid or basic leaching. The organism involved in bacterial leaching is called Thiobacillus ferrooxidans, a small rod-shaped cell about 1 micron ($_2$) for cell growth and oxidizes ferrous iron and sulfides to obtain energy for growth. Sulfuric acid is a product of the organism's metabolism. Sufficient dissolved oxygen must be available during these oxidation reactions. The other main growth requirements are ammonia, nitrogen, phosphate, and a suitable temperature (approximately 30 and acidity (approximate pH of 2.0). In general, higher or lower temperatures (5 lower acidities (pH of 0.5 or 4.5) will not kill the organism, but will severely curtail its activity. Thiobacillus ferrooxidans is usually present in a natural, acidic, sulfide environment. Some metals, such as mercury, silver, and (possibly) molybdenum, can retard or stop leaching by inhibiting or killing the bacteria. Bright sunlight or shallow ponds containing certain other bacteria can also inhibit their activity (Weiss 1985).

The chemical and biochemical reactions involved in microbial leaching of copper ore/minerals are complex. For chalcopyrite, a copper sulfide, it appears that bacteria must come into contact with the mineral to "catalyze" the oxidation reaction. The bacteria first oxidizes the ferrous ore to ferric iron. Ferric iron then chemically oxidizes the sulfide. This bacteria can also assist in the oxidation of sulfur to sulfuric acid. The same reaction also may proceed in the absence of the bacteria, but at a much slower rate. A similar type of reaction occurs for the oxidation of pyrite. These reactions dissolve the sulfide minerals and produce an acidic copper sulfate solution containing ferrous and ferric iron. Other copper sulfide minerals, such as chalcocite, digenite, bornite, and covellite, are more easily leached than chalcopyrite (Weiss 1985).

In some cases, the amount of copper released during the leaching of low-grade sulfide ores has been found to be directly proportional to the quantity of oxygen reacting with the ore. The rate of oxidation depends on a variety of factors; however, the rate can be maximized by maintaining a relatively low pH; the lower the pH, the faster the rate of oxidation. At pH levels above 2.5 or 2.6, the leaching of copper appears to slow considerably (U.S. EPA 1989e). For an in-depth discussion of the chemical reactions involved in the leaching of the principal copper minerals, refer to EPA 1989e.

Leaching with cyanide has been applied almost exclusively to gold and silver, but cyanide has been applied also to copper for both oxidized and low-grade sulfide ores. The effectiveness of cyanide in leaching depends on the ability of the cyanide ion to form stable complexes with the majority of transition metals. These complexes are strong enough to overcome the relative inertness of gold and silver and the insolubility of copper minerals, such as chalcocite, to form copper-cyanide complexes [CuCN, $Cu(CN)_2^-$, $Cu_2(CN)_3^-$, and $Cu(CM)_3^{2-}$] (Weiss 1985).

Figure 1-10

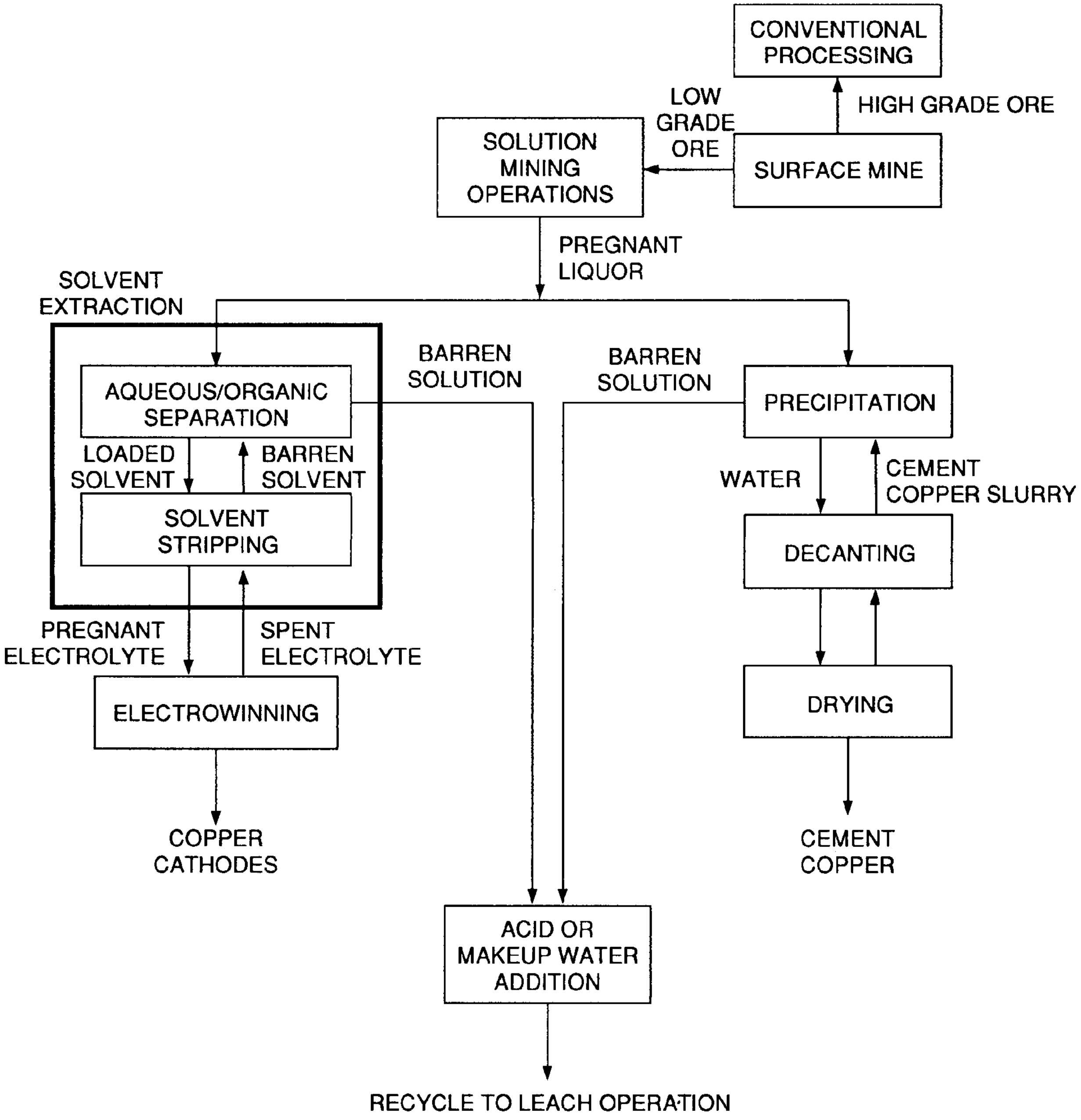


Figure 1-10. Hydrometallurgical Recovery of Copper

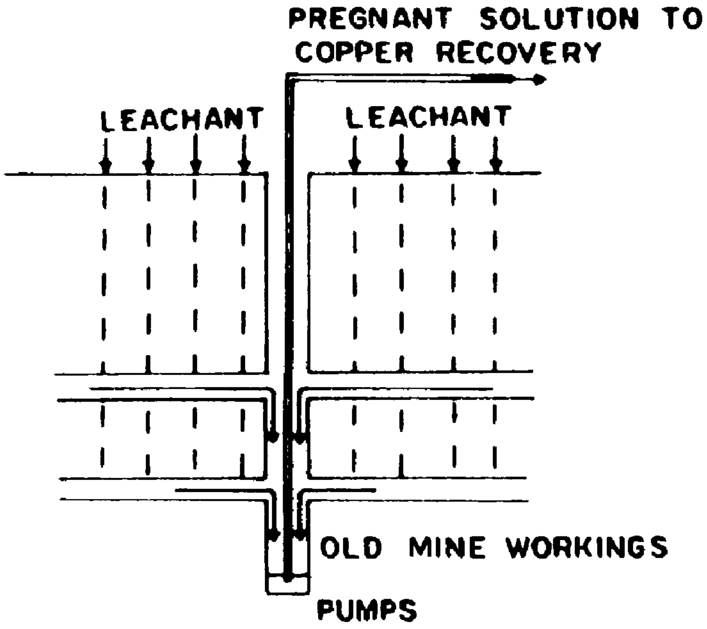
(Source: modified from U.S. EPA 1989e)

summarizes the hydrometallurgical techniques used for copper recovery. In these techniques, copper values are extracted in aqueous solutions from ore or concentrates. The metal and byproducts are then recovered from the PLS by chemical and electrolytic methods (U.S. Congress, Office of Technology Assessment 1988).

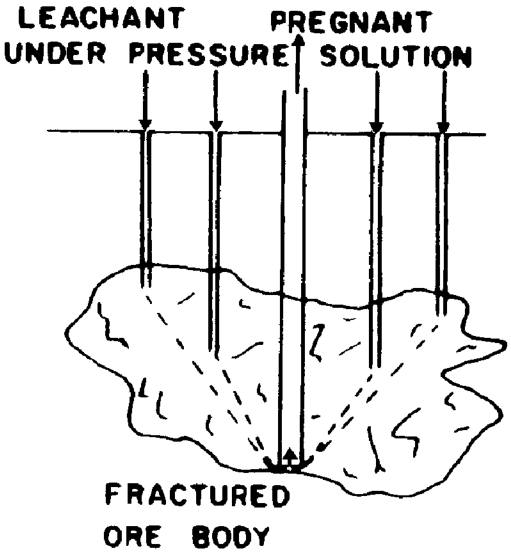
In Situ Leaching

Another leaching method, involving the leaching of low-grade copper ore without its removal from the ground, is known as *in situ* leaching. *In situ* leaching generally refers to the leaching of either disturbed or undisturbed ore. In either case, *in situ* leaching allows only limited control of the solution compared to a lined heap leach type operation. There are 18 *in situ* copper operations in the United States that leach disturbed ore in existing underground mines. *In situ* leaching has certain advantages over conventional mining and milling, including lower capital investment, lower operating costs, and faster startup times. *In situ* leaching of undisturbed ores is best suited for mining relatively deep-lying oxidized copper deposits. *In situ* leaching of disturbed (rubblized) ore is used for extracting copper from any porous or permeable deposits. *In situ* leaching of undisturbed ore, where the rock has not been moved from its pre-mining position, involves very different mining technologies from deposits that have been fragmented by mining operations (such as backfilled stope, and previous block-caving mining operations) or hydrofacted areas (U.S. EPA 1989e; Biswas and Davenport 1976, Graybeal and Larson, 1989).

In situ leaching, as shown in Figures 1-11a and 1-11b



a. *In Situ* Leaching of Old Mine Workings



b. *In Situ* Leaching of Fractured Underground Deposit

Figure 1-11. *In Situ* Leaching Operations

(Source: Biswas and Davenport 1976)

, extracts copper from subsurface ore deposits without excavation. Typically, the interstitial porosity and permeability of the rock are important factors in the circulation system. The solutions are injected in wells and recovered by a nearby pump/production-well system. In some cases (where the ore body's interstitial porosity is low), the ore may be prepared for leaching (i.e., broken up) by blasting or hydraulic fracturing. The chemistry of *in situ* leaching is similar to that of heap and dump leaching operations. The ore is oxidized by lixiviant solutions such as mine water, sulfuric acids, or alkalines that are injected from wells into an ore body to leach and remove the valuable minerals. Production wells capture and pump pregnant

lixiviant solution from the formation to the leach plant where copper metal is recovered by an SX/EW operation (Biswas and Davenport 1976; U.S. EPA 1984a; U.S. EPA 1989e).

Monitoring wells are used to monitor the ground-water system and detect any lixiviant migration beyond the leaching area. After the ore body is depleted, *in situ* leach operators may be required to restore the aquifer. *In situ* mining of undisturbed ore is being conducted on an experimental basis in the copper mining industry (Biswas and Davenport 1976; U.S. EPA 1984a; U.S. EPA 1989e; Graybeal and Larson 1989).

The economics of current mining and recovery methods often prevent the mining of ore that either contains insufficient metal values or requires extensive site preparation or operating expense. For this reason, the *in situ* leach method is gaining favor as a means of recovering additional copper from old mine workings (i.e., block-caved areas and backfilled stopes) from which the primary sulfide deposit has been mined. These types of operations tend to leave behind considerable fractured, copper-bearing rock that is expensive to mine and recover by conventional means (U.S. EPA 1989e).

Stope leaching is a specialized type of *in situ* solution mining that involves leaching of underground, low-grade ore deposits at active and inactive mines. Lixiviant solution is introduced into a worked-out underground mine; backfilled underground stopes; or collapsed block-caved areas (where the stopes were backfilled with low-grade waste rock). As the fluid flows through the stopes or caved areas, it dissolves the minerals and collects in lower levels of the mine (i.e., the sumps); from there it is pumped to the surface. There, the copper is recovered by the SX/EW method (U.S. EPA 1989e). One example of stope leaching occurs at Magma Copper Company's San Manuel facility, where *in situ* leaching, open pit mining, and underground mining are all conducted simultaneously in different parts of the same ore body. During the last few years, all of the production from Cyprus' Casa Grande property has been from *in situ* leaching, including a stope leaching project and testing of *in situ* leaching of virgin ground (U.S. EPA 1989e; Beard 1990).

Most abandoned underground mining operations leave halos or zones of low-grade ore surrounding tunnels, stopes, rises, and pillars. The underground mine development (i.e., the shafts and drifts) required in such mines normally provides the basic circulation needed for a leaching operation.

Usually, lixiviant solutions are introduced into the surrounding low-grade ore zones from above by injection through a series of drillholes. The main shaft is almost always used as a main drainage reservoir. Because drifts are designed to run upgrade, water or leach solutions flow naturally by gravity to the main shaft for recovery. Fluids flowing from the extraction drifts and haulage drifts are usually collected behind a dam placed across the main shaft and pumped to the surface. At block-caved operations, the caving method causes the area above the stope mine to be highly fractured and broken. This expands its volume, which increases the porosity of the low-grade ore. Thus, an ideal circulation system for stope leaching operations is created (U.S. EPA 1989e).

Dump Leach Operations (At Open-Pit Mines)

Dump leaching refers to leaching that takes place on an unlined surface. The term "dump leaching" derives from the practice of leaching materials that were initially deposited as waste rock; however, now it also is applied to of run-of-mine, low-grade sulfide or mixed grade sulfide and oxide rock placed on unprepared ground specifically for leaching. Copper dump leaches are typically massive, with waste rock piled into large piles ranging in size from 20 feet to over 100 feet in height. These may cover hundreds of acres and contain millions of tons of waste rock and low-grade ore (Biswas and Davenport 1976). Dump leaching techniques are illustrated in Figure 1-12

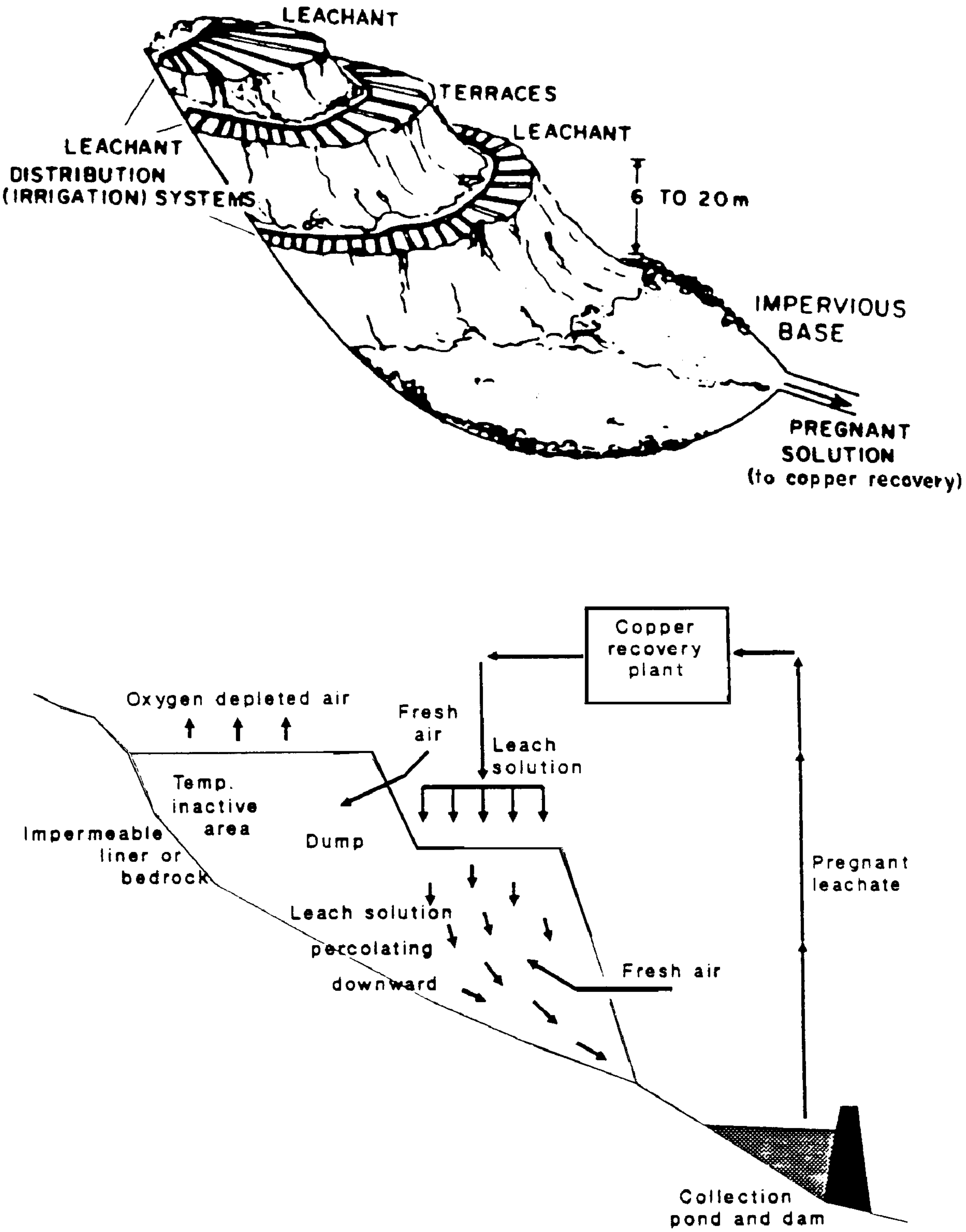


Figure 1-12. Leach Dump Operations

(Source: Biswas and Davenport 1976; U.S. Congress, Office of Technology Assessment 1988)

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In the 1920s, large-scale commercial leaching of waste piles was initiated to recover copper. These operations entailed the addition of low pH solution to the piles to accelerate leaching, the collection of PLS, and the extraction of copper by iron precipitation to generate "cement copper." The sites for these historic dump leaches were selected primarily to minimize haulage distances, thereby reducing costs (the extent of cost savings would have been based on site specific factors). Dump leaches were located and designed to prevent the loss of leach solution (U.S. EPA 1989e).

Waste rock, removed to expose the ore body, was placed in piles close to the pit site. Water seepage from the piles was found to contain high concentrations of copper. Miners realized the opportunity to recover copper at virtually no cost. The percentage of copper produced from leaching operations has increased in recent years due to the low operating costs of dump leaching relative to conventional milling operations (the extent of cost savings is based on site-specific factors). Dump leaching is usually associated with copper recovery, although uranium and gold may be leached through a similar procedure. Dumps are usually sited in an area where the slope of the native terrain provides the means for collection of pregnant liquor. The leach solution flows by gravity through the dump and then over the slope of the native ground beneath the dump to a collection point, usually a pond, at the downgrade toe of the dump (U.S. EPA 1989e).

The materials employed generally vary considerably in particle size, from large angular blocks of hard rock to highly weathered fine-grained soils. Most of the material is less than 0.6 meter in diameter. In most dump leach operations, the material is hauled to the top of the dump by trucks. Bulldozers are used to level the surfaces and edges of the dump. The material is typically deposited by end-dumping in lifts on top of an existing dump that has already been leached. Large dumps are usually raised in lifts of 15 to 30 meters. Some sorting of materials occurs when this method of deposition is used. Coarser fragments tend to roll down to the bottom of the slope, whereas finer materials accumulate near the surface of the dump. A degree of compacting in the top meter of each lift results from the heavy equipment and truck use. After the lift is completed, the top layer is scarified (by a bulldozer and a ripper) to facilitate infiltration of the leach solution (U.S. EPA 1989e).

Most dump leaches begin to settle as they are built and continue to settle after the leach solutions have been applied. This continued settling results, in part, from the percolating liquid moving the finer particles into the spaces between larger particles. The dump is compressed also by the added weight of the solutions and the destruction of the bridging rocks' competency by chemical reactions that depreciate the rock (U.S. EPA 1989e).

Natural precipitation, mine water, raffinate, makeup water, or dilute sulfuric acid may be used as leach solution (i.e., lixiviant). As the lixiviant infiltrates the pile and leaches out copper minerals by oxidizing the pyrite to form sulfuric acid and ferrous iron solution, the sulfuric acid solution reacts with the ore minerals to ionize the copper into solution. Once dissolved, the metals remain in solution. This leaching method is best suited to nonsulfide oxide ores rich in azurite, malachite, and other oxide minerals. Sulfide ores rich in chalcocite may also be leached using a similar method. In this method, the ore is leached by an active bacterial population that uses oxygen to convert ferrous iron to ferric iron, which reacts with chalcocite-liberating copper and generates ferric sulfide (U.S. EPA 1984a).

Several methods may be used to distribute leach solutions over the dumps, including natural precipitation, sprinkler systems that spray the leach solution over the piles, flooding of infiltration ditches or construction of leach solution ponds on top of the dumps, distribution of leach solution through perforated pipe on top of the dump (known as trickle systems), and the injection of leach solutions through drill holes into the dump. The leach solution percolates through the dump and PLS is collected in ditches or sumps at the toe of the dump. These ditches and sumps are lined at some sites, and are unlined at others. PLS is then treated by solvent extraction or cementation. Metals associated with the copper ores that dissolve (and are potential contaminants) include arsenic, cadmium, chromium, and selenium (U.S. EPA 1985a; U.S. EPA 1984a).

Heap Leaching

In contrast to dump leaching (described in the previous section), heap leaching refers to the leaching of low-grade ore that has been deposited on a specially prepared, lined pad constructed using synthetic material, asphalt, or compacted clay. In heap leaching, the ore is frequently beneficiated by some type of size reduction (usually crushing) prior to placement on the pad. Site-specific characteristics determine the nature and extent of the crushing and the leaching operations used (U.S. EPA 1989e).

Heap leach pads are constructed above one or more layers of impermeable liner material. Liners can be constructed using synthetic membrane [such as High-Density Polyethylene (HDPE)] and/or natural material (such as compacted native soils or clays or unfractured/unfaulted bedrock). Most leach sites are selected to take advantage of existing, less permeable surfaces and to utilize the natural slope of ridges and valleys for the collection of PLS. Land with this type of geology and terrain, however, is not always within a reasonable hauling distance of the mining operation.

A typical heap leach operation is depicted in Figure 1-13. The same basic principles and procedures discussed earlier with regard to dump leaching operations apply to heap leach operations. Heap leach

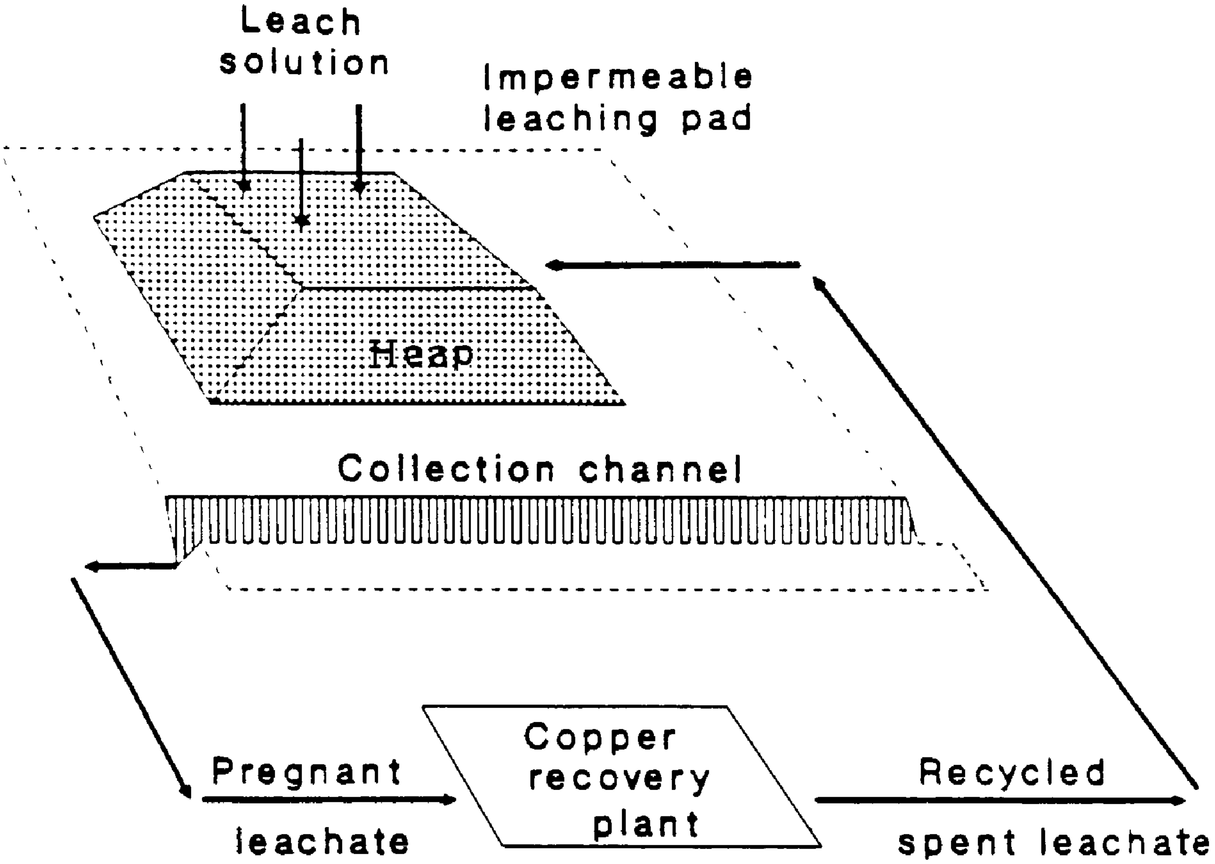

Figure 1-13. Heap Leach Unit Design

(Source: U.S. Congress, Office of Technology Assessment 1988) operations, as opposed to dump leach operations, have the following characteristics: (1) higher lixiviant concentrations generally are used; (2) leach piles may be neutralized after leaching operations are completed; (3) the leach pad design is substantially different (i.e., the size is smaller); (4) the ore is finer grained (i.e., usually less than 10 cm); (5) the leaching is considerably faster; and (6) the extraction of oxide copper is greater. The copper recovery of sulfide minerals in the heaps, as with dumps, is usually low due to shorter leaching times (100 to 180 days) and relatively poor lixiviant-sulfide dissolution kinetics (Biswas and Davenport 1976; U.S. EPA 1989e).

Heap leaching is generally suited to oxide ores for several reasons: usually oxide deposits are smaller than sulfide deposits; oxides leach more rapidly than sulfides; the oxide leachate has a higher copper content than the sulfide leachate; and high-grade refractory oxide ores are not recoverable in the standard sulfide flotation concentrator (U.S. EPA 1989e). Copper heap leach operations are much smaller than copper dump leaches. On the average, heaps contain between 100,000 and 500,000 metric tons of ore. Copper heaps are designed and operated to minimize truck traffic and bull-dozer work on the surface. This serves to reduce the compaction resulting from these activities, thereby improving the permeability of the heap. One method of constructing a new heap involves placement of the leach material in a strip along the center of the new heap. Subsequent loads are then dumped along the outer edge of the strip and pushed over the side with a bulldozer to build the heap to its full width. With this method of material emplacement, only the top meter of the heap becomes compacted. This layer is subsequently scarified to promote infiltration of the leach solution. The heap leaching cycle typically lasts between 60 and 180 days. Application of leaching solution is generally stopped after a specified period, which is dictated by the leaching cycle or when the copper content of the pregnant liquor falls below a predetermined concentration. Subsequently, the surface of the nearly barren area is scarified by ripping and another lift is begun on the surface (U.S. EPA 1989e).

Because most distribution methods do not provide completely uniform coverage, the rate at which the solution is applied to the heap will vary. The application rate is generally defined as the volumetric flow rate of the leach solution divided by the surface area to which the solution is actually being applied. The average application rate varies between 20 liters per square meter (l/m^2) per hour for sprinklers, to as much as 200 l/m^2 per hour for pond leaching (U.S. EPA 1989e).

In practice, most heaps are leached in sections. Near the end of the leach cycle, heap permeability diminishes because of the accumulation of decomposed clay materials and iron salt precipitates. This requires that the surface be scarified by ripping, after which leaching is resumed or another lift is begun on the surface. The alternate wetting and resting during the leach cycle promotes efficient leaching of sulfide minerals within the heap (U.S. EPA 1989e).

Under the influence of gravity, the leaching solution percolates down through the ore and carries the dissolved copper along with it. Uniform distribution of the leach solution throughout the heap is difficult to achieve. In sloped areas, channeling the solution down the slope accelerates runoff. Within the heap, alternate layers of coarse and fine materials (as a result of poor heap construction) promote horizontal solution flow, which may result in the discharge of the copper-bearing liquor from the sides of the heap rather than from the base. The total volume of leach solutions added to the heap must be controlled to prevent sloped areas from becoming saturated. Excess moisture in the pile can lead to slumping of large amounts of material (U.S. EPA 1989e).

When PLS reaches the bottom of the heap, it flows to a collection channel and/or holding pond at the toe of the dump. Holding ponds generally are located in natural drainage basins enclosed by a dam or excavated and bermed on level surfaces. The pregnant solution is pumped from the dam to the precipitation or solvent extraction plant, where the copper is recovered from solution (U.S. EPA 1989e).

Vat Leaching

The vat leaching process works on the same principles as the dump and heap leaching operations described above, except that it is a high-production-rate method conducted in a system of vats or tanks using concentrated lixiviant solutions. Vat leaching typically is used to extract copper from oxide ores by exposing the crushed ore to concentrated sulfuric acid (lixiviant) in a series of large tanks or vats. The vats are usually designed in a series configuration, which acts to concentrate the copper content of the solutions as a function of ore-lixiviant contact time (U.S. EPA 1989e).

Vat leach units may be large drums, barrels, tanks, or vats. The design capacity of the leaching units is dependent on the amount of ore to be leached. For example, a 25-meter-long, 15-meter-wide, and 6-meter-deep vat unit is capable of leaching between 3,000 and 5,000 tons of ore per cycle. Leaching operations may be conducted under a number of environments, including slightly subatmospheric, atmospheric, or superatmospheric pressures, and under ambient or elevated temperatures (Weiss 1985).

Vat leaching units can be constructed of a number of materials, depending on the composition of the slurried ore feed, the lixiviant used, and the conditions under which leaching will take place. Table 1-6 gives examples of some vat construction materials recommended for use with different lixiviants.

Table 1-6. Lixiviants and Recommended Construction Materials

Lixiviant	Recommended Construction Materials
Sulfuric Acid	Wood, lead, lead and acid brick, stainless steel, titanium, and HDPE
Hydrochloric Acid	Rubber-lined mild steel, rubber lining and acid brick (for a temperature of 70
Sodium Hydroxide	Mild steel
Ammonium Hydroxide	Stainless steel

(Source: Weiss 1985)

Vat and agitation (tank) leaching are usually performed on relatively higher oxidized ores. Tank methods tend to recover copper more rapidly using shorter leach cycle times than heap or dump leaching operations. Generally, copper recovery is higher, copper content in the leach solution is higher, and solution losses are lower with tank methods (U.S. EPA 1984a). Vat leaching has been preferred over heap leaching in cases where high-grade ore requires crushing to permit adequate contact between the leach solution and the copper minerals. The advantages of this method are high copper extraction rates and recoveries, short leach cycles, and negligible solution losses (U.S. EPA 1989e). The disadvantages are the low tonnages beneficiated, high suspended solids concentrations in PLS that cause problems in the SX/EW plant, and high operating costs.

In the tank leaching process, the ore is first crushed to approximately less than 1 cm. The ore is screened to separate the fines before it is placed in the vats. Most vat leaching operations use several large, rectangular tanks with floors that act as filters to facilitate the upflow and downflow of solutions. A typical vat measures 25 meters long, 15 meters wide, and 6 meters deep and contains between 3,000 and 5,000 metric tons of material. Vat leaching is a batch operation; its cycle involves vat loading; ore leaching, washing, and draining; and vat excavating. The crushed ore is immersed in 50 to 100 kilograms per cubic meter (kg/m^3) of sulfuric acid solution. The leaching usually takes place in a sequence of four to seven soak-drain cycles. The pregnant solutions from the first two or three soaks are used as electrolyte (after purification), while the remaining solutions (which are more dilute) are reused to leach subsequent fresh batches of ore. The solutions from the remaining soaks are recycled as leachate for subsequent batches of fresh ore (Biswas and Davenport 1976; U.S. EPA 1989e).

Continuous vat leaching, in which leachate flows continuously through ore in a sequence of vats, is now being practiced at several mines. Factors that affect the leach rate (in both batch and continuous leaching) include particle size and porosity, temperature, and acid strength. The overall cycle may take from 10 to 14 days. Vat leaching produces a PLS of sufficient copper concentration for electrowinning (30 to 50 kg/m^3 of copper). If the iron content of the solution is high, the PLS may be sent for solvent extraction prior to electrowinning. This is necessary because iron may reduce the efficiency of the electrowinning (this method is discussed in more detail later in this report) (Biswas and Davenport 1976; U.S. EPA 1989e).

Agitated vat leaching refers to the relatively rapid leaching of fine particles of copper oxide ore or roaster calcines with a strong sulfuric acid solution in agitated tanks. The tanks are stirred or agitated by mechanical devices or piped steam discharge. Compressed air is used in a similar method of agitation in a pressurized tank operation. A pressurized operation is used in several different types of autoclaves. Figure 1-14 illustrates two examples of pressurized vat leach systems (U.S. EPA 1989e).

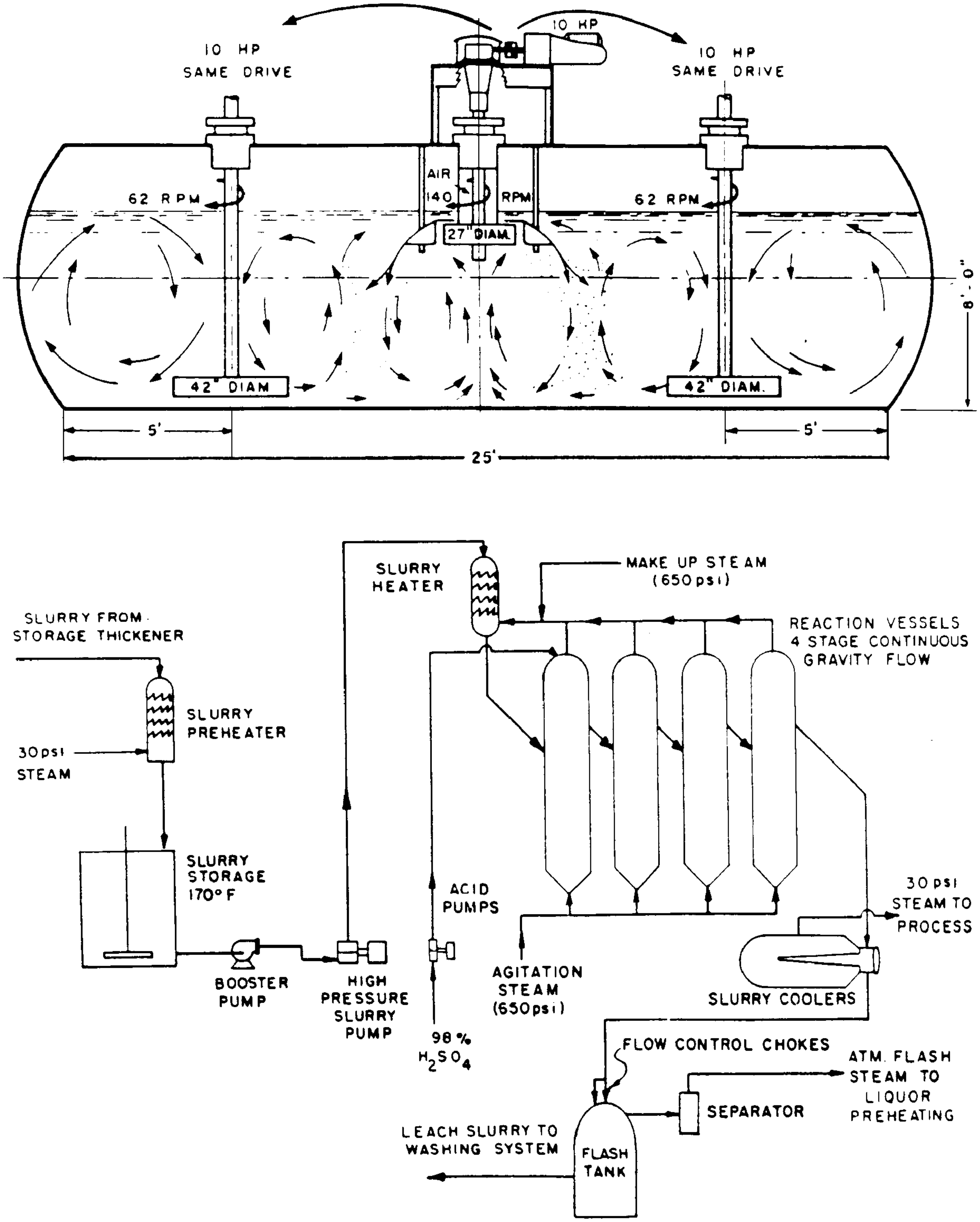

Figure 1-14. Typical Autoclave and Vat Leaching Circuit

(Source: Weiss 1985)

This leaching method has been used primarily in conjunction with vat leaching operations to recover copper from the fines filtered out of the vat material. Additional lean material is crushed and ground to a fine-sized particle [90 percent are less than 75 micrometers (µm)] and combined with the fines from the vat operation. This material is then mixed with the leach solution to form a slurry with a solids content of between 30 and 40 percent. The mixture is agitated by air or mechanical means in a series of three or six tanks [with a volume of 50 to 200 cubic meters (m^3)] for a period of two to five hours. On completion of the leach cycle, the pregnant liquor is separated from the acid-insoluble residue by concurrent or countercurrent washing. Because of the fine particle size of the solids, the strength of the acid solution, and the agitation of the leach slurry (which promotes better liquid-solid contact), agitation leaching demonstrates the highest recovery of copper. In some instances, recovery is greater than 95 percent. Vat and agitation leaching are generally more rapid, more efficient, and much more costly than dump or heap leaching (U.S. EPA 1989e).

Cementation

In the past, copper was recovered from leach solutions through a cementation technique (precipitation from solution by the replacement of copper in solution by metallic iron). This has been a source of relatively inexpensive copper; however, the cement copper produced is relatively impure compared to electrowon copper and must be smelted and refined along with flotation concentrates (Beard 1990).

In the cementation technique, PLS flows to a precipitator pond filled with scrap iron or steel. The copper chemically reacts with, and precipitates onto the steel surfaces. The iron is dissolved into solution, and the copper precipitates out (i.e., replaces) the iron. The cemented copper later detaches from the steel surfaces as flakes or powder when it is washed with high-pressure streams of water. Although subsequent treatment by a normal smelting/refining method is required, copper recovery from the pregnant solution is very high. Typically, cemented copper contains between 65 and 85 percent pure copper, with oxides of iron and other traces of silica and aluminum oxides (Beard 1990).

Swapping ions occurs whenever a metal ion in solution is reduced to an elemental state by a more reactive metal. Iron is more reactive than gold, mercury, silver, or copper; hence these metals easily precipitate. Iron is only slightly more reactive than lead, tin, nickel, or cadmium; and these metals do not easily precipitate since kinetics control the reaction. Chromium, zinc, aluminum, magnesium, calcium, and sodium are more reactive than iron and also do not precipitate. As a result, barren leach solutions remain very acidic and contain elevated levels of metals and salts that are more reactive than iron or are similarly reactive (U.S. EPA 1987).

There are numerous cementation precipitator designs and configurations. Typically, precipitators are shallow-round or stair-stepped wooden or concrete basins (U.S. Congress, Office of Technology Assessment 1988). The simplest and most common precipitation system used in the copper mining industry is an open-launder-type cementation system. PLS flows down a wooden or concrete trough or series of troughs filled with scrap iron. Launders vary in size and dimension depending on the amounts of leach liquor being treated; the launder may be straight or zigzagged. More modern units employ a series of wooden grids, positioned above the bottom of the launders. These permit the cemented copper to fall to the bottom, where it easily can be recovered (Biswas and Davenport 1976).

Several compact and dynamic cementation systems have been developed and are used industrially. The most successful is the Kennecott Cone System Precipitator, in which the PLS is forced upwards in a swirling motion through shredded steel scrap. The discussion of Bingham Canyon mine in Appendix 1-B presents a complete description of the Kennecott Cone System Precipitator. The Kennecott Cone System Precipitator has proven to be highly effective and has been used by many large-scale leaching operations. In this system, fine, undissolved solid particles (called pulp) are concentrated with the copper cemented particles. Consequently, the cement concentrates containing the pulp must be further beneficiated by flotation. The cemented copper is easily floated with xanthate or dixanthogen collectors (Biswas and Davenport 1976).

Ion Exchange

The use of ion-exchange recovery in the copper industry is not widespread. Generally, three circuits are used in an ion-recovery operation: the extraction circuit, the elution circuit, and the precipitation circuit. The system is designed to recycle lixiviant back to the leach operation. (U.S. Congress, Office of Technology Assessment 1988; Arizona BADCT Draft; Utah Department of Health, undated).

The extraction circuit extracts metals from the pregnant lixiviant. Copper complexes with the resin as the pregnant lixiviant flows through the resin in the ion exchange unit. Barren lixiviant leaving the ion exchange unit is refortified with chemicals and recycled. The copper metal is released from the loaded resin in the elution circuit. This is accomplished using a high-ionic-strength solution. The effluent is known as the pregnant eluate. The pregnant eluate proceeds to the precipitation circuit, where acid is added to destroy the copper complexes in solution and precipitate copper oxide. The barren electrolyte is reprocessed to the elution circuit, although electrolyte is constantly bled from the system to control the level of impurities (U.S. Congress, Office of Technology Assessment 1988; Arizona BADCT Draft; State of Utah, undated).

Solvent Extraction

The first SX/EW plant was developed during the 1960s at the Bluebird property near Miami, Arizona. Solvent extraction largely had been confined to copper oxides until recent developments in leaching methods. Figure 1-15 provides a flow diagram for a typical SX/EW plant. In the traditional solvent extraction circuit, copper is dissolved from the ore into an aqueous solution by weak sulfuric acid. The pregnant solution is then pumped to a solvent extraction plant, where it is mixed with an organic solvent (U.S. Congress, Office of Technology Assessment 1988; Arizona BADCT Draft; State of Utah, undated).

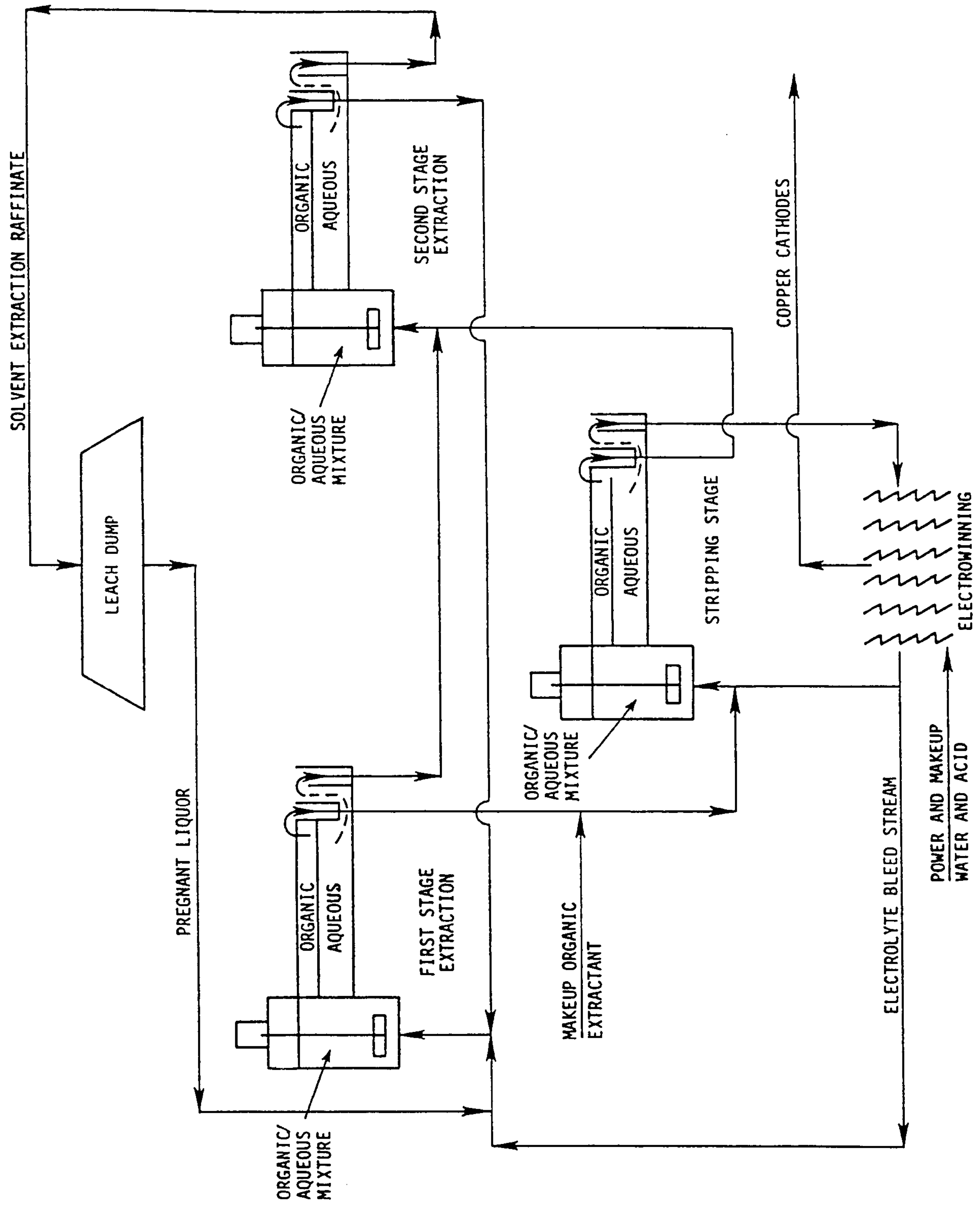

Figure 1-15. Typical Solvent Extraction/Electrowinning (SX/EW) Plant

(Source: U.S. EPA 1989e)

The solvent extraction operation is a two-stage method. In the first stage, low-grade, impure leach solutions containing copper, iron, and other base-metal ions are fed to the extraction stage mixer-settler. In the mixer, the aqueous solution is contacted with an active organic extractant (chelating agent) in an organic diluent (usually kerosene), forming a copper-organic complex. The organic phase extractant is designed to extract only the desired metal ion (i.e., copper), while impurities such as iron or molybdenum are left behind in the aqueous phase. The aqueous-organic dispersion is physically separated in a settler stage (U.S. Congress, Office of Technology Assessment 1988; Arizona BADCT Draft; U.S. EPA 1984a; Engineering and Mining Journal 1990).

Because of the development of the faster, more selective salicylaldoxime reagents, most of the new copper solvent extraction plants can use two-stage extraction with a single stripping stage in each circuit. Two stages of extraction have proven sufficient to remove 90 percent or more of the copper from leach solutions, whereas early plant designs required three or four stages of extraction (Engineering and Mining Journal 1990).

The barren aqueous solution, called raffinate, is recirculated back to the leaching units. The loaded organic solution is transferred from the extraction section to the stripping section. The major advantage of solvent extraction is that the electrolyte solution it produces is almost free of impurities (U.S. Congress, Office of Technology Assessment 1988; Arizona BADCT Draft; Engineering and Mining Journal 1990).

In the second stage, the loaded organic solution is stripped with concentrated sulfuric acid solution (spent tankhouse electrolyte) to produce a clean, high-grade solution of copper for electrowinning. The stripping section can have one or more mixer-settler stages. In particular, the loaded-organic phase is mixed with a highly acidic electrolyte (returned from electrowinning), which strips the copper ions from the organic phase. Then the mixture is allowed to separate in settling tanks, where the barren organic solution can be recycled to the extraction stage. The copper-enriched, strong electrolyte flows from the stripping stages to the strong-electrolyte tanks, where it is pumped to the electrolyte filters for removal of the entrained organics or solids. The clarified, strong electrolyte (which is the concentrated sulfuric acid from the solvent extraction operation) flows to electrolyte circulation tanks, where it becomes electrolyte for the electrowinning tankhouse. At the tankhouse, copper is plated out of solution onto cathodes (U.S. Congress, Office of Technology Assessment 1988; Arizona BADCT Draft; U.S. EPA 1984a; Engineering and Mining Journal 1990).

The solvent extraction method is dependent on the solubility of the reagents and the equilibrium constants that control the reactions. The higher the equilibrium constant, the more effective the reagent is in stripping copper from the pregnant solution. Specifically, two factors that are controlled by the equilibrium constant and that significantly impact the effectiveness of a reagent are the rate of the reaction and the range between loading and stripping for a particular reagent (U.S. Congress, Office of Technology Assessment 1988; Arizona BADCT Draft; U.S. EPA 1984a; Engineering and Mining Journal 1990).

The most prominent copper extractants used in the copper plant solvent extraction system are LIX and Acorga. These reagents are referred to by their individual trade names because their exact chemical compositions are listed as confidential business information (although some general information is available).

The most widely used LIX reagents are LIX 63, 64N, 622, 84, 860, and 984. The most widely used Acorga reagents are PT-5050, P-5100, M-5640, and M-5615. They are usually composed of either aldoxime or ketoxime compounds in an organic solvent of kerosene.

Modifiers are usually added to improve the reaction rates and/or phase separation. Data are very limited on the various types of modifiers used at copper extraction plants, although several were identified: LIX 6022 is a tridecanol-modified dodecyl-salicylaldoxime; PT-5050 is a tridecanol-modified nonyl-salicyladoxime; LIX 860 is an unmodified dececylsalicycylaldoxime; LIX 84 is an unmodified nonylacetophenone; and LIX 984 is 50-50 mixture of LIX 860 and LIX84.

The obvious advantage of solvent extraction is that cathode copper of salable quality can be produced directly from leach solutions. Therefore, smelting is not required. Interest in the SX/EW method has grown gradually. Twelve plants were in operation during 1989. Several expansions and new plants are being planned, while the cementation process is being phased out (except as a subsidiary method of copper production) (Beard 1990).

Electrowinning

Electrowinning is the method used to recover copper from the electrolyte solution produced by solvent extraction. Electrowinning uses inert (nondissolving) anodes made of lead (alloyed with calcium and tin) or stainless steel, referred to as sheets. (U.S. Congress, Office of Technology Assessment 1988).

To stabilize the tankhouse operating temperature and preheat the incoming electrolyte solution, strong electrolyte (after filtration) is passed through heat exchangers where heat is extracted from outgoing, warmer, spent electrolyte. After passing through starting-sheet cells, the strong electrolyte is received in a circulation tank. In the circulation tank, the strong electrolyte is mixed with spent electrolyte returning from the electrowinning cells. Water and any deposit-modifying reagents are added in this tank. The feed electrolyte is then pumped to the electrolytic cells continuously. The electrochemical reaction at the lead-based anodes produces oxygen gas and sulfuric acid by electrolysis. Copper is plated on cathodes of stainless steel or on thin-copper starting sheets. The cathode copper is then shipped to a rod mill for fabrication. The spent acid is recycled and pumped back to the leaching operation, while some of the electrolyte is pumped to the solvent extraction strip-mixer-settlers via the electrolyte heat exchangers (U.S. Congress, Office of Technology Assessment 1988; Engineering and Mining Journal 1990).

If the cathode copper is plated onto a stainless steel "blank," the copper plate is peeled off the blank prior to shipment and the blank is reused. This blank and the techniques developed to optimize its use are known as the "ISA" method. Magma Copper Co. has extended the use of this technology, which was first used in electroplating, to electrowinning operations (Engineering and Mining Journal 1990).

The ISA method uses conventional, insoluble anodes, but rather than using starting sheets to receive the cathode deposit, the ISA method employs 316L stainless steel blanks. About 7 days are required to complete a cathode side, and automated equipment is used to strip them (Engineering and Mining Journal 1990). This total production stripping system has numerous benefits, including the following:

- There is no starting-sheet deposition.
- There is no stripping labor, stripping, or sheet-fabrication equipment.
- The better-defined cathode (in the form of a rigid blank) is less prone to warping, and therefore, requires less rigid inspection.
- Shorter cathode cycles reduce the metal inventory.
- There are no suspension loops to corrode. Therefore, the incidence of cell-liner cutting is lower and crane handling is easier.

The elimination of sheet production and reduced inspection means the work force is up to 60 percent smaller than it is at a conventional plant (Engineering and Mining Journal 1990).

1.5 WASTES AND OTHER MATERIALS ASSOCIATED WITH COPPER EXTRACTION AND BENEFICIATION

This section describes several of the wastes and materials that are generated and/or managed at copper extraction and beneficiation operations and the means by which they are managed. As is noted in the previous section, a variety of wastes and other materials are generated and managed by copper mining operations.

Some, such as waste rock and tailings, are generally considered to be wastes and are managed as such, typically in on-site management units. Even these materials, however, may be used for various purposes (either on- or off-site) in lieu of disposal. Some quantities of waste rock and tailings, for example, may be used as construction or foundation materials at times during a mine's life. Many other materials that are generated and/or used at mine sites may only occasionally or periodically be managed as wastes. These include mine water removed from underground workings or open pits, which usually is recirculated for on-site use (e.g., as mill/leaching makeup water) but at times can be discharged to surface waters. As another example, leaching solutions are typically regenerated and reused continuously for extended periods. On occasion, however, such during temporary or permanent closure, the solutions are disposed as wastes via land application or other means. Finally, some materials are not considered wastes at all until a particular time in their life cycles. These include spent ore at dump leaching operations: here, only when active leaching for copper recovery ends is the spent ore that comprises the dump considered a waste.

The issue of whether a particular material is a waste clearly depends on the specific circumstances surrounding its generation and management at the time. In addition, some materials that are wastes within the plain meaning of the word are not "solid wastes" as defined under RCRA and thus are not subject to regulation under RCRA. These include, for example, mine water or process wastewater that is discharged pursuant to an NPDES permit. It is emphasized that any questions as to whether a particular material is a waste at a given time should be directed to the appropriate EPA Regional office.

The first subsection below describes several of the more important wastes (as defined under RCRA or otherwise) and nonwastes alike, since either can have important implications for environmental performance of a facility. The next subsection describes the major types of waste units and mine structures that are of most environmental concern during and after the active life of an operation. Figure 1-16

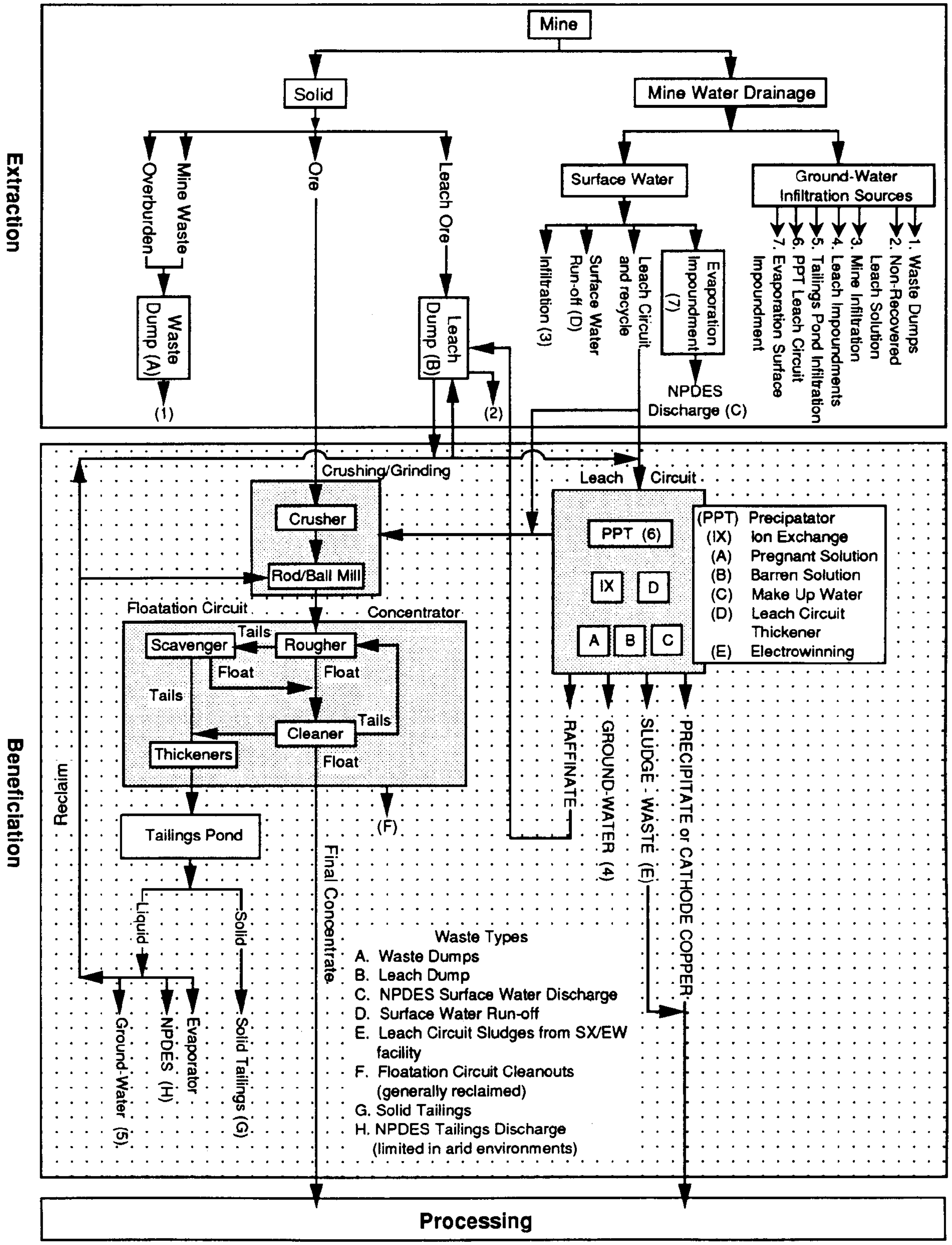

Figure 1-16. Schematic of Typical Copper Mining Extraction and Beneficiation Wastestreams

(Source: Modified from U.S. EPA 1985a)

identifies many of the typical wastes and materials and management practices employed by the copper industry.

1.5.1 Extraction and Beneficiation Wastes and Materials

The subsections below describe many of the wastes and materials generated and managed at copper sites. Notwithstanding the status of a particular waste or material, it should be noted that a number of factors determine whether that waste or material poses any risk to human health or the environment. Perhaps the most important are the inherent nature of the material (which is generally determined by its origin and the processes by which it is generated), the manner in which the material is managed, and the environment in which it is managed and to which it could be released. As noted above, questions concerning the actual status of any particular material or waste should be directed to the appropriate EPA Region.

1.5.1.1 RCRA Wastes

Waste Rock

For this discussion, waste rock is defined as all overburden and mine development rock moved during mining. These materials contain little or no recoverable mineral values. Industry uses the term "overburden" to refer to nonmineralized soils and rock that are above (over) an ore body. Similarly, mine development rock refers to material removed from underground mines to access the ore body. Waste rock is used by industry to refer to poor or nonmineralized rock that is within or surrounding the ore body at surface mines.

Waste rock and ore are relative terms in the context of copper porphyry ore bodies since few distinct boundaries exist. Usually, contacts between mineralization zones are gradational; there is a gradual increase in mineralization from nonmineralized areas to quality ore areas. Therefore, waste rock may contain some values. (U.S. DOI, Bureau of Mines 1965a).

Waste rock is typically hauled from the mine site to waste dumps for disposal. Waste rock piles may have high permeability to both air and water. Oxygen and sulfide minerals may be contained in the dump. The quantity and composition of waste rock generated at mines vary greatly by site. This material can be classified as either oxide or sulfide, with varying solubilities, depending on the composition of the ore body. Sulfur-bearing minerals, such as pyrite and pyrrhotite, can oxidize to form sulfuric acid. Factors that influence acid generation by sulfide wastes include: (1) the amount and frequency of precipitation, (2) the design of the disposal unit, and (3) the neutralization potential of the rock. Constituents of concern for waste rock include sulfur-bearing minerals that may generate acid and leach metals contained in the ore body and surrounding rock.

Tailings

Tailings are generated during flotation. Tailings are made up of very fine host rock (i.e., gangue) and nonmetallic minerals separated from the values during beneficiation. The physical and chemical nature of tailings varies according to the ore characteristics and the beneficiation techniques used. Tailings are a slurry of fine-grained rock material and process water. Liquid is removed from the tailings slurry in thickeners and the thickened tailings are discharged to the tailings impoundment. Water is usually reclaimed from the thickeners and recirculated to the mill to be used in beneficiation and dust control (U.S. DOI, Bureau of Mines 1965a).

In the arid southwest, where evaporation rates exceed precipitation, the mine-mill water balance usually requires that water recovered in the tailings pond be recycled to the mill as process water. At copper mines in the central United States (such as White Pine in Michigan) the reverse situation exists; precipitation exceeds evaporation rates and excess mine-related water must be discharged to the environment (U.S. DOI, Bureau of Mines 1965a).

In 1985, 195 million tons of copper and copper-molybdenum ores were treated by flotation concentration, resulting in the production of 5.8 million tons of concentrate using 97 million gallons of water and 0.32 million tons of reagents. More than 97 percent (189 million tons) of ore tonnage processed in 1985 was disposed of as tailings (U.S. DOI, Bureau of Mines 1987a).

Spent Ore from Heap, Dump, and Vat Leaching

Spent ore consists of the material remaining in either dump or heap leach piles when leaching ceases. Spent ore from heap, dump, and vat leaching may contain residual lixiviant and other constituents of the ore. Some operations may refer to wastes from vat leaching operations as tailings.

1.5.1.2 Materials

Mine Water

Mine water is generated when water collects in mine workings, both surface and underground, as a result of inflow from rain or surface water and from ground water seepage. During the active life of the mine, water is pumped out to keep the mine relatively dry and to allow access to the ore body for extraction. At surface mines, mine water may be pumped from sumps within the mine pit. Surface water is controlled using engineering techniques to prevent water from flowing into the mine. Pumped water may be used in extraction and beneficiation activities (including dust control), pumped to tailings impoundments, or discharged as a waste through an NPDES permit. Because mine water at copper mines is often rich in dissolved copper and other metal ions, some operations pump it to an SX/EW plant to recover the copper values (Cumming 1973).

The quantity of mine water generated at mines varies from site to site. The chemistry of mine water is dependent on the geochemistry of the ore body and the surrounding area. Water exposed to sulfur-bearing minerals in an oxidizing environment, such as an open pit or underground workings, may become acidified. This potential is greatly dependent on site-specific factors.

At underground mines, the quantity of water entering the mine depends on local hydrogeologic conditions. At some facilities, little or no water is encountered. At others, ground water may continually drain into the mine workings. Underground water inflows are often allowed to drain to low areas of the mine where sumps and pumps collect and pump the water from the mine. At some facilities, however, the inflow of water is so great that the capacities of the underground holding and pump mechanisms are exceeded, which leads to mine flooding. In such situations, a ground water-dewatering program is implemented, or the mine is abandoned (Cumming 1973).

Ground water-dewatering programs at both surface and underground mines involve draining the surrounding aquifer using a series of interceptor wells drilled around the mine. The water table is thus lowered around the vicinity of the mine, thereby reducing the flow of water into the mine. After cessation of mine operations, water diversion schemes are generally abandoned, and the mine is allowed to fill with water (Mining Engineering 1988). A more detailed discussion of several surface and underground mine dewatering systems is presented in the *SME Mining Engineering Handbook* (1973), A. Cumming (Chairman of Editorial Board), Society of Mining Engineers, AIME, New York, New York.

Of the mines studied, information pertaining to surface- and mine-water drainage indicated that they typically handled water using diversion ditches, collection and pump back/recycling systems, and/or holding ponds. Data on mine-water management were available for the following mines: Sierrita, Bagdad, Inspiration, Morenci, Bingham Canyon, and Pinto Valley. Sierrita, Bagdad, and Inspiration utilize systems of berms, ditches, and reservoirs to control surface-water runon and runoff. Pinto Valley has installed a diversion trench system lined with riprap to channel overflow caused by a 100-year storm event. The trench system directs overflow from its closed dump leach site to the tailings pond for evaporation. Bingham Canyon's mine-water drainage canals are constructed of epoxy-lined concrete. Morenci installed a sump and pump-back system in the bottom of the pit to capture fugitive mine drainage. The collected drainage is pumped to the leach plant circuit for copper recovery.

While specific information was not found, it is believed that many other operations have mine-water collection/pump-back systems to provide for recovery of dissolved copper, allow for makeup water for facility processes, and ensure compliance with State and Federal regulatory requirements.

SX/EW Sludge

Sludge is the semisolid gelatinous materials (i.e., soft mud, slime, slush, or mire) that can accumulate in SX/EW tanks. These sludges are colloids of suspended material (usually less than 5 angstroms in size) that cannot be easily settled or filtered.

The solvent extraction process specifically generates a "sludge," or, as it is known in the copper industry, "crud" or "gunk." This sludge consists of a solid stabilized emulsion of organic and aqueous solutions from solvent extraction. It is located at the organic/aqueous interface in the settlers and is periodically removed from the system, and centrifuged or otherwise treated to remove the organics. The aqueous solutions and the solids are disposed of and the organics are returned to the solvent extraction circuit for reuse. Depending on the characteristics of the ore body, SX/EW sludges may contain base or precious metals in quantities sufficient for recovery.

Spent Electrolyte

Spent electrolyte is generated during electrowinning activities. Historically, electrolyte went through a stripping step and was subsequently discharged to a tailings pond. Today, due to economics, this effluent is recycled to reduce capital costs associated with the electrolytic acids used in these operations.

Over time, electrolyte in the electrowinning cells becomes laden with soluble impurities and copper. When this occurs, the solution is removed and replaced with pure electrolyte (to maintain the efficiency of the solution and prevent coprecipitation of the impurities at the cathode). Purification of the spent electrolyte is done by electrowinning in liberator cells. Liberator cells are similar to normal electrolytic cells, but they have lead anodes in place of copper anodes. The electrolyte is cascaded through the liberator cells, and an electric current is applied. Copper in the solution is deposited on copper starting sheets. As the copper in the solution is depleted, the quality of the copper deposit is degraded. Liberator cathodes containing impurities (such as antimony) are returned to the smelter to be melted and cast into anodes. Purified electrolyte is recycled to the electrolytic cells. Any bleed electrolyte usually is neutralized with mill tailings and disposed of in a tailings pond (U.S. EPA 1984a).

Spent Leaching Solution

Barren solution (raffinate) is an acidic aqueous solution that has been stripped of copper but still has some carryover of the organic extraction/diluent used in the solvent extraction operation. The raffinate generated at hydrometallurgical plants is typically stored in ponds and recycled to the dump leaching operation. As a result, it does not become a waste until after the closure of the mine. Following mine closure, spent leaching solutions must be disposed of. No information was obtained on the quantity of raffinate generated or recycled at copper mine facilities (U.S. EPA 1984a).

Other Wastes and Materials

In addition to the wastes and materials described previously, extraction and beneficiation operations generate other wastes and materials typical of industrial operations, such as spent solvents, refuse, and used oil.

1.5.2 Waste and Materials Management

Wastes and materials that are generated as a result of extraction and beneficiation of copper ore are managed (treated, stored, or disposed of) in discrete units. For the purposes of this report, waste units are divided into three groups: (1) waste rock piles or dumps; (2) tailings ponds; and (3) spent ore piles once the leaching operation ceases in the case of heap leach operations. These units may be exposed to the environment, presenting the potential for contaminant transport. In addition, mine structures such as pits and underground workings are described in this section as they may expose constituents to the environment and increase the potential for transport.

1.5.2.1 RCRA Units

Waste Rock Piles

Waste rock removed from the mine is stored or disposed of in piles onsite. These piles may also be referred to as mine dumps or waste rock dumps. Often, these units are constructed without liners. Dumps may generate acid drainage if sulfide minerals, oxygen, and moisture are present in sufficient concentrations, and if adequate neutralization potential or other controls in the dump itself are not present.

Tailings Impoundments

Tailings impoundments are surface disposal units for tailings generated during flotation. The following discussion focuses on tailings impoundment design. Slurried tailings may be transported from the mill to the tailings pond by gravity flow and/or pumping through open conduits or pipes. Tailings slurries (both wet and thickened) are highly viscous and abrasive. This causes wear during operation of the tailings transport system. Pipe wear is a significant problem that may be mitigated by the use of rubber-lined steel or HDPE. In addition, the transport system can become plugged with settling solids if the minimum flow velocity is not maintained or if provisions are not made for pipe drainage during mill shutdowns. In most cases, water from the tailings impoundment is recycled to the mill for reuse. The general guidelines detailed below are applicable to the construction and operation of tailings impoundments.

There are three methods of construction for tailings impoundments: upstream, downstream, and centerline. Figure 1-17

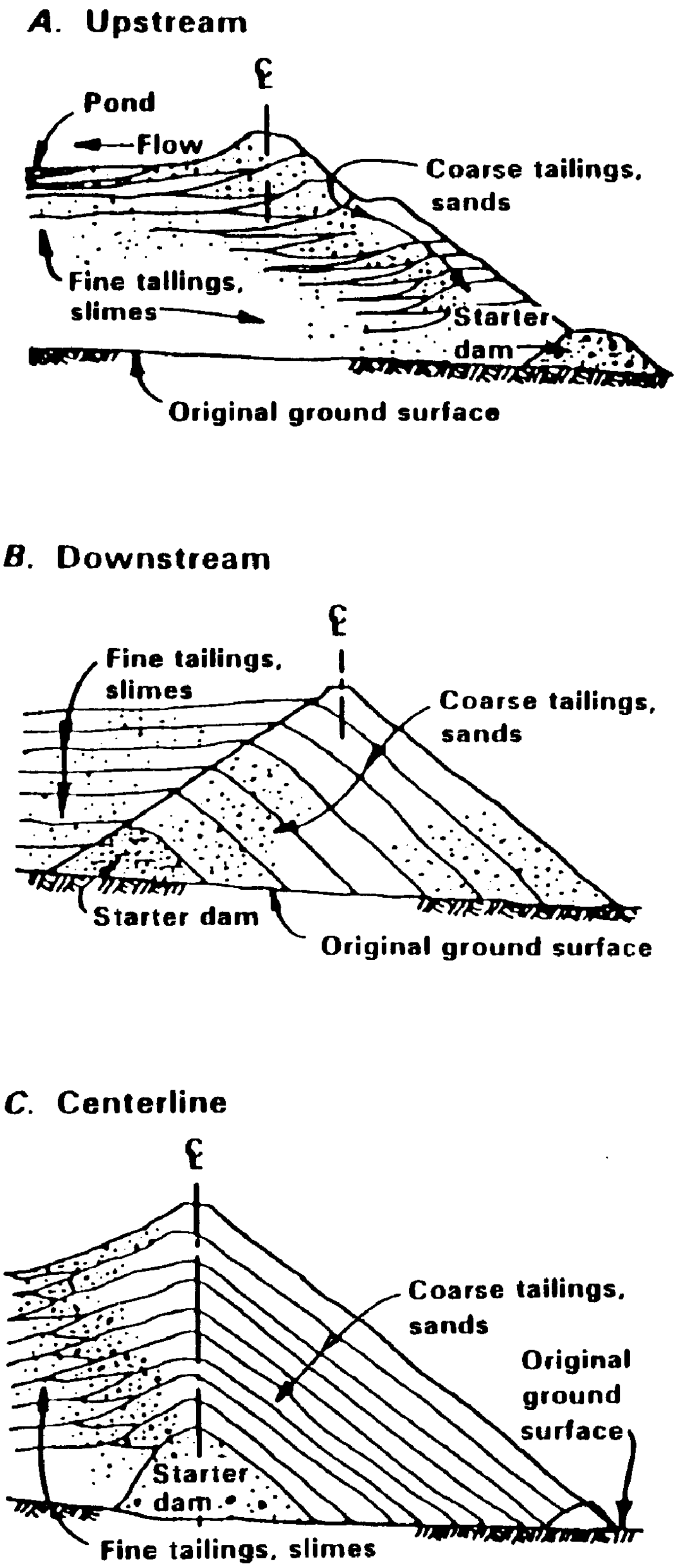

Figure 1-17. Upstream, Downstream and Centerline Methods of Construction

(Source: U.S. DOI, Bureau of Mines, 1984) includes diagrammatic representations of these three construction methods. The choice of construction method is dependent on local topography, the availability of construction materials, and the nature of the tailings. Less common methods of tailings disposal include underground mine backfilling and in-pit backfilling (U.S. DOI, Bureau of Mines 1984. For more information, refer to U.S. EPA 1994, *Design and Evaluation of Tailings Dams*.)

Upstream tailings impoundments are most commonly constructed in the copper mining industry. In this method, the embankment is erected by depositing successive layers of course material on top of the previous dike along the inside of its embankments (see Figure 1-17a). Thus, the centerline of the berm progresses upstream toward the center of the dam, while the outer slope remains stable (U.S. DOI, Bureau of Mines 1984).

In the downstream method, tailings are deposited along the outside of the dike so that the centerline progresses downstream, or away from the center of the pond (see Figure 1-17b). In the centerline method of berm erection, coarse material is deposited on top of the embankment so that the centerline of the dike does not move (see Figure 1-17c). Another modification of this method is to deposit material on both sides of the berms equalizing growth in both directions (U.S. DOI, Bureau of Mines 1984).

Placement of tailings impoundments may be influenced by a number of factors, including location and elevation relative to the mill and the hydrogeology of the area. When an embankment is to be built from tailings material, a starter or toe dam is often required to contain the tailings during the initial stage of deposition. The starter dam may be constructed of pervious or impervious material. If impervious material is used, a filter blanket can be installed, extending under the dam from the toe as far as necessary to drain the interior of the starter dam (Pfleider 1973).

Underdrains may also be installed under the toe dam. The purpose of the underdrain system is to lower the water level at the face of the toe dam and to prevent seepage where the tailings and the crest of the toe dam meet. Underdrains may be constructed with perforated asphalt-dipped pipe (in some instances, a layer of gravel or porous soil may provide sufficient drainage). The pipes are installed with the perforations face down on a suitable bed of gravel in a trench. The pipes are then covered with a layer of washed gravel, and the rest of the trench is filled with washed sand. The depth of the trench and the thickness of the gravel layers should be tailored to suit the existing conditions (Pfleider 1973).

Decanting is accomplished after the tailings have been discharged into the tailings pond. Two methods are available for decanting pond water: decant towers and pumping (usually from floating barges). Decant towers are vertical, concrete risers with intake ports that rise from the bottom of the impoundment to the surface. The tower is connected to a concrete conduit extending from the bottom of the decant tower to beyond the dam toe. In the pumping method, floating barges move to various parts of the pond and collect liquid material. The collected liquid normally is recycled to the mill, either directly or after it has been decanted in a separate decant pond.

Spent Ore Piles

Spent ore from heap and dump leaching may contain residual amounts of lixiviant and associated copper and other metal complexes. The spent ore itself typically contains unleached metals and other minerals characteristic of the ore body. Leach piles are reported to range in size from 20 feet to over 100 feet in height and may cover hundreds of acres and contain millions of tons of leached ore. When active leaching ends, the spent ore becomes a waste.

Side slope and valley dump leaches are located based on topography and design factors. Additional consideration is given to base preparation and geologic factors that may affect unit operation. These factors include the presence of limestone, porous materials (such as sandstone rocks), and geologic faults. Such factors can act to undermine the effective operation of the leaching process. Modern strip or radial leach piles are constructed using methods similar to those used to construct heap leach piles (Thompson, et al. 1984).

Most copper leaching operations are not typically constructed with synthetic liners (i.e., they are dump leach units, rather than heap leach units). However, at some mine sites, such as the Tyrone mine, new dump leaches are being designed with liners and other controls. For example, these dumps, constructed on a devegetated surface, are lined with compacted natural base materials, impermeable bedrock, or clay with drainage and have lined PLS collection systems. All surfaces and natural materials were tested and have permeabilities that range between 10^{-7} and 10^{-9}. Leach operations at the Sierrita, Chino, Morenci, and Pinto Valley mines are located on unspecified bedrock, while the Inspiration mine's dump leaches are situated on granite. According to the mine operators, bedrock is considered impermeable, or of very low permeability. The base of one of the Inspiration mine's newer dump leaches was prepared by devegetation, followed by soil cementation and coating with dilute tar.

In heap leaching operations, the use of specially constructed pads has been practiced to some extent by the copper industry. For example, heap leaching is currently performed at Magma's San Manuel facility. According to the Arizona Department of Environmental Quality (ADEQ), lined pads covering hundreds of acres and containing millions of tons of ore have been a proven technique in the gold mining industry on a scale comparative to the largest copper dump leach operation (Arizona BADCT 1990).

1.5.2.2 Non-RCRA Units

Mine Pits and Underground Workings

Mine pits may or may not be non-RCRA units during the operative life of a mine, depending upon whether or not RCRA wastes are placed in the pits. This is a complicated issue, not lending itself to generalities. Specific questions should be addressed to the nearest EPA Regional office.

Pits and underground workings may be allowed to fill with water when a mine closes or stops operation, since there is no longer a need for dewatering. This accumulated water may acidify through contact with sulfide minerals in an oxidizing environment resulting in acid generation. The acid, in turn, may mobilize metals in the remaining rock. In some cases pits and underground workings are backfilled with waste rock or tailings. The potential for contaminant release is dependent on site-specific factors.

Abandoned underground mines and mine shafts may be unprotected, and the mine may, with time, subside, though this is mostly a problem with historical mines. Deficiencies in mine shaft protection may be caused by the use of unsuitable materials, such as inadequate shaft cappings, or by unexpected occurrences that break capping seals, such as water surges in flooded mines (U.S. DOI, Bureau of Mines 1983a).

Solution Ponds (PLS and Raffinate Ponds)

During the operative life of a mine, solution ponds may or may not be non-RCRA waste management units, depending upon whether RCRA wastes are in the ponds. However, leach solution ponds become non-RCRA units upon mine closure, if they are left or reclaimed in place. (As in mine pits, this is a complicated issue; specific questions should be addressed to the nearest EPA Regional office.) These units may include pregnant solution ponds (where the copper-laden solution is collected), barren solution ponds (where lixiviant solution is held before being dispensed), surge ponds (to manage leachate during high precipitation events), make-up water holding ponds, and associated pipes or trenches. These units may be lined, depending on the quality of the solution contained and the permeability of the underlying formation. Any residual materials become wastes at closure.

PLS and raffinate ponds generally measure several hectares in size and, where the topography permits, are built into natural drainage basins. At most older copper leaching operations, the collection ponds and trenches through which the solutions flow were unlined. In addition, these areas received little or no surface preparation before leaching operations were initiated (U.S. EPA 1989e).

At newer leaching operations, liners have been installed in the collection ponds, and diversion channels have been installed to reduce seepage from the site and to increase the amount of solution recovery. This is particularly true of raffinate ponds that have been constructed within the last 10 years in conjunction with solvent extraction plants. Several facilities have also lined the pregnant liquid collection trenches and ponds. Generally, the trenches have been lined with concrete or a synthetic liner such as polyethylene. The collection ponds are typically lined with gunite, clay, or synthetics (U.S. EPA 1989e).

The San Manuel mine's PLS ponds have an unspecified type of liner. The Tyrone mine's PLS ponds are lined with compacted clay, HDPE, and gunite. The Morenci mine's PLS ponds are lined with 40-mil HDPE, with the Morenci Central Plant feed pond having a double liner and a leak detection and leachate collection system. The upper layer is a HDPE liner, and the lower layer is gunite with a leachate collection system located between the two liners. The PLS ponds at the Sierrita and Ray mines are located on bedrock with no leak detection systems or ground water monitoring systems. Most of the PLS ponds at the Inspiration mine are unlined on a bedrock base with concrete or concrete covering clay core dams. Bingham Canyon mine's PLS ponds are also clay-lined. Pinto Valley mine's PLS ponds are unlined and have rock shell and clay core dams that are keyed and grouted into bedrock. Recently, Cyprus replaced the Bagdad mine's old principal PLS hold pond with a new 100-mil HDPE-lined pond and collection system. In several of these cases, State or Federal regulations required that the PLS sumps be located at the base of heap and that the dump leaches have synthetic liners.

Little information was found for the raffinate ponds of the mines studied. However, raffinate ponds have been constructed in a manner similar to pregnant solution ponds. For example, Morenci's Central raffinate pond is constructed with a 40-mil HDPE liner over a gunite base forming a double liner with leak-detection and leachate-collection systems. Sierrita's raffinate ponds have clay liners over bedrock. Inspiration stores excess raffinate, mine water drainage, and process water in several unlined inactive pits. No data were found on Inspiration's raffinate ponds. It appears that some mines, including Bingham Canyon, pump their raffinate directly to their dump leaches, thus eliminating the need for a raffinate pond.

1.6 ENVIRONMENTAL EFFECTS

Mine pits and underground workings; waste rock piles; tailings and other ponds; spent leach piles are of particular concern in the copper industry, because these are the areas in which toxic contaminants are most commonly found. Not all of these are waste management units, but they have the potential to present harm to the environment and thus, are discussed here. Contaminants associated with these areas may include heavy metals and, from some, acid drainage. These contaminants may degrade ground water, surface water, soil, and air quality during mine operation and after mine closure. A discussion of potential environmental effects associated with copper mining is presented in the following sections, with specific examples included, as appropriate. Actual release incidents occurring at copper mine sites are described in the Damage Case Section of this report.

This chapter does not purport to be a comprehensive examination of damage that may occur or that actually occurred at mining operations. Rather, it is a brief overview of some of the potential problems that can occur under certain conditions. The extent and magnitude of contamination depends on highly variable site-specific factors that require a flexible approach to mitigation. EPA is aware that many of the potential problems can be, and generally are, substantially mitigated or avoided by proper engineering practices, environmental controls, and regulatory requirements.

1.6.1 Potential Sources of Contamination

1.6.1.1 Mine Dewatering

Surface and underground mines may be dewatered to allow extraction of ore. This can be accomplished in two ways: pumping from ground water-interceptor wells to lower the water table and pumping directly from the mine workings. Dewatering can create a hydrologic cone of depression around the mine area and can prevent contamination from reaching the surrounding aquifer. After a mine is abandoned, pumping is generally stopped and the pit or workings fill completely or partially with water. Over time, this may lead to uncontrolled releases of mine water. Mine water can be pH neutral; however, in some cases, it is acidic and contaminated with metals, as well as suspended and dissolved solids.

1.6.1.2 Releases from Active Leach Units

Although a large proportion of the PLS generated at dump leaches is typically collected and recirculated some contaminated leachate may flow or percolate through and contaminate surrounding soils and underlying aquifers. As noted previously, the ore being leached is not considered a waste until leaching ends.

Releases of PLS occur from active leach operations (including dump and heap units, PLS and raffinate ponds, other solution collection ponds, and transport systems). Releases may occur due to infiltration beneath the unit; collection system overflow at the base of operations during snowmelt or large storm events; or failures in piles. Liners may weather, degrade, or puncture, thus losing their effectiveness in preventing releases to ground water and surface water (U.S. EPA 1989e).

Recently, more leach ponds have been constructed with liner and leak detection systems or have been sited in areas which naturally retard releases to ground water. Furthermore, ground water-monitoring systems are being installed with increasing frequency. For example, ground water-monitoring systems are now being required at some copper mines under the Arizona Aquifer Protection Permit program. In New Mexico (since 1976), permits are required for all facilities that the State determines may discharge to ground water.

As noted in Chapter 3 of this report, sulfuric acid is typically used as the lixiviant in the copper industry. The copper concentration in PLS generally ranges from 1.0 to 2.5 grams per liter (g/l) and typically has a pH of approximately 2.0. Quantification of actual fate, transport, and availability to potential receptors is not clear, due to the site-specific nature of these impacts. Similar releases in different settings can have very different environmental impacts.

1.6.1.3 Releases from Leach Units During and After Closure

There remains some potential for releases from dump and heap leach piles during and after closure. After the operation has been closed, shut down, or abandoned, runoff and leachate from the spent ore will continue to be generated. Runoff may contain constituents associated with the ore, such as heavy metals and TSS, and may be highly acidic. Site-specific factors, such as type of ore, precipitation and evaporation rates, soil alkalinity, and bedrock liners under leach units will affect the potential for releases.

Waste leach piles typically have large surface areas and contain highly permeable waste material. These factors act to increase the exposure of waste material to infiltrating liquids. When pyrite and sulfide minerals are exposed to air and water, sulfuric acid may be produced. Sulfuric acid may leach metals, yielding an iron-rich, acidic solution that contains high metals concentrations. If this solution infiltrates the underlying ground surface, it could reach the water table and potentially contaminate ground water.

Operators may continue to collect drainage from inactive dump piles. Information on Management practices for any drainage collected from inactive piles was not obtained for this report. However, the design capacity of collection systems is often based on containment of a specific storm event (e.g., the 10-year or 25-year maximum storm event).

1.6.1.4 Releases from Tailings Impoundments

Mill tailings may be particularly susceptible to leaching due to increased surface area exposure of sulfide/oxide metallic minerals not extracted during the milling operation. Surface-water discharges and seepage from tailings ponds and dams can have elevated concentrations of metals leached from the tailings.

Although the tailings may be neutralized with lime during the discharge or prior to disposal, residual chemical reagents can also remain in the tailings water. Flotation reagents, however, typically are used at very dilute concentrations to promote specific surface chemical reactions, and process wastewater is generally recycled, rather than discharged. In addition, many of the organic collectors and frothers are relatively unstable and would rapidly volatilize or decompose if accidentally discharged into the environment.

Studies conducted by EPA in 1985 found that contaminants from waste storage impoundments (including tailings impoundment) are being released to underlying aquifers at most copper facilities. However, these releases may be caused by the use of outdated waste disposal practices (U.S. EPA 1985d). Many copper mines now are subject to permits that require the use of controls intended to protect ground water and surface-water quality. Such controls include, but are not limited to, liners, drainage collection systems, runon/runoff controls, ground water-interceptor wells, and ground water-monitoring systems.

Many mines have modified their operations to reduce the quantities of waste/wastewaters generated and have improved waste management practices to limit the potential for environmental releases. For example, at the Inspiration Mine, water and waste circuits historically have been managed to maximize the efficient production of copper through leaching and to minimize the water and wastewater disposal costs, while meeting the needs of the smelter and mine. Since 1986, the mine has altered the water and wastewater circuits to reduce the volume of "process wastewater" by isolating the beneficiation circuit from watershed runon, increasing reuse, and maximizing evaporation (U.S. EPA 1987). In addition, at other facilities, liners and ground water monitoring are more frequently used, and better facility-siting procedures are practiced.

1.6.1.5 Acid Drainage

Sulfide copper ores, such as chalcopyrite and bornite, typically contain sulfides of copper, lead, antimony, arsenic, and silver. During the mining of ore, the effects of the weathering may be increased due to the exposure of additional surface area and an increased oxidation rate. When the mineralized material is exposed to water, the oxidization of the sulfide minerals may lead to the formation of sulfuric acid (Doyle and Mirza 1990).

The generation of acids may then act to increase the dissolution, mobilization, and transportation of heavy and toxic metals noted above. Except for iron, all of these are toxic to humans and to aquatic life and are known to accumulate in the environment and concentrate in the food chain (Wills 1981).

Acid drainage refers to drainage that occurs as a result of the natural oxidation of sulfide minerals contained in rock that is exposed to air and water. This phenomenon is often referred to as acid mine drainage (AMD) or acid rock drainage (ARD); however, it is not necessarily confined to extraction activities and can occur wherever sulfide-bearing rock is exposed to air and water. Acid drainage can occur naturally without disturbance of the rock. Not all operations that expose sulfide-bearing rock will result in acid drainage. Acid drainage may not occur if the sulfide minerals are nonreactive or if the rock (such as limestone) contains sufficient natural potential to neutralize the acid (Berkeley Study 1985). Acid generation at mine dumps, ore piles, pits, and underground workings is dependent on the type of sulfide minerals in the ore and the surrounding rock, the climatic conditions, the hydrogeology of the area, and the availability of oxygen.

Water percolating through mine workings or tailings and waste rock piles may leach sulfides from the ore and surrounding rock and result in the formation of sulfuric acid. This acid solution may be discharged to ground or surface water, depending on the hydrology of the site. The acid generation potential, as well as the potential for release of other constituents, is increased after the rock is exposed to the atmosphere (i.e., an oxidizing environment). The rate of acid generation is also influenced by the presence or absence of bacteria. Bacteria, especially Thiobacillus ferrooxidans, are able to oxidize sulfur-bearing minerals. The effect of bacteria is pH-dependent; in some cases, lowering of pH over time produces a favorable environment for specific bacteria (leading to accelerated acid generation) once the pH reaches the appropriate level (Berkeley Study 1985).

In rock dumps, overburden piles, and other mine material piles that typically are unsaturated, acid drainage may start to form immediately. In contrast, because tailings piles may become dewatered over time, oxidation may lead to acid generation beginning long after the tailings have been deposited. In addition, the acid generation potential, as well as the potential for release of other constituents, is higher for tailings than for the in-place ore body because the tailings are finely ground or crushed, thus presenting greater particle surface area for oxidation to occur (Berkeley Study 1985). However, the moisture retention characteristics of tailings may act to inhibit an oxidizing environment.

The oxidation of sulfides may result in heavy metals and sulfosalts being solubilized (these include, but are not limited to, silver, cadmium, cobalt, copper, mercury, manganese, molybdenum, nickel, lead, zinc, arsenic, antimony, and selenium). Some metals will immediately form relatively insoluble oxysalts; others (notably cadmium, copper, and zinc) may accumulate in acid solutions (Berkeley Study 1985). In addition, heavy metals also may be found in any uncontrolled releases from leach circuits (see previous section on acid drainage).

Acid drainage has several characteristics (low pH, contaminants, and latency) that contribute to the severity of its effects. When pyrite is exposed to air and water by mining activity, it oxidizes, releasing acid which (in turn) can leach toxic metals from other minerals associated with the pyrite. The Berkeley study of problem mines in California notes that acidic drainage contains dissolved toxic metals (Berkeley Study 1985).

The latency of AMD is unique among the environmental hazards associated with mining wastes. When tailings are ponded, partial saturation and continual addition of basic material generally prevent acid release during the active life of the mine. After closure, however, acid formation may start and gradually migrate down through the tailings area, sometimes only reaching the ground water years or decades later (Berkeley Study 1985).

Both the acids and dissolved metals contained in AMD may be detrimental to aquatic life. Most sites generating large amounts of AMD also experience permanent elimination of, or damage to, aquatic life. This is typically confined to roughly 10 miles downstream from the point of discharge, although there are often more widespread fish kills during periods of high runoff. As the water moves downstream, the pH of the

AMD may be neutralized and the concentration of dissolved metals is reduced through dilution, adsorption, precipitation, and complexation (Berkeley Study 1985).

In ground water, AMD is diluted, attenuated by neutralization, and, possibly, chemically reduced as it moves from the site. The distance over which this occurs will vary with the reactivity of the aquifer. If ground water is used as a source of drinking water or for other purposes within this distance, the presence of AMD could pose risks to public health and the environment (Berkeley Study 1985). In many cases, particularly in the arid southwest (where many of the copper mines are located), the soils are alkaline and have ample capacity to neutralize acidic solutions and precipitate dissolved metals.

1.6.1.6 Beneficiation Reagents

In solvent extraction, the organic extractants are dissolved in kerosene or another nonreactive diluent. The extraction and stripping operations constitute a closed loop, with continual recycling of the organic extracts. All of the organic chemicals used in solvent extraction have low aqueous solubilities, and many circuits have a filter or similar operation to remove physically entrained organic chemicals from the raffinate. Consequently, loss of extractant is low, and there is little likelihood of significant discharge to the environment (Berkeley Study 1985).

1.6.2 Factors Affecting the Potential for Contamination

The potential for and impacts of environmental releases from wastes associated with copper mining activities are a function of many site-specific factors, including climate, geology, hydrogeology, access to and quality of local surface water, and distance to environmental receptors. Of particular note, many copper mines are located in scarcely populated, semiarid regions, where contaminant mobility is at least partially limited by minimal annual precipitation. However, heavy storm events can occur in these areas, which can increase the potential for releases to surface or ground water.

1.6.3 Affected Media

1.6.3.1 Ground Water/Surface Water

As described previously, mine workings (after mine closure), waste rock dumps, leaching operations, and seepage from tailings impoundments may be sources of ground water contamination. Contaminated ground water may recharge surface-water bodies (streams, ponds, and wetlands), impairing surface-water quality and providing an exposure route for contaminants. This may be especially important in alpine valleys, which usually have shallow alluvial aquifers. Withdrawal of contaminated ground water for use (i.e., drinking, agriculture, etc.) may be an exposure route for contaminants. Seasonal saturation due to snowmelt may also play a role in the transport of contaminants to ground water. It should be noted that, for those states with ground water protection programs, ground water quality is protected by permits that can require controls or demonstrations and verification that operations will have either minimal or no impacts on ground water.

Mine structures, waste rock dumps, and tailings impoundments may be constructed in areas that require the relocation of a surface-water drainage. Tailings impoundments are often constructed by damming valleys or other low-lying areas and dumping or slurrying tailings to these units. These units are normally designed to prevent discharges to ground water and surface water. During infrequent high magnitude storm events, tailing dams may fail, releasing tailings to the local drainage system. Similar high-sediment loads can be generated from waste rock dumps.

1.6.3.2 Soil

Three types of environmental impacts are commonly associated with soils: erosion, sedimentation, and contamination. Erosion and sedimentation may be caused by land disturbances and removal of vegetation related to mining activities. Soil contamination may result from runoff from pits, mine workings, and tailings impoundments, as well as overburden, waste rock, mine development rock, ore, and sub-ore piles. In addition, deposition of wind-blown particulates from piles may also be a source of soil contamination. Contaminated soil may further act as a pathway for contaminant transport to ground and surface water and, in some instances, as a source of air pollutants due to re-entrainment and/or subsequent deposition of particulates.

Erosion related to mining may increase the loading of sediments into receiving streams. Sedimentation may result in elevated mortality rates among salmonoid embryos and fry because of a reduction in the permeability of spawning gravels (which prevents oxygen replenishment) and the blockage of interchange between subsurface and surface water. Indirect effects of increased turbidity and sedimentation include a reduction of photosynthesis and interference with respiratory activity (specifically, of gilled organisms). Gill irritation also exposes fish to infection by fungi and bacteria (Berkeley Study 1985).

Precipitation, adsorption, and settling of particulates reduce metal concentrations in receiving waters, but greatly increase concentrations in sediments. Sediments immediately downstream from mine discharges often contain high concentrations of heavy metals. Resuspension and mobilization of sedimented heavy metals can contribute to downstream metal loadings.

Soils may be contaminated by substances found in seepage or runoff from waste materials. Specifically, tailings may result in heavy metal, radionuclide, or other toxic constituent contamination of soils. Other sources of soils contamination include spills of fuels, flotation reagents, and cleaning solutions, as well as spills of other chemicals often used or stored at the mine site.

1.6.3.3 Air

The primary sources of air contamination at mine sites are fugitive dust emissions from: (1) mine pits and underground workings; (2) overburden, waste rock, mine development rock, ore, and sub-ore piles; (3) dried tailings; and (4) haul roads. During the active life of the mine, water may be applied to piles to control dust and prevent entrainment. After mine closure, revegetation or other stabilizing methods may be used for dust control. In addition to direct human exposure through inhalation, air may provide additional exposure routes through deposition on surrounding soils and/or in local surface water. The potential contaminants are heavy metals, radionuclides, radon, and other toxics.

1.6.4 Damage Cases

Damages resulting from waste management from mining copper and associated minerals have been documented. Under the Comprehensive Environmental Response and Liability Act (CERCLA) (Superfund) and the Clean Water Act (CWA), EPA has documented contamination to ground water, surface water, air, and soil media.

1.6.4.1 National Priorities List

EPA has reviewed the copper mining sites on the NPL. Four sites on the Superfund NPL have problems related to copper extraction and beneficiation: the Celtor Chemical Works site in Humboldt County, California; the Torch Lake site in Houghton County, Michigan; and the Silver Bow Creek and Miltown Reservoir sites, both associated with the Clark Fork Superfund sites in southwestern Montana. Appendix 1-C provides general site descriptions and summaries of the environmental effects associated with these sites.

1.6.4.2 304(l) Sites

Section 304(l) of the Water Quality Act of 1987 requires States to identify bodies of water not meeting applicable water-quality criteria, to identify point source dischargers to these bodies of water, and to develop and require implementation of Individual Control Strategies for those point source dischargers that contribute significantly to exceedance of the water-quality criteria. Anaconda Minerals, Ferri Haggerty Mine, and Kennecott Utah Copper are sites identified under 304(l) as point source dischargers of contaminants related to copper mining activities. A summary of each is provided in Appendix 1-D.

1.7 CURRENT REGULATORY AND STATUTORY FRAMEWORK

Copper mining activities must meet the requirements of both Federal and State environmental regulations. Statutes administered by EPA, such as the CWA [33 United States Code (USC) Section 1251 et seq.] and the Clean Air Act (CAA) (42 USC Section 7401 et seq.), apply to mining sites regardless of their location. The extent to which other Federal regulations apply depend on whether a mining operation is located on federally owned land. Federal regulations exist for operations on lands managed by the Bureau of Land Management (BLM), the Forest Service (FS), the Fish and Wildlife Service (FWS), and the National Park Service (NPS). In addition, the Army Corps of Engineers has promulgated rules for construction and mining activities that have a potential impact on wetlands and navigable waters. Finally, operations must comply with a variety of State requirements, some of which may be more stringent than Federal requirements.

Federal air-quality regulations do not specifically address copper mining, but they do regulate sources of certain types of air pollution. Federal-water quality regulations, on the other hand, include effluent discharge standards for specific types of copper operations. Federal land management agencies have regulations that, in some cases, target particular types of extraction or beneficiation methods (e.g., placer mining turbidity issues). BLM has a policy for management of mining operations using cyanide and other leaching techniques. Similarly, State regulations do not usually target specific minerals, but regulate nonfuel mining in general.

This section summarizes the existing Federal regulations that may apply to copper mining operations. It also provides an overview of the operational permitting, water-quality, air-quality, waste management, reclamation, and wetlands protection regulations in the largest copper-producing State, Arizona.

1.7.1 Environmental Protection Agency Regulations

1.7.1.1 Resource Conservation and Recovery Act

The Solid Waste Disposal Act (SWDA) was first passed on October 20, 1965 (P.L. 89-272). In 1976, the Resource Conservation and Recovery Act (RCRA) comprehensively reenacted and amended the original act (P.L. 94-580, October 21, 1976). The statute was amended again on October 21, 1980, by the Solid Waste Disposal Act Amendments (P.L. 96-482). The next major amendments to the SWDA were the Hazardous and Solid Waste Amendments, enacted on November 8, 1984 (P.L. 98-616). The statute is now collectively referred to as "RCRA" and is intended to protect human health and the environment from problems associated with the management of solid and hazardous wastes. In 1978, EPA's proposed hazardous waste program identified a category of "special wastes," including mining wastes, that are generated in very large volumes. Under the RCRA Subtitle C hazardous waste program, special management standards were proposed for these wastes.

In 1980, prior to the promulgation of final hazardous waste regulations applicable to mining wastes, RCRA was amended to include what is known as the Bevill Amendment, Section 3001(b)(3)(A). The Bevill

Amendment provided a conditional exclusion from the RCRA Subtitle C hazardous waste requirements for wastes from the extraction, beneficiation, and processing of ores and minerals. The exemption was conditioned on EPA's preparation of a report to Congress on the wastes and a subsequent regulatory determination that regulation under Subtitle C was appropriate.

EPA met its statutory obligation with regard to extraction and beneficiation wastes with the 1985 *Report to Congress: Wastes From the Extraction and Beneficiation of Metallic Ores, Phosphate Rock, Asbestos, Overburden from Uranium Mining, and Oil Shale* and a subsequent regulatory determination (51 FR 24496; July 6, 1986). In the regulatory determination, EPA decided that extraction and beneficiation wastes (including copper mining, milling, and leaching wastes) should not be regulated as hazardous wastes but should be regulated under a RCRA Subtitle D program specific to mining wastes.

Although copper processing is beyond the scope of this profile, EPA's regulatory activities related to processing wastes were reviewed for the purpose of gathering information. Through a series of rulemakings in 1989 and 1990, EPA also identified 20 mineral-processing wastes that qualified for the Bevill exemption; the exemption was removed from all other mineral-processing wastes, and as a result, these wastes must be managed as hazardous wastes if they are listed as such or if they exhibit one or more characteristics of a hazardous waste. Three of the 20 exempt wastes were from the primary processing of copper ores: slag, calcium sulfate wastewater treatment plant sludge, and slag tailings.

EPA studied these wastes and in 1990 submitted the *Report to Congress on Special Wastes from Mineral Processing*. In the subsequent regulatory determination (56 FR 27300; June 13, 1991), EPA determined that regulation of these 20 mineral-processing wastes (including the copper-processing wastes) as hazardous wastes under RCRA Subtitle C was not warranted because they exhibit negligible or no hazardous characteristics, pose low risk, and/or are not amenable to the requirements of RCRA Subtitle C.

As discussed above, wastes from the extraction and beneficiation of minerals are generally excluded from RCRA Subtitle C requirements by the Bevill Amendment and EPA's subsequent regulatory determination. EPA interprets this exclusion to encompass only those wastes uniquely related to extraction and beneficiation of ores and minerals; the exclusion does not apply to wastes that may be generated at an extraction or beneficiation facility but are not uniquely related to these operations. For example, waste solvents that are listed as a hazardous waste under 40 CFR 241.31 (Hazardous Wastes from Nonspecific Sources) and are generated at an extraction or beneficiation facility by cleaning metal parts are considered listed hazardous wastes since such parts cleaning is not uniquely related to extraction or beneficiation. These wastes must be managed as any other hazardous waste, subject to the requirements in 40 CFR Parts 260 through 271, or to State requirements if the State is authorized to implement the RCRA Subtitle C program, including those for manifesting and disposal in a permitted facility.

1.7.1.2 Clean Water Act

Under Section 402 of the CWA (33 USC Section 1342), all point source discharges to waters of the United States must be regulated by permit under the National Pollutant Discharge Elimination System (NPDES), with the exception of some storm water discharges covered by the 1987 amendments to the CWA. A point source is defined as any discrete conveyance, natural or man-made, including pipes, ditches, and channels. NPDES permits are issued by EPA or delegated States.

Effluent limits imposed on an NPDES permittee are either technology-based or water-quality-based. The national technology-based effluent guideline limitations have been established for discharges from most active copper mines and mills under the Ore Mining and Dressing Point Source Category (40 CFR Part 440, Subpart J). These regulations govern discharges from all types of copper extraction and beneficiation techniques.

Discharges from regulated operations must meet Best Available Technology/Best Practicable Technology/Best Available Demonstrated Technology (BAT/BPT/BADT) standards for cadmium, copper, lead, mercury, zinc, Total Suspended Solids (TSS), and pH. The specific effluent limitation guidelines for these pollutants are summarized in Table 1-7.

Table 1-7. Effluent Limitation Guidelines for Copper Mines and Mills (40 CFR Part 440)

Pollutant	**BPT**		**BAT**		**NSPS**	
	Daily Maximum	**30-Day Average**	**Daily Maximum**	**30-Day Average**	**Daily Maximum**	**30-Day Average**
Mine Drainage (see 40 CFR 440.132 for definition) [40 CFR 440.102(a), 440.103(a), 440.104(a)]						
Cadmium	N/A		0.10	0.05	0.10	0.05
Copper	0.30	0.15	0.30	0.15	0.30	0.15
Lead	0.6	0.3	0.6	0.3	0.6	0.3
Mercury	0.002	0.001	0.002	0.001	0.002	0.001
Zinc	1.5	0.75	1.5	0.75	1.5	0.75
TSS	30	20	N/A		30	20
pH	6.0 - 9.0		N/A		6.0 - 9.0	
Discharges from mills that employ froth flotation processes alone or in conjunction with other processes to beneficiate copper ores [40 CFR 440.102(b), 440.103(b), 440.104(b)]						
Cadmium	0.10	0.05	0.10	0.05	No discharge is allowed except the volume equal to the net precipitation excess (see below) or when contaminants in recycled water interfere with recovery. In such cases, any discharge is subject to mine drainage limits.	
Copper	0.30	0.15	0.30	0.15		
Lead	0.6	0.3	0.6	0.3		
Mercury	0.002	0.001	0.002	0.001		
Zinc	1.0	0.5	1.0	0.5		
TSS	30	20	N/A			
pH	6.0 - 9.0		N/A			
Process Wastewater from mine areas and mill processes and areas that use dump, heap, *in situ*, or vat leach processes to extract copper from ore or ore waste material [40 CFR 440.102(c), 440.103(c), 440.104(c)]: No discharge is allowed except the volume equal to net precipitation excess (i.e., precipitation on the treatment facility and surface runoff to the treatment facility minus evaporation). In such cases, the discharge is subject to mine drainage limits.						
Combined waste streams (e.g., mine drainage and froth flotation discharge): The quantity and concentration of pollutants are calculated as if the waste streams were discharged separately.						
Storm exemption: Regardless of the applicable limitation, if a facility is designed to contain the flow from the 10-year/24-hour storm event plus normal process wastewater, then discharges resulting from precipitation are allowed to take place, even if they do not meet the limitations or if they otherwise violate 40 CFR Part 440, provided that the facility takes reasonable steps to maintain treatment, minimizes the amount of overflow, and notifies EPA/State under "bypass" and "upset" provisions (see 40 CFR 440.131 for the exact conditions under which discharges are allowed).						

(Source: 40 CFR Part 440)

For discharges addressed by these guidelines, permit writers can establish additional technology-based limitations at a specific facility based on Best Professional Judgment (BPJ). For discharges not addressed by these guidelines, technology-based effluent limits are based solely on BPJ.

The CWA requires each State to develop water-quality standards to protect the designated uses of all receiving waters in the State. Permit writers must determine whether technology-based effluent limitations (i.e., BAT/BPT/BADT) are adequate to ensure that applicable water-quality standards are met. Where technology-based limits are not sufficiently stringent, water-quality-based effluent limitations must be developed. As a result, an NPDES permit may include technology-based effluent limitations for some pollutants and water-quality-based effluent limitations for others.

Contaminated storm water runoff from some mining operations has been documented as causing water quality degradation. In the past, point source storm water discharges have received limited emphasis under the NPDES program. However, EPA recently promulgated regulations that specifically address point source discharges of storm water from industrial facilities, including active and inactive/abandoned mine sites (55 FR 47990; November 16, 1990). These regulations require NPDES permits for all discharges of contaminated storm water. EPA has developed general permits that can authorize storm water discharges from mining facilities. EPA Regions and authorized States also may develop general permits or require individual storm water permits.

Some discharges from mine sites do not meet the traditional definition of a point source discharge. Specifically, diffuse runoff from tailings piles, overburden, waste rock piles, ore and sub-ore piles, and other mine areas often is not controlled through a discrete conveyance. As a result, this type of discharge frequently has been considered a nonpoint source discharge. Under Section 319 of the CWA, states are required to prepare nonpoint source assessment reports and to develop programs to address nonpoint sources on a watershed-by-watershed basis. Each state must report to EPA annually on program implementation and resulting water-quality improvements.

1.7.1.3 Clean Air Act

Under the CAA, Section 109 (42 USC Section 7409) established National primary and secondary air-quality standards for six criteria pollutants. These are known as the National Ambient Air Quality Standards (NAAQS). NAAQS set maximum acceptable concentration limits for specific airborne pollutants, including lead, nitrogen oxides, sulfur dioxide, carbon monoxide, ozone, and suspended particulate matter of less than 10 microns in diameter. To attain the air-quality goals set by CAA, States and local authorities were given the responsibility of bringing their regions into compliance with NAAQS (see CAA Section 110, 42 USC Section 7410). In addition, States were granted the authority to promulgate more stringent ambient-air-quality standards. Although fugitive dust control is not an explicit requirement of the CAA, most States require fugitive dust suppression measures as part of their State Implementation Plans (SIPs) to achieve NAAQS for particulate matter. Of the major mining States, only Alaska has no specific requirement to control fugitive dust.

Mining operations located in areas where NAAQS for one or more criteria pollutants are being exceeded ("nonattainment" areas) may be required to apply "reasonably available control technology" to limit the release of airborne pollutants from industrial and land-disturbing activities. Major new and modified sources constructed in areas where the NAAQS are not exceeded must undergo preconstruction review and apply "best available control technology." Such sources constructed in nonattainment areas are subject to the more stringent "lowest achievable emission rate" and may be required to obtain emissions offsets.

New Source Performance Standards (NSPS), authorized by Section 111 of the CAA, have been promulgated for metallic mineral-processing plants and can be found in 40 CFR Part 60, Subpart LL. Processing plants are defined as "any combination of equipment that produces metallic mineral concentrates from ore; metallic mineral processing commences with the mining of the ore." However, all underground processing facilities are exempt from NSPS. Also, NSPS particulate emission standards apply to stack emissions, but not to fugitive emissions. NSPS require operations to contain stack-emitted particulate matter in excess of 0.005 grams per dry standard cubic meter (dscm). In addition, stack emissions must not exhibit greater than 7 percent opacity, unless the stack emissions are discharged from an affected facility using a wet scrubbing emission control device. However, on or after 60 days following the achievement of the maximum production rate (but no later than 180 days after initial startup), operations must limit all fugitive emissions created during operation to 10 percent opacity.

State ambient air standards promulgated to meet or exceed Federal NAAQS are generally maintained through permit programs that limit the release of airborne pollutants from industrial and land-disturbing activities. Fugitive dust emissions from mining activities are often regulated through these permit programs, typically by requiring dust suppression management activities (e.g., water sprays).

Several other pollutants are regulated under the CAA by the National Emission Standards for Hazardous Air Pollutants (NESHAP). NESHAP provisions address health concerns that were considered too localized to be included under the scope of the NAAQS. The 1990 amendments to the CAA, however, require new emission limits for many airborne toxicants, including cyanide. These standards will be applied to specific industrial categories over the coming years. It should also be noted that the scope and stringency of NAAQS were increased under the 1990 CAA amendments.

Under the 1990 amendments to the CAA, Congress required EPA to establish technology-based standards for a variety of hazardous air pollutants, including cyanide compounds. In November 1993, EPA published a list of source categories and a schedule for setting standards for the selected sources. Furthermore, if a source emits more than 10 tons per year of a single hazardous air pollutant or more than 25 tons per year of a combination of hazardous air pollutants, the source is considered a "major source." Major sources are required to utilize the maximum available control technology (i.e., BAT) to control the release of the pollutants (CAA Section 112).

1.7.2 Department of the Interior

1.7.2.1 Bureau of Land Management

Copper mining operations on Federal lands generally are conducted on mining claims located pursuant to the General Mining Laws (the number of copper operations actually located on Federal lands was not determined). Under the 1872 Mining Law, a person has a statutory right to go upon the open (unappropriated and unreserved) public lands of the United States for the purpose of prospecting for, exploring, developing, and extracting minerals. Once a person has made a valuable mineral discovery and has properly located the claim pursuant to the mining laws, the person has broad possessory rights to develop the minerals upon which the claim was based.

Because of the broad nature of the claimant's possessory rights, the Federal agencies having management responsibilities over the lands upon which the claim is located cannot, in most cases, restrict mining operations entirely. Nonetheless, the surface managing agency can subject the mining operations to reasonable regulation to prevent "unnecessary and undue degradation" of Federal lands and resources. BLM's authority to regulate mining claim operations under this "unnecessary and undue degradation" standard derives from the Federal Land Policy and Management Act of 1976 (FLPMA), the statute which sets out BLM's general land management and planning authority.

BLM's general surface management regulations governing mining claim operations, which include copper mining operations, are found in 43 CFR Part 3809. These regulations cover general design, operating and reclamation standards, monitoring requirements, bonding requirements, environmental review requirements, and remedies for noncompliance. They establish three general use categories for mining operations, each eliciting different levels of oversight by BLM. These categories are:
(1) casual use operations (i.e., those that normally result in only negligible disturbances of Federal lands and resources and that require no prior notice to or approval from BLM), (2) notice-level operations (i.e., those that involve disturbances of 5 acres or less for which the operator must notify BLM prior to commencing surface disturbing activities), and (3) plan-level operations (i.e., those that involve disturbances of greater than 5 acres, and operations in some specified areas, for which the operator must obtain BLM approval of a plan of operations prior to commencing activity).

All operations, including casual use and operations under either a notice or a plan of operations, must be conducted to prevent unnecessary or undue degradation of the Federal lands. All operations must also be reclaimed and must comply with all applicable State and Federal laws, including air- and water-quality standards such as those established under the CAA and the CWA, and standards for the disposal of solid waste established under RCRA.

All mining operations are subject to monitoring by BLM to ensure that they do not cause unnecessary or undue degradation, and that all operators are responsible for fully reclaiming the area of their claim. In early 1992, BLM promulgated its *Solid Minerals Reclamation Handbook*, which is intended to ensure uniform reclamation standards on Federal and Indian lands (U.S. DOI, BLM 1992). Short-term goals are to stabilize disturbed areas; long-term reclamation goals are to restore (by shaping, stabilizing, revegetating, or otherwise treating) disturbed areas to provide a "self-sustaining, safe, and stable condition that provides a productive use of the land which conforms to the approved land-use plan for the area" (U.S. DOI, BLM 1992). The guidelines cover reclamation of exploration, development, and mining of all solid minerals, including copper. They require operators to develop, in consultation with regulatory agencies, reclamation plans that will serve as "binding agreements." These should be submitted with the plan of operations, notice, exploration plan, or mining plan and should include the requirements and mitigation measures recommended in Environmental Assessments (EAs) or Environmental Impact Statements (EISs).

By an internal Instruction Memorandum (IM) issued in 1990 (U.S. DOI, BLM 1990a), BLM established uniform standards for surface management of mining operations that use cyanide and other chemical leaching methods for mineral extraction on public lands. (Directed primarily at gold heap leaching operations, the policy also applies to "operations that use other leaching techniques for extractive purposes" and that use "potentially toxic or lethal concentrations in solution as the leachate medium." It was not determined if BLM applies the policy to copper-leaching operations that use sulfuric acid as the leachate medium.) This IM directs BLM Area and District offices to inspect all such operations at least four times a year. All facilities employing cyanide or other leaching techniques must be fenced and must ensure protection of the public, wildlife (including migratory birds), and livestock. Other requirements include the following:

- Facilities must be designed to contain the maximum operating water balance in addition to the water from a 100-year, 24-hour storm event. Containment ponds must be included in all containment systems.

- Leakage detection and recovery systems must be designed for heap and solution containment structures. Monitoring of ground and surface water through closure and final reclamation is required.

- Cyanide solution and heaps must be neutralized or detoxified.

BLM policy for bonding was established by a 1990 IM (U.S. DOI, BLM 1991). Under this IM, BLM does not require bonds for most casual use or notice-level operations. All plan-level operations, regardless of operation type, are required to post a bond. Bond amounts are set at the discretion of BLM (up to $2,000 per acre, except as noted below), depending on the nature of the operation, the record of compliance, and whether it is covered by a satisfactory State bond. A 100-percent reclamation bond is required from all operators who have established records of noncompliance. Additionally, the IM requires the posting of a 100-percent reclamation bond for all operations that use cyanide or other leachates. The 100-percent bonding requirement applies only to portions of the operation that use cyanide or other leachates (i.e., leach heaps, pads, or dumps). (It was not determined if BLM applies this 100-percent bonding requirement to copper leaching operations.)

Mining claims located in BLM wilderness study areas are generally subject to stricter regulation than other mining claims. The regulations covering mining in wilderness study areas are found in 43 CFR Part 3802. The IM discussed above for cyanide management applies to relevant operations in wilderness study areas in addition to the 43 CFR Part 3809 regulations.

BLM has the authority to issue leases for copper on certain acquired (as opposed to public domain) lands. Although this is rarely done, such leases would be covered by the general regulations applicable to hardrock leasing found in 43 CFR Part 3500.

1.7.2.2 National Park Service and Fish and Wildlife Service

Generally, location of new mining claims is prohibited in most areas managed by NPS and FWS (both of which are under the DOI). Regulations in 36 CFR Part 9 govern activities on land managed by NPS under patented and unpatented mining claims already in existence prior to the time the lands were included within units of NPS. The regulations in 50 CFR Part 29 govern mining activities under mineral rights on lands managed by FWS that vested prior to the acquisition of the land by the United States. It was not determined whether any copper operations are located on lands administered by NPS or FWS.

1.7.3 Department of Agriculture

1.7.3.1 Forest Service

Although BLM has general management authority for the mineral resources on FS lands, BLM regulations governing activities under mining claims do not apply to units of FS. Instead, surface uses associated with operations under mining claims on FS lands are governed by regulations in 36 CFR Part 228, Subpart A. FS regulations generally mandate that operations under mining claims be conducted to minimize adverse environmental impacts on FS surface resources.

FS regulations are similar to BLM regulations and provide for FS consultation with appropriate agencies of the DOI in reviewing technical aspects of proposed plans of operation. However, FS regulations differ in that the general use categories do not specify acreage, as opposed to BLM's regulations, where the use category is based on the acreage disturbed. FS regulations require that persons proposing to initiate any operations that might disturb surface resources must file a notice of intent to operate with the district ranger with jurisdiction over the area to be affected. If the district ranger determines that the operations will likely cause a significant disturbance of surface resources, the operator must submit a proposed plan of operations. Neither a notice of intent to operate nor a proposed plan of operations are required for the locating or marking of mining claims; mineral prospecting or sampling that will not cause significant surface disturbance; operations that do not involve the use of mechanized equipment or the cutting of trees; or uses that will be confined to existing roads.

A proposed plan of operations must include a thorough description of the proposed site, the nature of the proposed operations, and measures for meeting environmental protection requirements. Operations must comply with applicable environmental laws and must, where feasible, minimize adverse environmental effects on FS resources. FS conducts environmental assessments of proposed plans of operations and, if necessary, prepares EISs pursuant to the National Environmental Policy Act.

The regulations specify standards for reclamation and provide that the district ranger may require a reclamation bond to cover the cost of reclamation. Where State bonding regulations exist, FS has established memoranda of understanding with the States to prevent double bonding. In these cases, the bond amount must meet the more stringent standard, whether it is that of the State or FS. Regulations specific to mining operations on FS Wilderness Areas are found in 36 CFR Part 293.

1.7.4 Army Corps of Engineers

Under Section 404 of CWA, the U.S. Army Corps of Engineers regulates activities with the potential to physically restructure wetlands or "navigable waters." In 1986, the Corps and EPA entered into an agreement (updated in 1990) on the definition of "fill material" for Section 404 permitting. The agreement provided that jurisdiction of some mining discharges would be determined on a case-by-case basis. Since then, the Corps has been responsible only for dredge and fill activities accessory to mining operations. These activities can include construction of sediment ponds and roads and placement of waste materials into "waters of the U.S." (which can include ephemeral drainages). Mining operations subject to Section 404 are generally regulated through Nationwide general permits issued by the Corps.

1.7.5 State Programs

1.7.5.1 Arizona

Arizona's optimal system presented in the Best Available Demonstrated Control Technology (BADCT) Draft guidance is described below for each type of mining unit. It is important to note that, besides the optimal technologies, the Arizona BADCT Draft guidance also presents several other less-protective options for each technology described. In addition, the BADCT Draft guidance presents alternative systems that may be considered as optimal under the specific conditions described for each scenario. These alternative systems may be substituted if justified by site-specific conditions. Finally, optimal BADCT recommendations only represent guidelines, and the specific design for each site is to be based on:

- Site suitability
- Extent to which site characteristics can function to control discharges
- Discharge control performance of other design elements
- Chemical characteristics of the discharge.

Arizona Optimal Waste Dump Disposal Technology

As previously stated, only "mining overburden returned to the excavation site, including any common material which has been excavated and removed from the excavation site and has not been subjected to any chemical or leaching agent or process of any kind" is exempt from BADCT requirements (Arizona Revised Statutes 49-250.B.5). However, no specific BADCT recommendations are stipulated for waste rock dumps in the Arizona BADCT Draft guidance.

Arizona Optimal Leach Dump Disposal Technology

The key minimal BADCT components in copper leaching operations are those design elements that address natural subgrade bases and surface-water run-off controls. The optimal technologies for these design elements are similar to those applicable to tailings ponds and heap leach units.

Arizona Optimal *In Situ* Disposal Technology

Control of the leach solution is the primary consideration. Ground water modelling and pilot-scale testing; controlled recovery of PLS by overpumping; proper grouting of abandoned exploration, injection, and recovery wells; and aquifer restoration are the key BADCT design components for *in situ* leaching operation design.

Arizona Optimal Tailings Disposal Technology

The key elements of BADCT for tailings disposal address depositional practices, tailing dam and impoundment design and construction, reuse of tailings water, and surface-water control. The specific design of tailings impoundment must accommodate a variety of factors, including the topography of the site, the availability and cost of materials, and the required storage capacity of the impoundment. Size is generally the main factor in the design of the tailing pond.

The treatment and deposition of tailings can effectively reduce contaminant levels in leachate, tailings permeability, and hydraulic head within tailings piles. Arizona's BADCT Draft guidance dictates that tailings should be thickened prior to deposition to remove the maximum amount of water practical for reuse in the mineral processing operation (i.e., using as little water as necessary to transport the tailings to the impoundment yet enough water to minimize pipeline wear). Thickened tailings may require treatment to neutralize pH or reduce contaminant levels. They should be deposited upstream from the tailings dam in a manner that achieves maximum size separation. Size separation will allow pervious, coarse material to be deposited close to the dam, while finer silt and clay-sized material is deposited further upstream.

Arizona's BADCT Draft guidance suggests recycling water from tailings ponds, after fine sediments have settled out, to reduce seepage. Several recommended methods are available to collect this water such as decant towers, bilge pumps, siphon systems, and drainage ditch systems. Removal of the water from the tailings impoundment can lessen the hydraulic head within the tailings pile and can help prevent the infiltration of leachate below the pile. The relative effectiveness of each is dependent on a number of site-specific factors; the main factor is the configuration of the impoundment itself. Proper maintenance is required for any of these systems to operate effectively.

Leachate collection systems at the base of tailings impoundments are recommended to collect leachate for reuse or treatment and disposal; this will reduce the potential for infiltration. Dams should be constructed to prevent failure and/or surface discharges since such discharges may adversely affect surface and ground water. Dams should be constructed in a stable area with a substrata able to bear the weight of the dam system. The types of materials used to construct the dam should prevent seepage through the dam when compacted. The height of the dam must be sufficient to maintain enough freeboard to allow for the retention of tailings water and runon. Dam areas also should be protected from erosion and revegetated during the lifetime of the facility. This also should be done after closure, where practicable. Finally, the dam design should be integrated with the design of the leachate collection system.

In the copper industry, there is no single system which is best for all conditions. However, all acceptable systems must be able to retain waste material under the most intense storm event (100-year flood) expected during the life of the facility.

Site preparation, such as vegetation clearing and rough grading the surface, provides an added benefit in seepage reduction. The BADCT recommendation for copper tailings pond construction incorporates the use of the fines fraction of the tailings (slimes), as part of the liner system to seal the floor of the impoundment.

The effectiveness of using slimes as a liner is dependent on the particle-size distribution of the tailings solids. Laboratory or pilot-scale tests can verify whether or not the tailings slimes will perform well as a low-permeability liner for the impoundment.

Arizona Optimal Heap Leach Disposal Technology

The minimal BADCT requirements for heap leach operations are single liners of natural or synthetic material, on prepared subgrade with surface-water controls. The topography of the leach site generally determines the pad configuration. In some steep terrain, construction or installation of a liner may be technically infeasible and may not be necessary if site characteristics achieve performance similar to a liner system. In relatively flat areas, pads can be designed to drain to a single solution collection ditch outside the heap.

The degree to which subgrade preparation is necessary is dependent on the liner type and thickness, the physical characteristics of any overliner (drainage blanket), and characteristics of the ore to be leached. The extent of compaction is dependent on several factors: soil type; ability of the soil to function as an impermeable liner; and chemical attenuation. Installation of synthetic liners requires a smooth, stiff subgrade to avoid punctures and tears of the liner. Geotextiles installed beneath the liner may also be used to protect liner integrity.

Many heap leach operations utilize synthetic liners to maximize leach solution recovery. The appropriate liner type and thickness should be determined to maximize liner integrity based on consideration of the loading weight of the heap, the puncture properties of the subgrade and the overliner, and the resistance to chemical degradation by the leaching solution.

In addition to these recommendations, an overliner of porous sand or geonet/geotextile material is also recommended for heap leach designs to protect liner integrity. Drainage pipes may also be necessary to reduce head on the liner and promote collection of PLS.

Arizona Optimal Vat Leach Disposal Technology

Although the Arizona BADCT Draft guidance does not specifically address "vat leaching," it does designate vat leaching operations as facilities that may impact ground water. As such, they are required to follow BADCT for pollution prevention. Other facilities associated with vat leaching operations (such as chemical storage, processing areas, and pipelines) are also addressed in sections of the BADCT Draft guidance.

Arizona Optimal Leach Circuit Surface Impoundments

The technologies presented here are generally appropriate for new facilities. However, it may be feasible at some existing sites to employ some of these controls (depending on the amount of discharge reduction that could be achieved). The same design parameters apply to the construction of all types of leach circuit impoundments (including the pregnant, barren, and makeup water impoundments). Therefore, the BADCT Draft guidance document described below is applicable to all of these units.

Surface impoundments should be designed and constructed with double liners and leak detection systems installed between the liners, unless site conditions provide for equally effective alternatives. Such alternatives might include siting on impervious bedrock or application of other technologies (i.e., grouting). Liners may be constructed of two synthetic liners or with a primary synthetic liner and a natural secondary liner. The specific types of liner material (i.e., synthetic or natural) and thicknesses should be based on site-specific conditions and chemical and physical characteristics of the materials to be contained in the impoundment.

All new ponds should be designed with leak detection and collection systems. For double-lined ponds, these systems generally consist of a pervious layer between the liners, with any type of seepage being collected and recovered. The specific design of each system will depend on the site topography. Other types of systems may rely on preparation of the subgrade and surface-water controls similar to those described above.

Leach circuit auxiliary units, which are used for industrial chemical storage, waste discharge, waste storage and disposal, or wastewater treatment, should be designed according to Arizona's BADCT Draft guidance. Specifically, acid solutions used in the electrolytic process should be stored and handled in a manner consistent with the chemical storage BADCT considerations. Plastic pipes or pipes coated with other inert materials should be used in all leach circuit pumping. A single liner (synthetic or natural) may be used for solution drainage ditches.

Usually, the SX/EW units are combined into one leach circuit. In some cases, where the mine uses a cementation-type leach circuit, it may have an independent electrowinning circuit. The same BADCT design recommendations described above for the leach circuit impoundment apply to all electrowinning processing facilities with separate surface impoundments. Additionally, all ancillary equipment (i.e., plumbing and drainage ditches) must meet the requirements described above.

Arizona Optimal Mine Water Handling Technology

The operator should ensure that mine-water collection points or holding ponds have sufficient holding capacity for storm events. In areas where the quantity of mine water generated exceeds the need for process water, it may be discharged in an acceptable manner. The decision on whether to discharge should be based on types and concentrations of pollutants compared to State and Federal CWA standards.

Mine sites should be designed with offsite and onsite runoff controls to prevent discharges caused by runoff from precipitation. Surface-water containment areas (such as holding ponds) are required to be capable of handling a 100-year, 24-hour storm event; and drainage ditch diversions systems capable of controlling the 100-year, 24-hour storm event. Surface-water drainage systems should be designed separately from the process-water systems (such as tailings impoundments). At many mines, the tailings pond may also act as a surface-water holding pond during storm events. The use of a tailings pond for storm water control must be carefully planned to prevent increasing the potential for overtopping, erosion, berm washout, or seepage. This method of surface-water control may be advantageous as a source of makeup water for mines located in dry climates where water supplies are scarce.

Surface-water runon should be rerouted by drainage diversion systems whenever possible to avoid unnecessary flooding of process areas/units. The design of runon control systems should be sufficient to handle the maximum amount of runon generated by a 100-year, 24-hour storm event (as necessary to protect the specific watershed). However, at the same time, a facility's need to "harvest" additional water also should be considered.

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APPENDIX 1-A

SUMMARY OF COMMENTS AND EPA RESPONSES

Comments on the Draft Industry Profile and EPA Responses

A draft of the *Industry Profile: Copper* was provided for review and comment to the following organizations: U.S. Department of the Interior (DOI) Bureau of Mines, the Western Governors' Association, the Interstate Mining Compact Commission, the American Mining Congress (AMC), and environmental organizations for their review and comment. A large number of comments were submitted to the U.S. Environmental Protection Agency (EPA) by the following 10 reviewers: U.S. DOI Bureau of Mines and Bureau of Land Management; Arizona Department of Environmental Quality (ADEQ); State of New Mexico Energy, Minerals, and Natural Resources; State of Utah Department of Natural Resources, Division of Oil, Gas, and Mining; ASARCO Incorporated; Cyprus Minerals Company; Kennecott Corporation; Magma Copper Company; and Phelps Dodge Corporation. The comments included technical and editorial changes, as well as comments on the scope of the profile and how it relates to authorities provided under the Resource Conservation and Recovery Act (RCRA) Subtitle D.

Because several general concerns were raised by a number of commenters, EPA has grouped the comments into two categories. The first includes five general concerns that were raised by all commenters. These are addressed in the first section below. The second category of comments includes technical comments on this profile, which were raised by specific reviewers, rather than the group as a whole. These are addressed in the second section below. All other comments, including minor technical and marginal notes, have been incorporated into the revised profile; EPA believes these comments have served to improve the document's accuracy and clarity. EPA would like to thank all the agencies, companies, and individuals for their time and effort spent reviewing and preparing comments on the profile.

General Issues Pertaining to All Profiles

1. Comment: Several commenters objected to the use of hypothetical phrases like "may cause" or "may occur." Their use was characterized as misleading and inappropriate in describing environmental impacts in an industry profile of this type.

   Response: EPA believes that the descriptions of conditions and impacts that may occur regarding potential effects is appropriate in many cases, since the intent of the relevant sections of the profiles is to describe potential impacts that may occur as a result of extracting and beneficiating ores and minerals. As noted in the responses to related comments below, EPA has extensively revised the sections of the profiles addressing environmental effects. They are now more focused and direct; they describe, in general terms, a number of specific types of impacts that can occur under particular conditions or in particular environments.

2. Comment: A related issue raised by commenters was that EPA did not balance the profiles by describing environmental protection practices currently followed by the mining industry. Instead, the commenters were critical that EPA selected the worst sites to describe, which represent only a small number of mines.

   Response: EPA believes the profile (and related site reports) represents current environmental management practices as described in the current literature. EPA discusses current waste management practices in specific site visit reports, which are part of the Copper Technical Report.

3. Comment: Reviewers were concerned that the sites described in the discussion of environmental effects were under some other regulatory authority [e.g., the Comprehensive Environmental Response, Compensation, and Liability Act (CERCLA)].

   Response: As noted above, the relevant sections of the profiles have been revised extensively. However, EPA believes that, with proper qualification, sites under other regulatory authorities, including CERCLA, are relevant to any examination of actual or potential environmental effects.

4. Comment: Commenters were concerned that the profiles considered materials other than those considered "wastes" under RCRA.

   Response: EPA believes it is proper to consider in the profile both wastes and materials that have the potential to pose risks to human health and the environment.

5. Comment: Many commenters recommended that the mitigating measures used to control potential environmental impacts be discussed.

   Response: As noted above, EPA has revised the relevant sections of the profiles, including the addition of language that emphasizes the site-specific nature of potential environmental impacts. The regulatory section of the Profile discusses permit requirements, which often establish specific mitigation requirements.

Technical Issues Specific to the Copper Profile

A large number of technical comments were received on the Preliminary Draft of the Profile. These comments addressed both minor and major technical issues. With the following exceptions, all technical comments have been incorporated into the revised Draft Profile.

6. Comment: The details of the flowsheet in Section 1.4 are out of date.

   Response: Information for the Profile was assembled from publicly available information. A generic copper flowsheet showing extraction and beneficiation was not located for this draft.

7. Comment: Expand discussion of State regulations by including more information and expanding the number of States covered.

   Response: As noted in the text, EPA has described regulations in the Nation's largest copper-producing State. The description of Arizona regulations has been revised to reflect Arizona DEQ comments.

8. Comment: In the section on the Ray mine (Appendix B), it was requested that the discussion pertaining to settlement cracks in tailings disposal ponds be deleted.

   Response: The company did not explain their rationale for this request. Consequently the brief discussion regarding the tailings dam remains in the current text.

APPENDIX 1-B

CASE STUDIES OF PUBLISHED INFORMATION ON MINE WASTE MANAGEMENT PRACTICES AT COPPER MINES

1. Kennecott Utah Copper; Bingham Canyon Mine; Salt Lake County, Utah
2. ASARCO Inc.; Ray Complex; Pinal County, Arizona
3. Cyprus Sierrita Corporation; Sierrita Mine; Pima County, Arizona
4. Cyprus Bagdad Copper Company; Bagdad Mine; Yavapai County, Arizona
5. Magma Copper Company, Pinto Valley Mining Division
6. Cyprus Miami Mining Corporation, Cyprus Miami Mine and Smelter, Gila County, Arizona
7. ASARCO Inc.; Mission Mine; Pima County, Arizona

This appendix presents information concerning the waste management practices employed by the copper mining industry. Specifically, site descriptions are presented for seven of the top-producing copper mine sites (as now listed in Table 1-1) in the United States. Three others are the subject of site visit reports in subsequent chapters of this document. These ten copper mines produced over 90 percent of all domestic copper in 1992.

DATA SOURCES

Copper mining companies, the Arizona Department of Environmental Quality (ADEQ), the U.S. Department of Interior (DOI) Bureau of Mines, the U.S. Bureau of Land Management, and published materials provided the information presented in the site descriptions. The scope of this appendix includes extraction and beneficiation operations; for information on processing operations, see the U.S. Environmental Protection Agency's (EPA's) 1990 *Report to Congress*.

EPA directly contacted two mining companies to obtain information about the current waste management practices at five mining facilities, these include: the Cyprus Mining Company (Sierrita, Bagdad, and Cyprus Miami mines), and the American Smelting and Refining Corporation (ASARCO) (Mission mine). Each company provided varying levels of detail on its mines, due to time constraints on data collection activities. Some material for this appendix originated from U.S. DOI, Bureau of Mines, which collected data on mine sites for a technical report entitled *Methodology to Measure the Economic Impact of Mining and Mineral Processing Waste Regulations* (U.S. DOI, Bureau of Mines 1990a). The Cyprus Mining Company gave permission to U.S. Bureau of Mines to release data to EPA for use in this report.

Computer literature searches for this project were conducted. The purpose of these searches encompassed determining what information is publicly available on waste treatment/management technologies associated with copper extraction and beneficiation operations. The results of these literature searches, coupled with follow-up information gathering, provided much of the information for the site descriptions.

DISCUSSION OF THE MINE WASTE MANAGEMENT PRACTICES AT NINE COPPER MINE SITES

1. Kennecott Utah Copper; Bingham Canyon Mine; Salt Lake County, Utah

The Kennecott Utah Copper Corporation's Bingham Canyon mine is located near Salt Lake City (see Figure 1-18

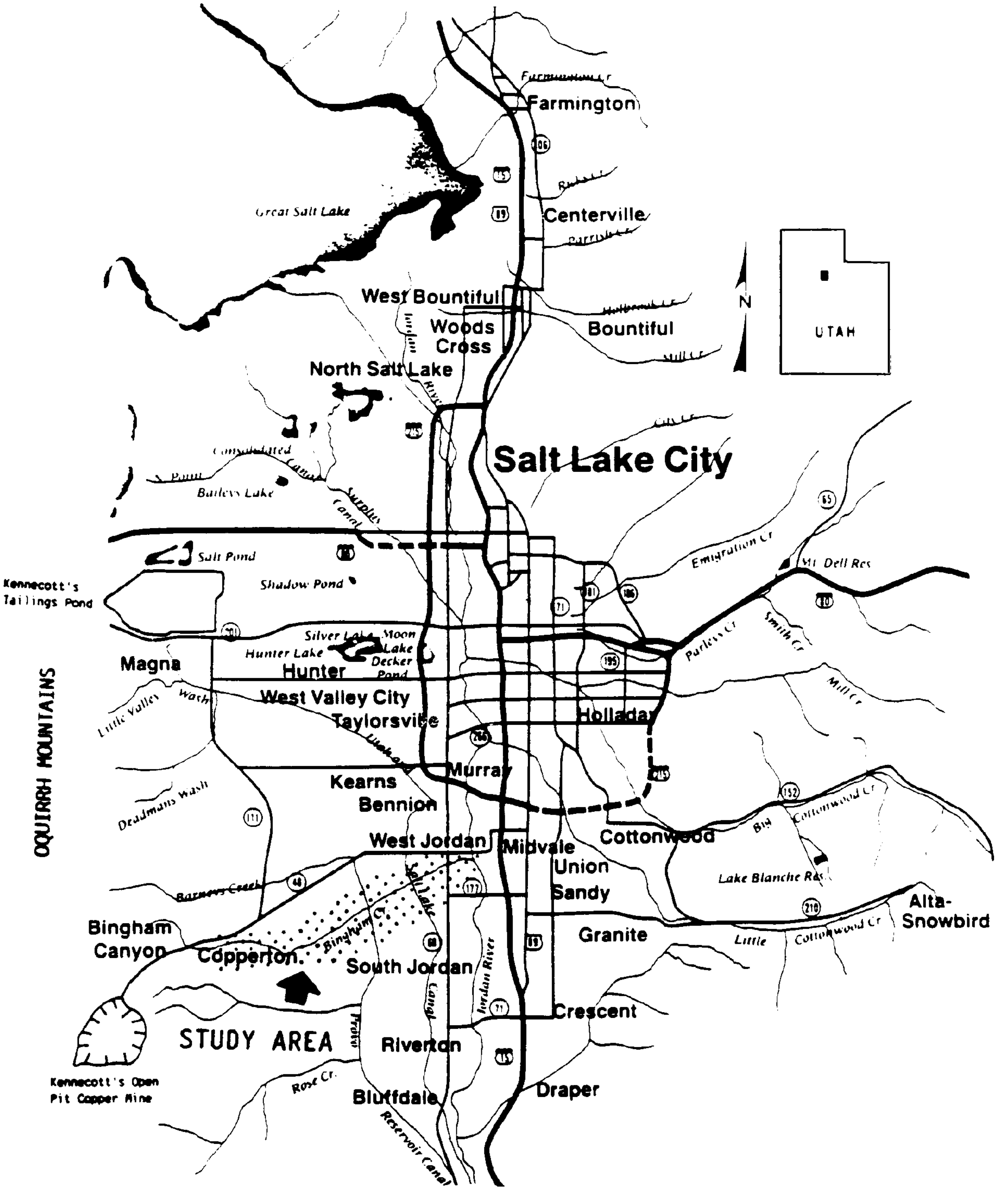

Figure 1-18. Location of Bingham Canyon Mine

(Source: State of Utah 1987)

) in the Oquirrh Mountains in northern central Utah. Land use in the immediate vicinity of the mine is predominantly rural. The town of Magna (population 8,600) is located 15 miles north of the site. Salt Lake City, a major metropolitan area, is located 20 miles northeast of the site (U.S. EPA 1989e).

This is a fully integrated facility comprised of extraction, beneficiation, and processing operations. The extraction operations consist of an open-pit copper mine, dump leach systems, and a precipitation plant. The beneficiation facility includes an in-pit crusher and two concentrators. The metal byproducts of copper mining at Bingham Canyon are molybdenum, gold, silver, uranium, selenium, platinum, and palladium (U.S. DOI, Bureau of Mines 1965a; State of Utah, undated; U.S. EPA 1989e; Kennecott 1992).

The Bingham Canyon ore body is monzonite porphyry intruded into sedimentary rocks, which have been severely altered. The predominant mineralization is disseminated sulfide ore of chalcopyrite and pyrite. The deposit is zoned with the most intense mineralization near the two main intrusive stocks (U.S. DOI, Bureau of Mines 1965a).

Sulfide mineralization is associated with the Bingham stock, a complex granite and granite porphyry body that intrudes quartzite and metasilicate country rock. The copper ore body is exposed over a lateral area of approximately 0.66 of a square mile and through a vertical depth of approximately 1,500 feet. In the primary, intrusive ore zone, chalcopyrite is the principal copper mineral, although bornite is common in ore from the central portion of the ore body. The central portion of the ore body is also characterized by a relatively low pyrite content and a relatively high molybdenite content. Surrounding the intrusive granite and granite porphyry is a halo that is several hundred feet wide and is composed of ore-grade sulfide mineralization in fractured quartzitic country rock. This zone was secondarily enriched with chalcopyrite, which is the principal copper mineral in the quartzite ores. This halo also is characterized by a very high pyrite content, which is several times higher than the average pyrite content of the rest of the ore body (U.S. EPA 1986, 1989e).

Extraction

The open-pit mine is about 2.5 miles in diameter and 0.5 mile deep. Since the Bingham Canyon Mine began operation in 1906, over 5 billion tons of material have been excavated. The mine produces about 300,000 short tons (st) of copper, 300,000 ounces (oz) of gold, 2.3 million oz of silver, and 12 million pounds (lbs) of molybdenite annually. In 1988, the mine produced approximately 107,000 short tons per day (stpd) of ore. The mine utilizes conveyor belt systems in addition to a standard rail system for ore haulage (U.S. EPA 1986; Salt Lake Tribune 1988).

About 3 billion tons of mineral waste rock have accumulated in waste dumps at the Bingham Canyon Mine since open-pit operations began in 1906. All mine waste is hauled from the pit to waste dumps via truck. Mine wastes are segregated according to their metal content in the rock and sent to different dumps. The material is dumped over the edge of a hillside to form sideslope dumps. Most of the dumps are contained within steep, narrow canyons. In some drainages/canyons, the dumps are up to 1,000 feet thick and will increase in thickness as mining continues and the dumps are extended beyond the confines of the drainages (Kennecott 1992).

Beneficiation

Approximately 125,000 tons of material between 0.0 percent copper and 0.325 percent copper are extracted at the mine daily. Approximately one-third to one-half of this material is between 0.15 percent copper and 0.325 percent copper and is amenable to leaching. This material is placed on terraced dumps and leached to recover copper. This tonnage will tend to decrease with time (Kennecott 1992).

Dump leaching and cementation operations at the Bingham Canyon Mine were initiated in 1923. The leach dumps (east and west) currently occupy approximately 2,110 acres (3.3 square miles) and contain an estimated 1.5 billion tons of material. The active leach dumps (east and west) occupy approximately 800 acres (1.3 square miles) and contain an estimated 700 million tons of material. Both the east and west side dumps are being leached (Kennecott 1992).

Approximately one-third of the total area of the east and west side dumps is leached at one time. A typical leach cycle involves 30 to 60 days of leaching and 60 to 120 days of resting (Kennecott 1992). To minimize the buildup of iron precipitates on the surface of the dumps, the top 4 to 5 feet of material is ripped by a bulldozer after each rest cycle. After about two cycles, the top layer is scraped off and pushed over the edge of the dump (U.S. EPA 1989e).

The low-grade ore is leached with a dilute solution of *in situ*-generated sulfuric acid (sulfuric acid is not added for leaching). The lixiviant has been applied by using either infiltration ponds, trickle leach, or rainbird sprinkler methods. The pregnant leach solution (PLS) is collected at the base of the dumps in clay-lined ponds. The PLS, which has fairly high concentrations of dissolved copper, is transported to the precipitation plant via concrete ditches and pipeline. Any excess PLS flows to a double-lined pond with a leak detection system, where it is held for treatment to remove the copper. After the copper has been recovered from the PLS, the barren solution from the cones flows to a sump in the central pump station; from there, it is pumped back to the top of the terraced leach dump piles and recirculated. The pH of this solution ranges from 2.5 to 3.0 (U.S. EPA 1989e; Kennecott 1992).

Each of the PLS ponds have unlined overflow ponds to collect any overflow from the PLS ponds due to a rainfall event or equipment malfunction. The PLS ponds were created by constructing concrete cutoff walls across natural drainages; the walls are keyed into bedrock to prevent subsurface losses. From the ponds, the PLS is conveyed via a main collection canal, which is constructed of epoxy-lined concrete, to the precipitate plant (U.S. EPA 1989e).

Kennecott's east collection system is "state-of-the-art." In addition to the main collection canal, a second, emergency overflow canal (constructed of epoxy-lined concrete) collects excess storm water runoff and conveys it to a 500-million-gallon overflow reservoir. This large reservoir is partially lined with clay (i.e., the face of the dam and the bottom of the pond extending away from the dam for several feet are lined). The reservoir is being upgraded to include a plastic liner (Kennecott 1992). This excess storm water is used within the concentrating process. Site personnel have stated that this collection system does not contribute to existing ground water-contamination problems at the site (U.S. EPA 1989e).

Kennecott has two concentrator plants, the Copperton Concentrator and the North Concentrator, with a combined design throughput of 142,000 tons per day (tpd). The Copperton plant, commissioned in 1988 and expanded in 1991, utilizes four conventional semi-autogenous (SAG) mill/ball mill circuits (102,000 to 150,000 tpd) for size reduction with the slurried product feeding a rougher/scavenger froth flotation circuit. Here, copper-, gold-, silver-, and molybdenum-bearing minerals are concentrated. This concentrate is then subjected to subsequent cleaning steps to remove gangue. In addition, the concentrate is further treated in the molybdenite froth flotation circuit where copper, gold, and silver minerals are chemically depressed, recovering a molybdenite concentrate as froth. The copper/gold/silver "tailing" from this step is then thickened and pumped via a 6-inch slurry pipeline to a filter plant, which is adjacent to the smelter some 18 miles away. The molybdenite concentrate is subjected to four further flotation cleaner steps, dried, and then packaged onsite for sale.

The North concentrator utilizes conventional primary, secondary, and tertiary crushing, then a 4-line rod mill/ ball mill circuit (30,000 tpd). This portion of the plant, known as the Bonneville plant, was constructed in 1967. Slurry from the mill is gravity fed 1.8 miles to a froth flotation circuit, constructed in 1984 at the Magna plant. This circuit, although smaller, is identical to the flotation circuit at Copperton. No molybdenite recovery circuit is in use at Magna, but plans call for installation in 1992 (Kennecott 1992). Table 1-8 details the major beneficiation equipment at each plant. Table 1-9 shows reagent type and usage in the flotation process. Reagent type and consumption are the same at both plants.

Table 1-8. Beneficiation Equipment by Plant

Equipment	Copperton	North
Primary Crusher	60/109 Gyratory at mine	54/74 Gyratory
Secondary Crusher	--	2 7-foot standard cones
Tertiary Crusher	--	3 7-foot shortheads
SAG Mills/Rod Mills	3 34-foot 12,000 HP (SAG) 1 36-foot 16,000 HP (SAG)	4 12- x17-foot 1,250 HP (ROD)
Ball Mills	6 18- x 28-foot 5,500 HP 2 20- x 30-foot 7,500 HP	4 12- x17-foot 1,250 HP 8 12- x 15-foot 1,250 HP
Mechanical Flotation Cells	54 3,000 (ft^3) 16 1,000 ft^3 14 500 ft^3	20 1,500 ft^3 14 500 ft^3
Column Flotation Cells	22	--
Thickeners	3 400-foot' tailings 1 200-foot conc. 2 180-foot conc. 1 200-foot clarifier	2 75-foot conc.

(Source: Kennecott 1992)

Table 1-9. Reagent Consumption in Flotation Process

Reagent	Lbs/Ton-Ore Copper Circuit	Lbs/Ton-Conc. Molybdenite Circuit
Lime	1.2	2.0
Collector	0.024	--
Sodium Cyanide	0.003	--
Frother (Alcohol)	0.048	--
Fuel Oil	0.026	0.28
Sodium Hydrosulfide	--	6.8
Sodium Silicate	--	0.65
Flocculent	0.02	--

(Source: Kennecott 1992)

The Arthur and Magna concentrators recover 90 percent of the ore values in the concentrates. Very fine particles interfere in the flotation process and reduce recovery. By installing two new tailings pretreatment plants (one at each concentrator), Kennecott has been able to recover some lost copper-molybdenum concentrates. The new plants receive mill tailings from the concentrators by pipeline. Four clusters of nine cyclones are used to separate the fine slimes (material less than 200 mesh in size) from the coarser tailings. The cyclone overflow (slimes) is routed to the tailings pond (see the discussion below), while the underflow is diluted with water to 45 percent solids and recycled to rougher flotation units. Rougher concentrate is then pumped to a regrind circuit consisting of closed-circuit ball mills with cyclone hydraulic separators. The product from this process is subjected to three stages of cleaner flotation, which recovers the copper concentrates (Engineering and Mining Journal 1971a).

Tailings Disposal

Tailings from the North Concentrator are gravity fed to a single point discharge into the 5,300-acre tailings pond. Tailings from the Copperton Concentrator are gravity fed through a 13-mile long pipeline to a peripheral discharge system and a point discharge on the same tailings impoundment. Approximately 70 percent of the Copperton tailings are distributed through the peripheral system with the balance through the point discharge. Additional inflows to the tailings impoundment are tailings from the Smelter slag concentrator, effluent from the wastewater treatment plant, and ash sluice from the power plant. Water is recovered for recycle from the pond using siphons. Excessive storm water can be discharged at two permitted discharge points. The water flows through the C-7 ditch to the Great Salt Lake under the conditions of a Utah Pollutant Discharge Elimination System (UPDES) permit (Kennecott 1992).

As in normal practice, the tailings impoundment perimeter is constructed of tailings. An upstream dike construction method is used. The overall slope of the dike rises at 5 to 1. The pond level increases at 6.5 feet per year (Kennecott 1992).

Dust is controlled through operating practice, road dust control, and revegetation. The perimeter tailings discharge system is used to keep the interior surface wetted. Magnesium chloride is used for dust control on roads. Surfaces exposed due to dike raising activities are prepared and seeded with a variety of grasses and seedling trees (Kennecott 1992).

Table 1-10

Table 1-10. Summary Data Sheet for the Kennecott Copper Company - Bingham Canyon Mine Tailings Pond Analysis

Substance (Total)	Concentration By Sites/Date					
	Arthur Tailings 3/13-3/19/78 (in µg/l)	**Magna Tailings 3/20-2/26/78 (in µg/l)**	**Tailings Pond Recycle 3/20-3/26/78 (in µg/l)**	**Treatment Plant Influent 3/20-3/26/78 (in µg/l)**	**Treatment Plant Effluent 3/20-3/26/78 (in µg/l)**	**Water-quality Standards (in µg/l)**
Antimony	<500	<500	100	<100	<100	100-50
Arsenic	5,000	5,000	100	400	30	50
Beryllium	150	80	<5	<5	<5	1
Cadmium	<25	<25	<5	<5	<5	5
Chromium	12,000	6,900	20	80	30	100
Copper	100,000	95,000	260	8,000	60	1,300
Lead	3,000	2,500	<20	400	<20	________
Mercury	2.0	1.0	0.8	________	0.7	2
Nickel	9,900	5,700	20	<20	70	100
Selenium	100	200	10	50	10	50
Silver	<100	<100	<20	<20	<20	50
Thallium	<200	<200	<100	<100	<100	1-2
Zinc	5,800	3,600	<20	500	30	5,000
Cyanide	1.24	0.10	0.60	<0.02	0.08	.2
Phenol	0.75	0.325	0.200	0.006	0.025	---

(Source: U.S. EPA 1986)

presents the results of chemical constituent analyses of tailings generated at the Bingham Canyon site (U.S. EPA 1986). Tailings composition varies as a function of ore composition (Kennecott 1992). These analyses were performed on the influent tailings stream containing 30 percent solids (Kennecott 1992). The results of the treatment plant and recycled tailings water sampling show significant improvement (U.S. EPA 1986).

Dried tailings can be a potential source of wind-blown dust. To address this problem, Kennecott has developed a soil-stabilization program for the tailings pond. In addition, Kennecott has tested many chemical additives to reduce dust blow. A chemical emulsion called Coherex, manufactured by Witco Chemical Corporation, has been effectively used at Kennecott for dust control. Coherex contains petroleum oils and resins mixed with water; it is sprayed onto the dry periphery of the tailings pond by trucks or sprinkler systems. Coherex imparts a permanent cohesiveness to the tailings by coating dust particles and forming an adhesive membrane over adjacent particles (Engineering and Mining Journal 1971b; U.S. EPA 1986).

In 1988, Kennecott upgraded the peripheral tailings distribution system so that up to 30,000 gallons per minute (gpm) of tailings are continuously distributed around the inside of the impoundment's dike. This has resulted in the ability to keep at least 95 percent of the tailings impoundment surface wet. The system is part of the Utah PM_{10} State Implementation Plan (Kennecott 1992).

Kennecott has a program to revegetate the faces of the tailings pond dam. A variety of grasses and trees (mainly Russian Olive) have been planted with some success. Kennecott has been cooperating with U.S. DOI, Bureau of Mines, United States Soil Conservation Service, and the Utah State University in developing, maintaining, and analyzing vegetation test plots on the tailings. Some areas of the tailings pond dam are supporting a thicker vegetative cover than the surrounding native soil. However, some slopes, notably the south-facing slopes, are more difficult to revegetate due to the intensity of the sun (State of Utah, undated).

Leach Circuit

For many years, Kennecott has operated a precipitation plant which employs a cementation process to extract copper from aqueous solutions. The precipitate plant contains 12 cones and operates on a continuous basis. Annual precipitate production in 1985 was 6,000 tons (Kennecott 1992).

At Kennecott's precipitation plant, the pregnant solution flows through a cone precipitator filled with iron shavings that works similar to a cyclone (U.S. Congress, Office of Technology Assessment 1988).

The precipitation system consists of a cylindrical wooden tank (measuring 6 to 7 meters high and 4 to 6 meters in diameter) in which a 4-meter-long stainless-steel cone is fixed (apex down). The upper third of the cone is constructed of stainless steel screen. The PLS is swirled into the cone via openings in two rings attached to the inside of the cone (the openings are one-third and one-half the way up the cone's side). The PLS swirls upwards through the shredded iron scrap, causing the copper to precipitate (Biwas and Davenport 1976; U.S. EPA 1989e).

The swirling action washes the copper from the iron surfaces, and the particles become suspended in the solution. The copper particles are carried upwards to near the top of the cone, where, as the velocity decreases due to increased cone width, they sink through the screened section into a collection area at the bottom of the tank. Kennecott's cone system is a high-capacity unit, which can handle up to 10 cubic meters (m^3) of solution per minute. Furthermore, the system is flexible, and two or more cones can be placed in series to maximize copper recoveries and/or handle solutions with high copper concentrations. Finally, an additional advantage of this system is that it has a low iron consumption rate (Biwas and Davenport 1976).

Process Wastewater Management

Makeup water for the concentrators is obtained from deep wells, Utah Lake (through a canal), surface drainage, springs, Bingham pit mine water, and mine dump runoff. The water supply is supplemented from two wells, located in the valley about three miles east of the mine. The depth to ground water at these wells is 300 to 400 feet (State of Utah, undated; U.S. EPA 1986; Kennecott 1992).

At the Copperton concentrator, makeup water is commingled with recycled tailings water in the ore feed slurry. Lime is added to a pH of 12. After beneficiation, the tailings slurry (approximately 51,000 gpm) is piped to the tailings pond. Additional wastewater flows to the tailings pond include approximately 15,000 gpm of process water from the Magna concentrator, 1,500 gpm of process water from the slag tailings concentrator, 2,500 gpm from the smelter and refinery wastewater treatment plant, and 1,000 gpm of ash sluice wastewater. Before it is discharged to the tailings pond, the process wastewater from the smelter and refinery is treated with lime and ferric chloride (State of Utah, undated; Kennecott 1992).

Runoff

The mine pit and leach and waste dumps border two surface-water drainages (Bingham Canyon and Butterfield Canyon). Surface runoff from these drainages flows east toward the Jordan River, which feeds the Great Salt Lake (Kennecott 1992).

Of particular note are ground water and surface-water impacts associated with contamination in Bingham Creek. Bingham Creek is a small ephemeral stream which flows east and south of the town of Copperton. Bingham Creek originates in the Oquirrh Mountains in Bingham Canyon and has historically flowed towards the northeast into the Jordan River. Bingham Creek and its floodplain have drained a large, mineralized area for millions of years. The creek was also used for many years by non-Kennecott mining companies for mine waste disposal. Some of the first mining activity in the Salt Lake area included placer mining of Bingham Creek gravels followed by underground mining of lead/silver ores throughout Bingham Canyon. Kennecott has mined only copper ores; their involvement dates from 1936. Various mining companies used the creek to some extent for mine drainage, tailings disposal, leachate collection, and run-off control. Bingham Creek channel currently does not carry any flowing water (State of Utah 1987; Salt Lake Tribune 1988; Kennecott 1992).

During 1983 and other particularly wet years, Kennecott diverted excess snow melt runoff into a series of evaporation ponds located a few miles south of Bingham Creek. Some runoff water may have seeped into the ground and into the aquifer in the South Jordan area (State of Utah undated; Kennecott 1992).

To eliminate the need for using the lower portion of Bingham Creek for excess storm water and snowmelt runoff, Kennecott has installed a series of dams in the drainages above the mine area to capture runoff water before it crosses the mine and disturbed land areas. This clean water is then piped to the copper company's Copperton concentrator for makeup water supply. An 11-mile canal now extends completely around the mine drainage area. This canal was constructed to collect potentially contaminated runoff and any leach liquor that escaped the leaching system and divert it to a Kennecott reservoir. This reservoir is currently being lined (State of Utah, undated; Kennecott 1992).

2. ASARCO Inc.; Ray Complex; Pinal County, Arizona

The ASARCO Ray Complex is located about 80 miles southeast of Phoenix in the Mineral Creek mining district. The mine lies in the Mineral Creek Valley approximately 5 miles north of the Gila River.

From 1911 to 1955, the mine was an underground block-caving operation. In 1955, the current open-pit mining operation started. Forty-three million tons of overburden were excavated during the initial development of the pit (U.S. DOI, Bureau of Mines 1965a). The site consists of the open-pit mine, heap and dump leaches, a 30,000-tpd concentrator and a 40,000-tpy SX/EW plant. Sulfide ore is also transported to a 30,000-tpd concentrator at Hayden. An active 625,000-tpy smelter and a 1,600-tpd acid plant are also located at Hayden.

The Ray Mine and associated dump leaching operations are constructed on both sides of Mineral Creek Valley, in a surface-water-flow channel restricted by bedrock. Mineral Creek is diverted around the mine through a 3.25-mile, concrete lined tunnel. The Ray mine is underlain by bedrock (primarily Precambrian pinal schist, Apache group sediment, diabase, and porphyry formations). The pits and surrounding bedrock are relatively dry (from a hydrogeologic perspective); no alluvial aquifers exist. Bedrock yields small, but sustained, ground water flow. Water is present at depth in isolated fracture zones, but none of the bedrock formations are capable of supplying significant or sustained yield (U.S. EPA 1989e).

The ore body is quartz porphyry diabase intrusive and approximately 40 percent Precambrian schist. The schist and porphyry ores are mineralized principally with supergene chalcocite, although the predominant mineral in the diabase is chalcopyrite. Copper oxide silicates of chrysocolla, cuprite, malachite, tenorite, and native copper comprise about 20 percent of the ore (U.S. DOI, Bureau of Mines 1965a). The mineralogy at the Ray mine site is quite complex and contains zones of sulfide, oxide, and native copper. The ore body has undergone extensive faulting, oxidation, and erosion, producing the two major forms of supergene copper mineralization. Early mining focused on the higher grade chalcocite, while the lower-grade chalcopyrite is being mined today. The oxide copper zones are characterized by high-grade chrysocolla (Engineering and Mining Journal 1988; U.S. EPA 1989e).

Extraction

Mining

Mining is being conducted in the Sonora, Sharkey, Calumet, and Pearl Handle mining areas. There are six active, low-grade copper sulfide ore dumps (Nos. 1 through 5 and No. 7 waste dumps) and one active copper silicate ore heap leach (see description of heap leaching below). Figure 1-19

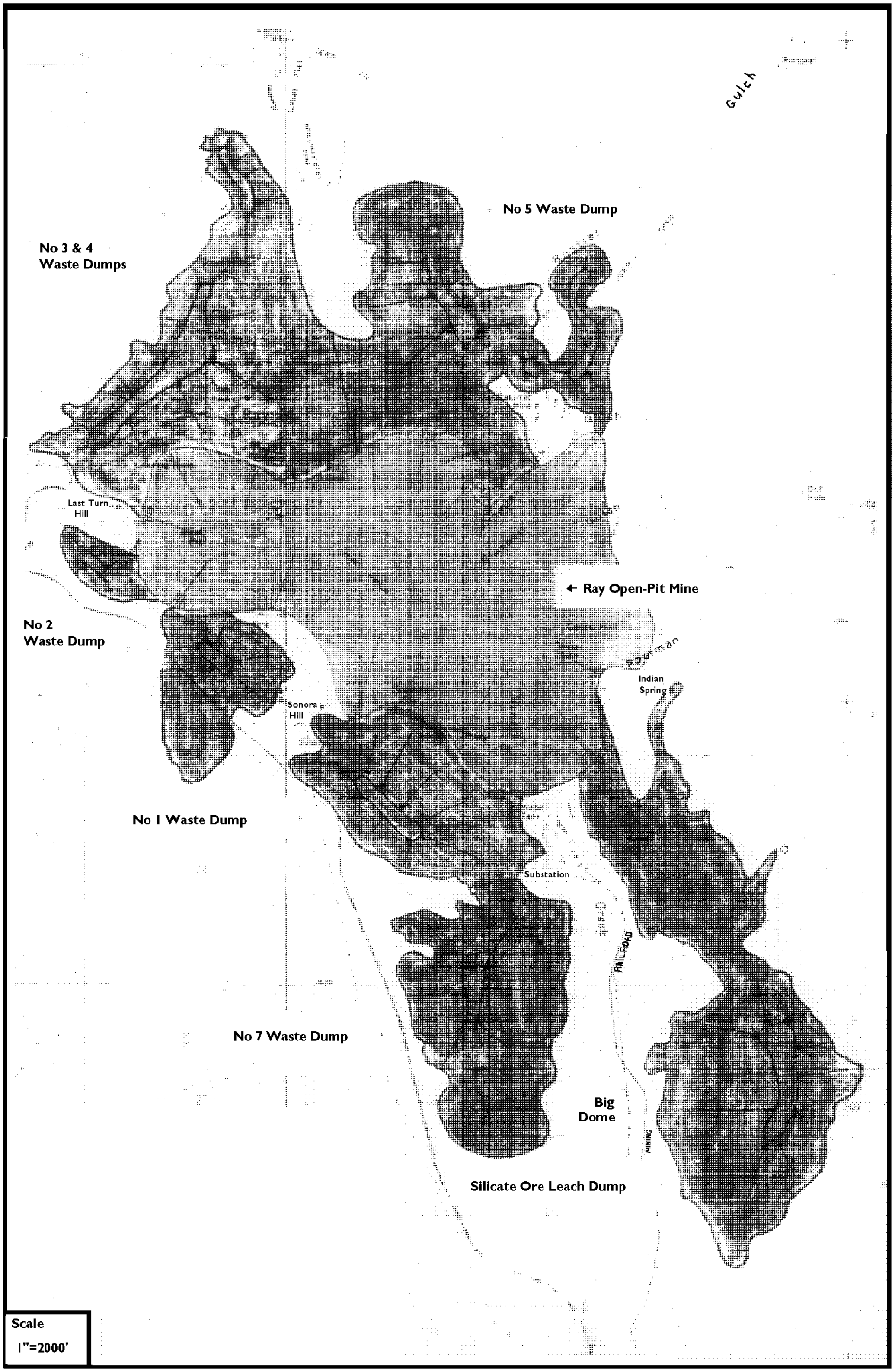

Figure 1-19. Existing Ray Mine Site Disturbed Areas

(Source: ASARCO 1990)

shows the locations of the Ray open pit and each of the dumps (U.S. EPA 1989e; ASARCO 1991).

Extraction operations are conducted at the Ray pit using electric shovels supplemented by front-end loaders. The total materials production rate is 220,000 tpd. Each day, 60,000 tons of sulfide ore undergo primary crushing to minus 8 inches at the mine site and 30,000 tpd shipped by rail to the Hayden concentrator. About 13,000 tpd silicate ore are crushed and sent to heap leaches. The rest of the material generated is low-grade ore (which is hauled to dump leaches) and waste rock (which is sent to waste dumps). The stripping ratio is 2.5 to 1 (waste to ore).

Mine planning is predicated on the sulfide operation. As a result, silicate ore is stockpiled when in excess and is fed from the stockpile to the primary crushers to maintain heap placement.

Mine overburden is separated into barren and copper-bearing portions. Only barren material is placed on those dump sites on the northeast side of the mine to prevent pollution of Mineral Creek from dump drainage (U.S. EPA 1989e).

Dump Leaching

Onsite leaching of copper has become an integral part of the Ray mine operation. Twenty-five percent of the mine's production originates from leaching operations. For many years, the sulfide dumps were leached to recover cement copper, which was precipitated in cones or launders loaded with shredded iron. The cement copper was then shipped to the Ray smelter for further refining. With the advent of SX/EW technology, operation of the cementation precipitation unit operation was suspended (Engineering and Mining Journal 1988).

The sulfide leach dumps surround the southwest side of the mine workings (see Figure 1-19). Approximately 1,100 acres are available for the low-grade copper sulfide ore leaching processes (only 10 to 15 percent of which is being flushed with water at any one time). The remaining area is at rest under oxidizing conditions. The dump leach piles are located directly on the existing topography.
According to ASARCO, there was no special surface preparation prior to the initial deposition of dump material since the bedrock is nearly an impervious conglomerate.

Mine-run ore is hauled to the leach piles by truck and spread with bulldozers. After each lift is completed, the surface of the pile is ripped to a depth of approximately 5 feet. The leach solution is usually applied to a pile in 30- to 60-day on-off cycles. The solution that is applied to the dumps has a pH of approximately 2.0. It is delivered to the dumps at 8,700 gpm and applied through a series of impulse sprinklers. The pregnant liquor is collected behind concrete storage dams, keyed to bedrock but unlined, and is reapplied to the dumps until the copper concentration is sufficient for SX/EW recovery. It was originally pumped to either the north or south precipitation plants. The pregnant solution influent to the precipitation plants generally contained approximately 0.42 g/l of acid and had a pH of 2.8. The tail water from the precipitation plants was redistributed onto the leach surfaces. Since 1987, pregnant solution from dump leaches is directed to the SX/EW plant at the site (U.S. EPA 1989e; Beard 1990).

Stope Leaching

Another source of precipitate copper at the Ray mine was from historic *in situ* stope leaching of the caved areas in the underground mine during Kennecott's ownership of the mine. Stopes, once used to access and mine the copper ore, are now used to improve leaching recovery. Acidic mine water was recovered from block-caved sections of closed underground mining operations. The ore chutes or draws were originally sealed off as soon as the oxidized capping or the broken protore began to appear in the ore drawn from the chutes. As a result, a large quantity of low-grade, oxidized copper-bearing broken rock remained in the mine. This ore was subjected to the slow natural leaching process by downward percolating meteoric waters that produced copper-rich leachate or Acid Mine Drainage (AMD). As of 1982, copper leachate or AMD was pumped out of the mine and sent to the cementation unit. The cementation precipitates, containing about 80 percent copper, were then being dried and shipped to the Chino Branch smelter at Hurley, New Mexico (Intermountain Pay Dirt 1982).

The quantity of copper produced from *in situ* leaching cementation operations increased from 3,600 tons in 1954 to 17,700 tons in 1957. Additional pumps were installed in 1959 to expand stope leaching capacity into additional caved areas (U.S. DOI, Bureau of Mines 1965a). Stope leaching operations, however, are no longer in operation at the Ray Complex.

Beneficiation

Milling

At the concentrator, secondary crushing further reduces the sulfide ore to minus 0.5 inch. Then, the material is conveyed to fine ore bins, which feed 130 to 280 tons per hour (tph) of ore to each of seven grinding sections. The ore is combined with water to form a slurry that is concentrated in a sulfide flotation circuit. The reagents used at the concentrator in the primary flotation circuit are Raconite (sodium butyl xanthate) for the collector, and MIBS (shell 10) for the frother. The average pH in the flotation circuit is 11.5 (Crozier 1979). Previously stockpiled copper smelter slag is currently being reclaimed and fed to the mill in small quantities (Beard 1990).

In 1989, a $12-million expansion project was completed to maintain production capacity as the pit deepened and the hardness of the ore increased. In addition, a $254-million expansion was completed at the Ray mine site in early 1992. A 60,000-tpd relocatable in-pit crusher and conveyor system was installed, and a 30,000-tpd concentrator was built at the mine site (ASARCO 1992).

Silicate Ore Vat Leaching Operations

Because oxide copper in the silicate ore could not be recovered by conventional sulfide recovery methods, the Silicate Ore Leaching Process Plant (SOLP) plant was built in 1965 to process the oxide ore. The plant processed 10,000 tpd of copper silicate ore. It was designed to produce 24,000 tpy of copper. The SOLP consisted of a vat leaching system containing a crushing facility; 14 vats (100 x 110 x 20 feet, each holding about 10,000 st of ore); a conveyor system to fill them with crushed ore, and a gantry crane with bucket to unload the leached material. The SOLP plant was designed to use sulfuric acid lixiviant. A 750-tpd acid plant was also built as part of the $35-million project. A 14-day cycle consisting of 10 days for leaching, 1 day for unloading, 2 days for washing, and 1 day for reloading was used in the SOLP. The PLS solution was sent from the SOLP to an electrowinning circuit. During leaching, aluminum salts, iron, and other contaminants detrimental to electrowinning were dissolved, along with the copper. These impurities significantly reduced cathode quality (Engineering and Mining Journal 1980; Intermountain Pay Dirt 1982; Intermountain Pay Dirt 1985).

In 1976, an agitation leaching facility was constructed to work in conjunction with the vat system. This facility increased the amount of ore that could be handled from 10,000 to 14,000 stpd. The facility produced 30-45 stpd of copper precipitate per year that required smelting and electrowinning (Engineering and Mining Journal 1980).

Also in 1976, the silicate leach plant was expanded by an additional 40 percent. No new vats were added; the increased capacity instead came from separate processing of the fine material and from modifications in other areas. The ore was processed by a three-stage crushing circuit and then was split into two sizes by 1/2-inch diameter screens. The fines were subjected to leaching and the coarse fraction was treated in the 14-day, vat-leaching cycle. The SOLP system was closed in 1982 and was subsequently replaced in 1985 by the heap leaching operation (Intermountain Pay Dirt 1982; Engineering and Mining Journal 1988).

Heap Leaching

A copper silicate ore heap leaching operation is currently used to recover copper from copper silicate mineralized ores. The silicate heap leach dump is located southeast of the mine workings (see Figure 1-19). The silicate ore grade averages about 1.10 percent total copper, of which 0.77 percent is readily soluble. Mine-run ore is delivered by haulage trucks to the primary crusher (which reduces the ore to minus 8-inch size). The crushed ore is then conveyed to a secondary/tertiary crushing facility (which reduces the ore to minus 3/4-inch size). This finely crushed product is then conveyed to a fine-ore building, which has a capacity of approximately 35,000 tons. The crushed ore is fed from fine-ore storage onto a series of conveyors which move the ore to a truck hopper area adjacent to the copper leaching area. On the conveyor, the ore is prewet with concentrated sulfuric acid at 93 percent and at a rate of 60 pounds per ton (lbs/t). Trucks then load and transport the ore to the heap site. Each lift on a pad is 8 feet and ore is added at a rate of 13,000 stpd to a volume of 40,000 tons. Prior to building each new 8-foot lift, the existing lift is ripped. After placement on the pads, the ore is sprinkled with sulfuric acid solution (recycled raffinate) containing from 18 to 19 g/l H_2SO_4, at a flow rate of 7,000 gpm using a Drisco pipe network and sprinklers. Each lift is leached for 33 days. The leachate is collected in a PLS collection basin and sent to the SX/EW plant. The PLS, which contain less than 2.0 g/l of free acid and 4.5 g/l of copper, are collected in unlined ponds and pumped to the SX/EW plant. After the leaching cycle is completed, new materials are added to the pads (ASARCO 1992).

Leach Circuit SX/EW Plant

In 1980, ASARCO started up a new copper SX plant at the SOLP. The SX plant was designed with a maximum capacity of 108 stpd. In 1980, SX plant production was almost 90 stpd. The SX plant has two parallel trains of mixer-settlers for extraction and stripping, along with other tanks and vessels associated the system. There are a total of 29 stainless steel, 3 carbon steel, and 14 reinforced concrete tanks. Stainless steel is also used in 80 percent of the piping and in all of the pumps, mixers, centrifuge, and tank liners (Engineering and Mining Journal 1980; Intermountain Pay Dirt 1985).

In 1987, ASARCO modified the SX plant to use the full 40,000-tpy capacity of the SX/EW system. A portion of the increased output, 7,500 tons, came from processing of PLS from sulfide dump leaching. The remainder of the additional output of the SX/EW plant came from increased processing of solutions from heap leaching of copper silicate ores. During the expansion, the existing SX trains were converted to accommodate the lower-grade dump leach solutions (Southwestern Pay Dirt 1987a).

Prior to the expansion, leach solutions from the sulfide dumps were processed in the precipitation plant at Ray, then sent offsite for smelting and refining. After the modifications were completed, ASARCO began sending PLS from the sulfide dumps to the SX/EW plant, and the precipitation plant was closed. Within the SX plant, sulfide ore leach solutions are processed in one train, while silicate ore leach solutions are processed separately in the other train. The 1987 modifications further made it possible to sweeten the low-grade pregnant solutions from the sulfide waste dumps. The lean electrolyte produced from the sulfide dumps is commingled with rich electrolyte from the silicate train, yielding a combined electrolyte for the tankhouse with a copper concentration of 48 to 50 g/l (Southwestern Pay Dirt 1987a; Engineering and Mining Journal 1988).

The silicate train operates at a solution flow rate of 4,000 gpm carrying 4.5 g/l of copper and includes two extraction and two stripping stages. The flow rate to the sulfide dump train is 8,000 gpm of 0.77 g/l copper, divided between two lines of vessels arranged to provide one extraction and one stripping stage. The organic extractant used is Acorga P-5397 carried in 90 percent solution of Phillips SX-7 (kerosene). The plant is equipped with eight settlers, each having a surface area of 56 by 54 feet. Each settler has three 5,000-gallon tanks with Mixco agitators (ASARCO 1992).

Tailings

The tailings pond for the Ray concentrator is located three miles to the southeast of the Ray mine. The Hayden concentrator tailings ponds are located 18 miles southeast of the Ray mine near the town of Hayden, Arizona. Tailings are piped from the concentrator to the tailing ponds. The site is in the San Pedro Valley at the confluence of the San Pedro and Gila Rivers. The valley is bounded to the northwest by the Dripping Springs Mountains and to the southwest by the Tortilla Mountains. Tailings Ponds AB-BC and D are sited on floodplain deposits along the Gila River. The alluvial soils are predominantly granular with appreciable amounts of silt and clay. The thickness of alluvium in the valley is about 200 to 500 feet. The elevation of Tailings Pond AB-BC is 1,950 to 2,000 feet. The ground surface beneath Tailings Pond AB-BC slopes at approximately one percent from north to south. Tailings Pond D is constructed on the side of a hill at a six percent slope to the southwest (ASARCO 1991).

Tailings Pond AB-BC has been in operation since the early 1900s. Tailings disposal operations commenced in 1910 at a rate of about 4,000 tpd. The rate of deposition was increased to about 16,000 tpd in 1952; 21,000 tpd in 1960; and gradually since then to the present 30,000 tpd (ASARCO 1991).

No information was found concerning the details of the initial dam construction of Tailings Pond AB-BC. From 1910 to the 1960s tailings were discharged into the pond from a single point near the eastern end of the pond. Decanted water may have been discharged directly into the Gila River. Since the early 1950s, the dam has been raised in 10-foot lifts by bulldozers that pushed dried-out material from the beach area of the ponds to the edge and spread it out to form a crest. In the 1960s, the pond surface was divided into three pond areas by dikes to start a cycle of one pond area being filled with tailings, one area being dried out, and the other area being formed by diking. The Tailings Pond is now divided into two sections AB and BC (each divided into the three areas described above) (ASARCO 1991).

Peripheral spigotting of tailings to Pond AB-BC was started in the mid-1960s. However, the crest of the dam was still being raised with bulldozers and draglines. Seepage has occurred, and is still occurring, along the contact point between newly spigotted materials and the previously deposited tailings, primarily along the western half of Pond AB-BC. The seepage appears to originate from relatively pervious sands tailings resting upon relatively impervious silty tailings. Tailings water at the surface of the ponds percolates through the peripheral coarser materials until it reaches the much finer-grained tailings; from there it spreads and travels horizontally towards the face and seeps out at the contact between the two material types (ASARCO 1991).

A slope failure occurred near the northwest end of the dam on December 2, 1972. It was reported to have produced a gap in the crest of the dam estimated to be about 500 feet across, 30 to 50 feet deep, and 200 feet into the reservoir. A smaller failure occurred on February 3, 1973, about 1,000 feet east of the previous slope failure. It left a gap estimated to be about 500 feet wide, 20 to 40 feet deep, and 150 feet into the reservoir. After this failure, water was found seeping out of the failed portions and active channeling was observed at several points near the contact of the sandy and silty tailings. A geotechnical investigation was conducted in the failure areas that resulted in new dike construction being set approximately 200 feet inside the original dam to flatten the overall slopes and increase stability (ASARCO 1991).

The starter dike for Tailings Pond D was constructed to an elevation of 2,020 feet between November 1981 and July 1982. The starter dike is approximately 8,700 feet long, up to 48 feet high, and has an upstream slope of 2 to 1 (horizontal to vertical) and a downstream slope of 2.5 to l. The starter dike was constructed from alluvial soils excavated from upstream of the inside toe of the dike. The eastern 3,000 feet of the dike was constructed of more coarser grained soils than the western portion (ASARCO 1991).

After 29 weeks of tailings disposal in Pond D, a settlement crack with an associated tailings seep was observed by mine employees. The crack shape and surface evidence indicated that the upstream point of the dike had settled relative to the downstream part. It was believed that the cracks were the result of differential settlement between the coarser and finer grained dike sections upon exposure to the wetting front induced by the tailings. Varying embankment heights might also have accounted for some of the differential settlement cracking.

Tailings pond water control at both Ponds AB-BC and Pond D is accomplished using siphons. The system consists of a 30-inch (diameter) pipe mounted on buoys that extend 1,000 feet from the berm into the center of the pond. Approximately 7 million gallons per day (MGD) are recycled (Weiss 1985).

Electrowinning Plant

The electrowinning plant was built by Holmes and Narver and contains one starter sheet section comprised of 36 cells. Each cell is equipped with 40 titanium blanks to make 11-lb starter cathodes used in the commercial section. The rated capacity of the starter sheet section is 20,000 lb/day. Electrolyte flow rate through the cells is 10 gpm (Engineering and Mining Journal 1988).

The commercial section consists of 400 cells divided into four electrical and flow circuits. These cells contain 41 lead anodes and 40 copper cathodes. Electrolyte flow through these cells is 50 gpm. Each cell produces 225 to 275-lb copper cathodes at a DC cell voltage of 1.9 and a current of 9,000 to 17,000 amps. The tankhouse is capable of producing 130 stpd of cathode copper (ASARCO 1992).

Mine-water Management

The Gila River receives all drainage in the area and flows southwest to the Ashurst-Hayden diversion dam near Florence, approximately 15 miles below Kelvin, where the river is totally diverted for use as agricultural irrigation water. Mineral Creek, which was the original drainage course through the Ray mine, meets the Gila River at the town of Kelvin. To prevent contamination of Mineral Creek and the Gila River, ASARCO has constructed a large flood control and diversion dam north of the mine site which diverts the flow of Mineral Creek into a 3.4-mile tunnel that conveys the flow of Mineral Creek around the mine site (Intermountain Pay Dirt 1982; U.S. EPA 1989e).

The entire Ray mine area is underlain with bedrock. All solution recovery dams are keyed into bedrock to ensure containment of pregnant solutions. Dams and associated pipelines which lie above gradient are designed to flow into containment areas during any upset condition. Dams lying downgradient of the headwater reservoirs are equipped with primary and backup pumping capability. In the event this capability is lost or is insufficient for incoming flows, each dam is designed to overflow into the plastic lined Big Dome Reservoir, a 14-million-gallon capacity pond (U.S. EPA 1989e).

The PLS from the leach dumps is retained by a dam constructed across the downgradient side of the drainage channel. Waters which might overflow the leach dams are collected in the Big Dome Reservoir. Process water spills and runoff from process areas would also be contained in this pond. This water is either pumped back to the leach dumps or treated at the lime neutralization/precipitation facility (U.S. EPA 1989e).

All natural surface- and ground-water drainage from the area drains to diversion ditches. These have been constructed around the sulfide ore leach dumps located west of the open pit workings to minimize the amount of surface water entering the process/water system (U.S. EPA 1989e).

Both pits are confined by bedrock and extend well below the elevation of Mineral Creek (U.S. EPA 1989e).

3. Cyprus Sierrita Corporation; Sierrita Mine; Pima County, Arizona

The Sierrita Mine and related operations occupy approximately 20,000 acres on the east slope of the Sierrita Mountains south of Tucson. The complex, as shown in Figures 1-20

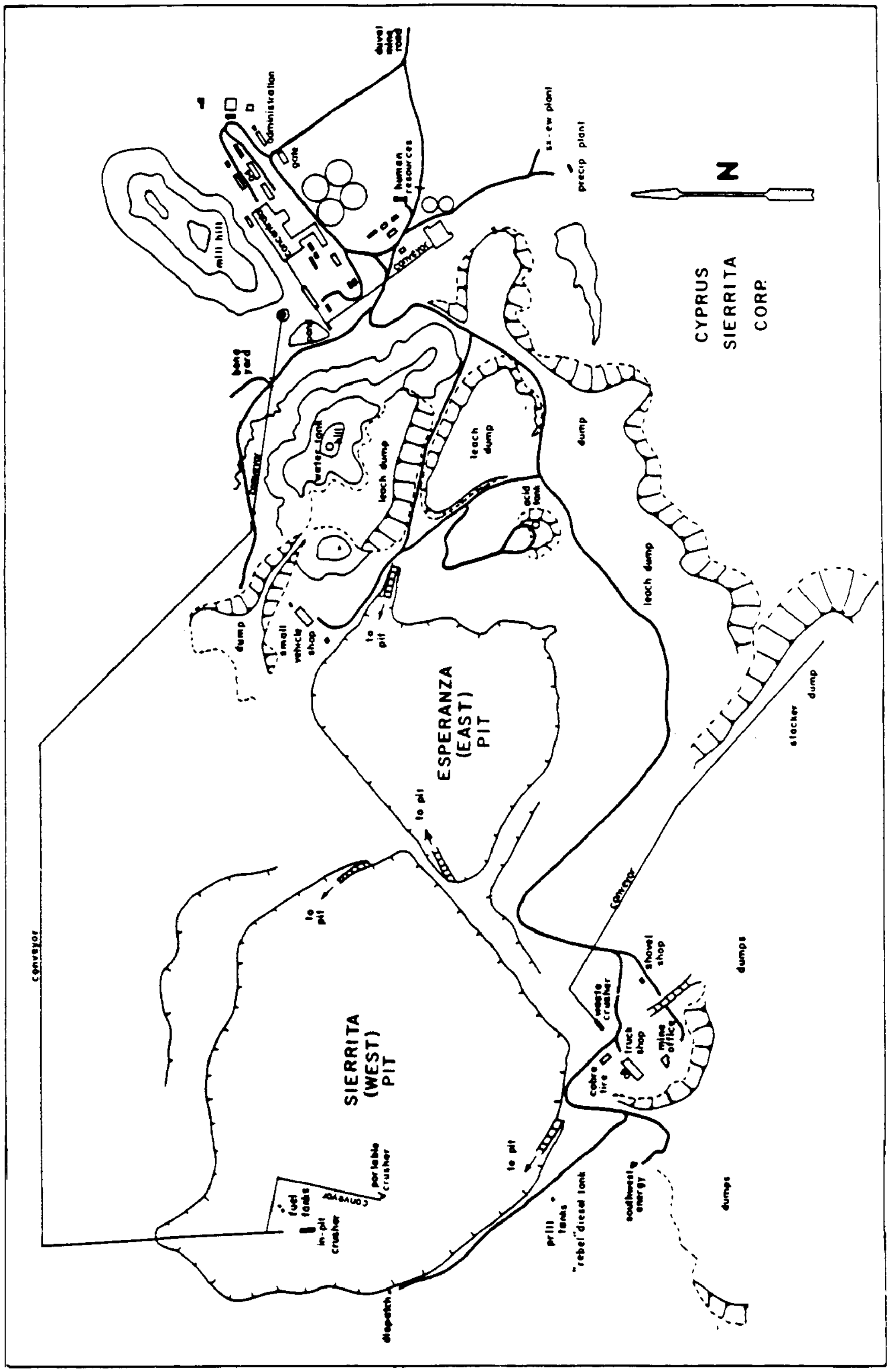

Figure 1-20. Cyprus Sierrita Process Location Map

(Source: Cyprus Sierrita Corporation 1989) and 1-21, is located in Pima County, Arizona, about 22 miles south of the intersection of Interstates 19 and

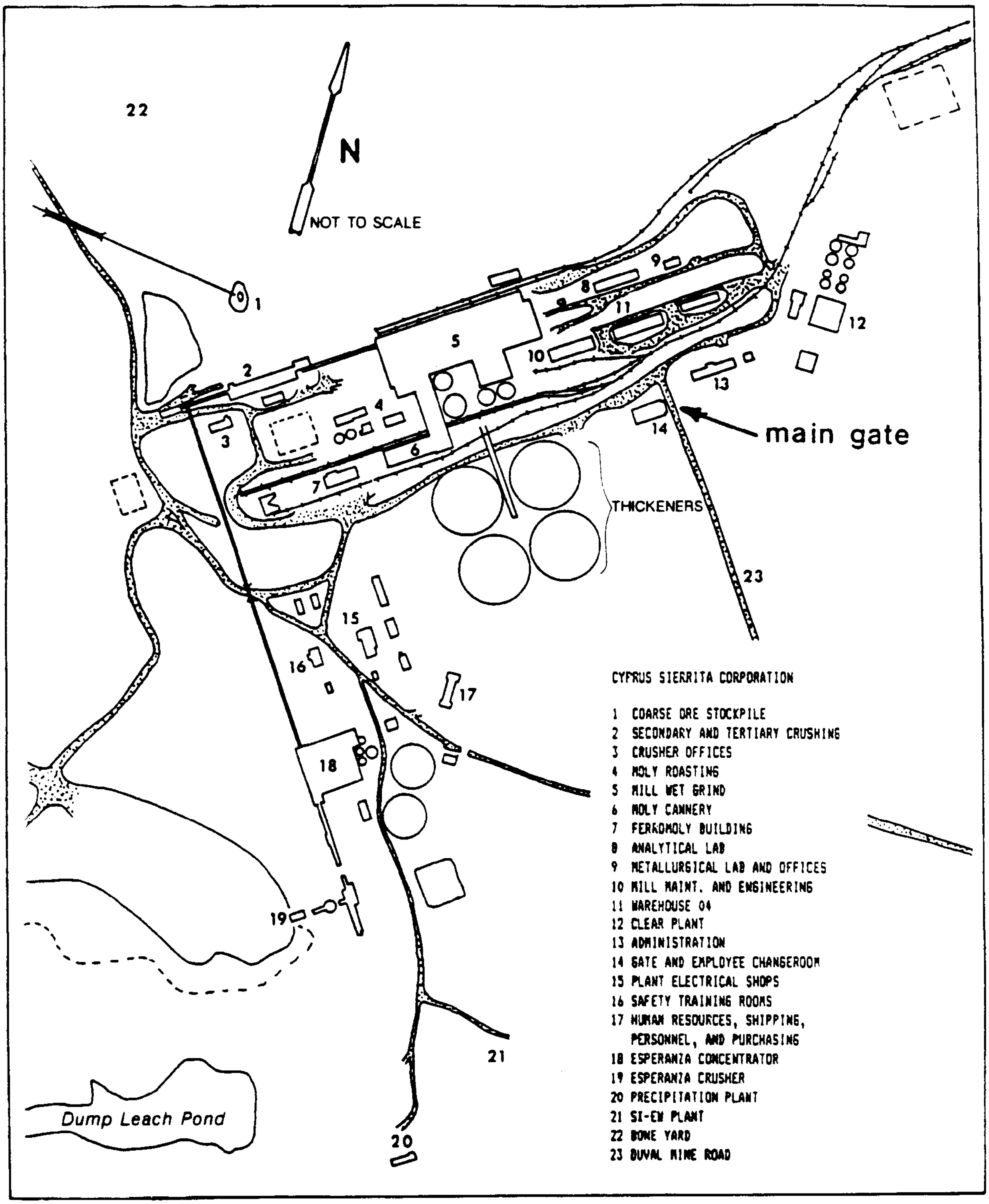

Figure 1-21. Cyprus Sierrita Facility Map

(Source: Cyprus Sierrita Corporation 1989)

10 and 7 miles west of I-19 near the town of Sahuarita, Arizona (28 to 32 miles south of Tucson). Cyprus Sierrita Corporation operates a sulfide ore concentrator, a heap leaching operation, a ferro-molybdenum plant, a rhenium plant, and an SX/EW plant in conjunction with mining operations at the Sierrita pit (U.S. DOI, Bureau of Mines 1965a; Reed and Associates 1985).

The ore body is a sulfide-enriched zone (primarily chalcopyrite) in a quartz monzonite porphyry intrusion (U.S. DOI, Bureau of Mines 1965a). Ore is mined from two adjacent open pits (Sierrita and the Esperanza pits). Ore crushing, grinding, flotation, and molybdenum roasting stages follow extraction operations. Beneficiation consists of a sulfide copper molybdenite concentrator with a 100,000-tpd capacity (U.S. EPA 1986). Copper concentrates, molybdic oxide, ferro-molybdenum, and rhenium (as ammonium perrhenate), and silver byproduct are produced at the site (U.S. EPA 1988b).

Three principal wastestreams are generated at the Sierrita mining operation. They are overflow from the dump leaching operation, tailings that result from operations at the Sierrita mill and concentrator, and septic system effluent (U.S. EPA 1989d).

Extraction

Mining

Overburden stripping operations at the Esperanza open-pit mine began in 1957, utilizing a combination of scrapers, conveyors, and dump wagons; ore extraction began in 1959. The Sierrita Mine began production in mid-1970. The two mining operations were combined in 1988 and are no longer referred to separately; they are identified together as the Sierrita Mine. The combined production rate is 60,000 to 70,000 tpd. The metals output per year is 200,000,000 lbs for copper, 20,000,000 lbs for molybdenum, and 1,000,000 oz for silver. (U.S. EPA 1988b).

Conventional drilling and blasting open-pit mining methods are used for extraction of the sulfide ores. Blast holes are drilled in groups in 50-foot-high benches by a fleet of eight rotary drills, then filled with an Ammonium Nitrate and Fuel Oil (ANFO) blasting mixture. Electric shovels and scrapers load the ore onto electric-drive dump trucks, which take it to primary crushers in the pit. The ore is transported to either the portable or stationary pit crushers and crushed to about 6 inches, then discharged onto a 3-mile-long conveyor belt system and delivered to the coarse ore stockpile. Waste rock and alluvium are transported by truck to the surface where they are crushed, then transported by conveyor to a dump (U.S. EPA 1988b).

In 1989, the stripping ratio at the Sierrita Mine was 0.77 to 1. Over 113 million tons of overburden have been excavated, and 540 million tons of waste rock and 61 million tons of dump/heap leach waste have been generated. Waste rock covers approximately 5.6 million square yards and dump/heap leach waste covers 1.3 million square yards. Waste rock and leach dump waste management units do not have liners, leachate detection systems, or ground water monitoring systems (U.S. DOI, Bureau of Mines 1965a; U.S. EPA 1986; Beard 1990; U.S. DOI, Bureau of Mines 1990a).

Approximately 3 million gallons of water per year are pumped from the Sierrita pit into a surface impoundment. This impoundment has no liner, leachate collection system, or ground water monitoring wells (Reed and Associates 1985).

Dump Leaching Operations

Cyprus Sierrita Corporation maintains a dump leaching operation for recovering copper from oxide ore. The leaching operation is comprised of the leach dumps, a series of ponds for containment of PLS and raffinate, and an SX/EW plant for copper extraction (U.S. EPA 1989d).

Dump leaching of low-grade ore began in 1964 at the Sierrita Mine site and in 1959 at the Esperanza pit (shortly after mining operations began). A sprinkler system is used to spray dilute sulfuric acid solution onto the leach dumps (no other chemicals have been used in the leach dumps). As the dilute acid solution leaches through the dump, bacteria flourish in the low-pH environment (the pH is monitored and kept above 2.3) and contribute to the release of copper. The PLS percolates through the oxide ore. According to Cyprus, the bedrock serves as a collection surface for the resultant PLS. Bedrock topography beneath the leach dumps appears to allow the PLS to discharge to the surface. Most of the PLS flows into the Amargosa wash drainage, although smaller amounts discharge to the Esperanza wash drainage. A series of ponds located along Amargosa wash and Esperanza wash (near the base of the leach dumps) have been constructed for containment of PLSs (see the discussion below) (U.S. EPA 1989d).

PLS discharged to the Amargosa wash drainage is contained by the No. 1 dam. The copper leach pregnant pond at the No. 1 dam is approximately 1 acre in size and 6 feet deep. It has the capacity to store 6 acre-feet of PLS per year. This impoundment is located on bedrock. There is no leachate collection or ground water monitoring system around this pond, although the unit has a seepage collection pond which acts as a runon/runoff control system (U.S. DOI, Bureau of Mines 1990a). Collected PLS is pumped by pipeline to the SX/EW plant, where copper is extracted.

After organic extraction, barren solution is discharged from the SX/EW plant into the raffinate pond (a clay-lined pond underlain by bedrock). The solution is then pumped through a 20-inch line from the raffinate pond to a booster station and an intermediate tank located about halfway up the dump. To maintain the pH, additional sulfuric acid is added to the barren solution from two nearby tanks. The recharged acidic solution is then pumped onto the tops of the various leach dumps (U.S. EPA 1988b, 1989d).

Two additional ponds have been constructed in the Amargosa wash downstream of the raffinate pond. These ponds are designed to receive potential overflow from the raffinate pond. Overflow from the raffinate pond is initially contained by the dam at pond A, located about 500 feet downstream from the raffinate pond. Overflow from pond A is contained by the dam at pond B, located about 1,600 feet downstream from the pond A dam. Water or solution which accumulates in ponds A and B is pumped back to the raffinate pond for delivery to the leach dumps. Any overflow from pond B is diverted from the Amargosa wash and discharged to the Duval canal. This canal conveys the overflow from pond B and pond C to a natural drainage system south of the Esperanza tailings impoundment. This natural drainage directs runoff to the Sierrita tailings impoundment at a point near the reclamation pond. The Duval canal is constructed from fill material and is unlined (U.S. EPA 1989d).

PLS entering the Esperanza wash is contained downstream by dam No. 3-1/2 and further downstream by dam No. 4. The average pumping rate from the No. 3-1/2 dam is about 100 gpm, and the average pumping rate from the No. 4 dam is about 50 gpm. According to Cyprus, the dump leaching operation is isolated from the regional ground water aquifer, because the heap leach operation is completely contained by bedrock (Reed and Associates 1985).

In August 1988, overflow was observed at Ponds A, B, and C. Specifically, overflow from Pond B to the Duval canal was observed at a rate of about 75 gpm. This overflow may have resulted from recent summer rainfall in the area, or from mechanical problems with the pumping equipment (Berkeley Report 1985).

Cyprus Sierrita Corporation has recently expanded the storage capacity of the No. 4 dam. This expansion has increased the capacity of the No. 4 dam to meet design requirements for the 25-year flood. The increased storage capacity of No. 4 dam, together with the ability to pump fluids from the No. 4 dam to either the No. 1 dam or the Sierrita tailings thickeners, substantially reduces potential for overflow (U.S. EPA 1989d).

Beneficiation

Milling

A secondary and tertiary crusher system further crushes the ore from the coarse ore stockpile. Fine ore, about 0.75 inch in size, is transported by conveyor to the surface mill building. Water is subsequently mixed with the fine ore in each of 16 separate divisions of ball mills in the sulfide mill building. The resulting slurry circulates through a closed system which diverts the finest ore particles (pulp) to the concentrator flotation cells (U.S. EPA 1988b).

The capacity of the Sierrita concentrator is 100,000 tpd (Beard 1990). It uses various inorganic and organic compounds to aid in the extraction of copper and molybdenum. These compounds serve as frothers, flocculents, collectors, flotation modifiers, depressants, leachants, dewatering aids, and water treatment agents. Quantities of organic compounds used at the concentrator are regulated so that these compounds largely remain with the metal concentrate product and are not discarded with process water (U.S. EPA 1989d).

The following inorganic and organic chemical compounds are utilized during ore processing at the Sierrita mill and concentrator: lime, potassium amyl xanthate, allyl ester of amyl xanthate, alkyl sulfonate, Methyl Isobutyl Carbinol (MIBC), petroleum hydrocarbons, anionic polyacrylamides, phosphates, sodium hydrosulfide, sodium sulfosuccinate, and ferric chloride. Frothing, collector, and flocculent reagents are added to the slurry to facilitate the separation of the sulfide-bearing minerals in the flotation cells (U.S. EPA 1988b). The quantities of some of the reagents used at Sierrita mill are listed in Table 1-11 below:

Table 1-11. Reagent Quantities Used at the Sierrita Concentrator

Reagent Name	Quantity (pounds per ton of ore)
Potassium amyl xanthate	0.02
Allyl ester of amyl xanthate	0.0002
Petroleum hydrocarbons	0.004
MIBC	0.08

(Source: U.S. EPA 1988c)

The primary copper-molybdenum flotation circuit is a conventional roughing, cleaning, and recleaning operation in which the rejects from both the cleaning and recleaning stages are returned to the prior flotation step. In the copper and molybdenum flotation cells, low pressure air is introduced and forms a froth, causing the copper and molybdenum sulfide minerals to float to the surface for removal. The finely ground waste rock pulp that remains in the bottom of the flotation cells is the tailings (U.S. EPA 1988b).

All primary copper-molybdenum flotation reagents are added to the ball mill feed. Pulp alkalinity is maintained with 0.25 to 0.30 lbs of available lime per ton of mill water. Flotation reagents are selected not only for maximum recovery in the primary circuit, but also with consideration of their effect on the subsequent copper-molybdenum separation process (U.S. EPA 1988b).

Concurrent utilization of both automatic froth level control and mechanical froth skimmers is practiced on all flotation cells to ensure positive froth removal and to reduce operator dependency. Froth level is maintained with electrical resistance air-froth interface probes controlling air-operated darts in each transition and tail box. Mechanical froth skimmers on each flotation cell mechanize and meter froth removal from both sides of each flotation cell (U.S. EPA 1988b).

Molybdenum separation from copper and gangue minerals is conducted with a circuit using sodium hydrosulfide as the sole copper mineral depressant. The separation is accomplished through sulfidization in rougher, cleaner, and recleaner flotation; single-pass regrinding; and second recleaner flotation. The molybdenum flotation circuit floats the molybdenum sulfides while the copper sulfides remain as tailings. The copper sulfides (tailings) are then thickened, filtered, and shipped by rail offsite to a copper smelter. Water reclaimed from the thickeners (and waste from the tailings ponds) is recovered and recycled to the process. Each of the molybdenum flotation cells contains several stages of flotation for optimal extraction. Filtered final molybdenum flotation concentrate is dried and leached to reduce copper and lead levels to less than 0.15 percent and 0.05 percent, respectively. This process consists of a chloride metal salt leach at an elevated temperature. The leached concentrate is washed free of chloride salts and dried (U.S. EPA 1988b).

Dried, leached molybdenum concentrate is roasted in multiple hearth roasters to produce commercial molybdenum trioxide. The roasting and gas cleaning system includes two 23-foot hearth roasters, two lime-slurry scrubbers, and two acid/PCB scrubbers. The molybdenum trioxide product is processed and packaged for sale in various containers including drums, cans, and bags. The packaging facility also includes a hydraulic press and blending facilities for producing self-reducing molybdenum oxide briquettes (U.S. EPA 1988b).

Esperanza Tailings Disposal

The Esperanza tailings impoundment is located about 4 miles southeast of the Sierrita mill. Disposal of mine tailings from the Esperanza concentrator to the Esperanza tailings impoundment was continuous from October 1959 to December 1971 and from January 1973 to December 1978, and it was intermittent from January 1979 to December 1981, when the tailings impoundment was closed (U.S. EPA 1989d).

The inactive Esperanza tailings pond, which covers about 650 acres, was drained and solidified. The maximum height of the solidified tailings dam embankment face is about 100 feet. A tailings-capping program was initiated by Sierrita after it discontinued operation of the Esperanza tailings pond. This closure program required the placement of about 1 foot of alluvial soil over the surface of the former tailings pond to control dust emissions. The capped tailings pond surface area has been reseeded with native plants and grasses to control erosion and maintain aesthetics (U.S. EPA 1988b).

Sierrita Tailings Disposal

Cyprus Sierrita operates the Sierrita tailings impoundment, located about 4 miles southeast of the Esperanza tailings impoundment. The largest wastestream generated at the Sierrita operation is tailings slurry from the concentrator, which is discharged to the Sierrita tailings impoundment via pipeline. Disposal of mine tailings from the Sierrita concentrator in the Sierrita tailings impoundment began in March 1970 and still continues (U.S. EPA 1988b).

The tailings pond contains approximately 430 million tons of tailings (U.S. EPA 1986). The pond encompasses more than 4,000 acres, with a crest length along the dam face of about 25,000 feet. The tailings dam face is currently about 150 feet high (U.S. EPA 1988b).

The tailings slurry from the concentrator, which consists of finely ground host rock and water, is sent to four 350-foot (diameter) tailings center-drive thickeners. These thickeners, located south of the mill, serve to reduce the water content of the tailings. Fresh makeup water is added near the center well of each tailings thickener. Flocculent application is necessary to control slime levels in the thickeners. The thickener overflow water is recycled to the concentrator while thickener underflow pulp, which is from 54 to 56 percent solids, is discharged through two spigots per thickener into a common 42-inch (diameter) tailings line and transported by gravity to the tailings pond. The underflow pulp density is regulated with hydraulically operated pinch valves on each spigot. Pulp density controllers, receiving density measurements from gamma gauges located on each thickener underflow pipe line, adjust the control valve on each thickener spigot (Weiss 1985; U.S. EPA 1988b).

Tailings slurry is discharged to the Sierrita tailings impoundment through spigots installed on pipelines positioned along the tailings impoundment dam. The slope of the tailings impoundment surface permits decanted water to flow to the west, where the water collects at the reclamation pond. Decanted water is then recycled to the Sierrita mill (U.S. EPA 1989b).

The Sierrita tailings impoundment is divided into two separate discharge areas (the north area and the south area) by a central divider dike. Tailings deposition into the impoundment alternates between the two areas (allowing the tailings embankment in one section to be raised while the other section is being filled). Approximately 1 year is required to fill each discharge area to its temporary capacity (U.S. EPA 1989d).

Each tailings area is further subdivided by shorter berms constructed perpendicular to the embankment crest. The shorter berms are situated on approximately 1,000-foot centers around the periphery of the impoundment. These berms allow relatively even distribution of tailings discharged into the pond and direct the slimes and free water away from the embankment crest. By employing hydraulic classification (using hydrocyclones), the coarsest fraction of the tailings is deposited near the embankment crest. The surface of the tailings slopes away from the embankment at a grade of about 1.5 percent. As a result, the free-water pond is forced against the hillside in the back or the west side of the impoundment (Weiss 1985; U.S. EPA 1988b).

The tailings embankment is raised in lifts by construction of a new dike on top of the crest of the embankment following each period of filling. The dikes are constructed of the coarse tailings or beach sands deposited adjacent to the embankment crest. Bulldozers are used to spread the sand, and a sheep-foot roller is used for compaction. The height of the dike for each lift is about 8 feet. On completion of every third lift, or each 24-foot increase in embankment height, the dike for the next lift is constructed approximately 40 feet inside the crest of the embankment. Thus, a 40-foot wide bench is provided on the slope of the tailings embankment. The tailings discharge line is then dismantled, and the line segments are raised with a crane onto the new bench and reassembled. The overall slope of the tailings embankment is 3 to 1 (horizontal to vertical) (U.S. EPA 1989d).

The starter dam at the base of the tailings embankment is a homogeneous section constructed of onsite alluvial materials. The starter dam extends along the entire eastern side of the tailings impoundment and continues for several thousand feet along its north and south sides, with a maximum height of approximately 70 feet. The upstream slope of the starter dam was constructed at 1.5 to 1; the downstream slope at 2 to 1 (Weiss 1985; U.S. EPA 1988b).

According to Cyprus, chemical analyses of the tailings pond water show that it meets Federal primary and secondary drinking-water standards, with the exception of high TDS and sulfate levels. Because high-alkaline conditions are maintained during the flotation process at the Sierrita concentrator, the tailings slurry discharged to the Sierrita tailings impoundment has an elevated pH (U.S. EPA 1989d).

As water is decanted from the free-water pond, pumps positioned near the shore line in the back of the impoundment return the water to the mill for reuse. However, a fraction of the tailings pond water may percolate through the alluvial soils and reach the aquifer, which is currently about 400 feet below the ground surface. Results of the water balance calculation indicate that seepage from the Sierrita tailings impoundment has ranged from about 10,469 acre-feet in 1981 to about 5,085 acre-feet in 1987. The average volume of seepage from the Sierrita tailings impoundment during the period from 1979 to 1987 was about 41 percent of the total water delivered to the tailings impoundment (U.S. EPA 1989d).

Elevated sulfate and TDS levels have been detected in numerous ground water-monitoring wells (both onsite and offsite) (U.S. EPA 1988b). Specifically, the 13 monitoring wells in the vicinity of the tailings pond have shown "a slight increase in sulfate concentration." Eleven interceptor wells have been installed a few hundred feet downslope from the east and south sides of the tailings embankment surrounding the tailings pond. These wells may create a hydrologic barrier designed to prevent tailings pond water migration (Reed and Associates 1985; U.S. DOI, Bureau of Mines 1990a). Water obtained from these interceptor wells is pumped to milling operations. Sierrita currently collects and diverts potentially contaminated runoff (generated by heavy rainfall during winter storms) away from the tailings (U.S. EPA 1988b).

An instrumentation program has been undertaken to monitor the performance of the tailings embankment. Open-well piezometers installed along five profiles through the tailings embankment observe the location of the pentiometric surface within the embankment. In addition, several inclinometers have been installed to measure the magnitude and direction of horizontal movements within the embankment. Survey monuments are installed on the east face of the dam for deformation measurements, and subsidence monuments are located at the toe of the dam to measure potential subsidence resulting from interceptor well pumping (U.S. EPA 1988b).

Leach Circuit

Historically, copper cementation-type plants extracted copper from PLS. Copper was extracted by adding shredded iron to the PLS, which formed a copper and iron precipitate containing about 65 percent copper. In March 1987, the copper cementation precipitation plants were replaced by an SX/EW plant. The average daily production of the SX/EW plant is about 28,000 lbs of copper (U.S. EPA 1988b).

The PLS from the leach dumps (see the previous discussion) is gravity fed to the SX/EW plant, where the copper is extracted. The extraction process uses a phenolic oxime/kerosene mixture as an extractant. The copper is then stripped from the organic extractant with a sulfuric acid solution, forming a purified copper electrolyte that flows to the electrowinning cells in the plant. The following compounds are used in the SX/EW plant operation: sulfuric acid, kerosene, phenolic oxime, and cobalt sulfate (U.S. EPA 1989d).

The SX plant consists of 25 mixer-settler units that mix the aqueous and organic components. Both components are immiscible and are allowed to separate. The mixer-settler units are arranged in five identical, parallel rows, each utilizing three extraction units, one wash unit, and a stripping unit. Organic flow is countercurrent to the aqueous flow and flows continuously through the stripping, washing, and extraction phases. After extraction of the copper from the organic liquid, the barren aqueous solution, or raffinate, is collected in a raffinate pond and pumped back to the leach dumps at a rate of 3,300 gpm (U.S. EPA 1988b; U.S. DOI, Bureau of Mines 1990a).

The electrowinning plant contains 60 fiberglass cells, each containing 22 anode sheets and 21 cathodes. The plant has a capacity to produce 30,000 lbs of finished cathodes per day (U.S. EPA 1988b).

The SX/EW process is a closed system; there are no effluent discharges. All of the reagents and intermediate liquors are stored in covered storage tanks in accordance with applicable regulations (not specifically identified in the available references) (U.S. DOI, Bureau of Mines 1990a).

Other Wastes Associated with the Sierrita Operation

Other wastestreams disposed of at the Sierrita site include sanitary wastes, discharges from the rhenium extraction plant, and discharges of tailings from the Cyprus Twin Buttes Corporation oxide plant. These wastestreams discharge to the Sierrita tailings thickeners (U.S. EPA 1989d).

Sanitary Wastes

Sanitary wastes generated at the Sierrita operation discharge to a series of septic systems serving different parts of the facility. A total of nine septic systems are currently in use at the Sierrita operation. Estimated daily discharges to septic systems range from about 75 gallons for septic systems No. 4 and No. 9 to about 9,500 gallons for septic system No. 1 (U.S. EPA 1989d).

Septic system No. 1 serves the Sierrita mill area. Effluent discharged from septic system No. 1 is treated with potassium permanganate disinfectant prior to discharge to the Sierrita tailings impoundment to mitigate potential biological contamination. Mitigation of nitrate and phosphate discharges is accomplished through dilution in the tailings wastestream. Average daily discharge of water to the Sierrita tailings impoundment from the Sierrita mill for 1987 was more than 14 MGD; as stated above, the effluent discharge from septic system No. 1 is about 9,500 gallons per day (gpd), or about 0.07 percent of the total water delivered to the tailings impoundment (U.S. EPA 1989d).

Septic system No. 6 serves the Sierrita truck shops, change rooms, and engineering offices. It receives about 7,750 gpd. Effluent discharged from septic system No. 6 is delivered to a leach field. Seepage from the leach field is believed to migrate to bedrock and flow along the Esperanza wash channel. This seepage water may then flow into springs, which discharge to Esperanza wash upstream of the No. 4 dam. These discharges are contained in the wash by the No. 4 dam, and are pumped to the Amargosa wash drainage for processing at the SX/EW plant (U.S. EPA 1989d).

According to Cyprus, "no usable ground water underlies the septic tanks, and there is no opportunity for discharge from septic tanks to enter the regional aquifer system" (Reed and Associates 1985). Accordingly, Cyprus does not propose to submit a ground water-permit application for any of its septic tanks (U.S. EPA 1989d).

Rhenium Plant Wastes

The element rhenium occurs in minor amounts with molybdenum ore. In 1982, a rhenium plant was constructed in the vicinity of the Sierrita mill to recover rhenium oxide from the acid demister effluent generated by the molybdenum roaster. The recovery process is a closed-system, ion-exchange, evaporation-crystallization process that has a capacity to recover up to 15 lbs of rhenium per day (as ammonium perrhenate). Reagents used at the rhenium plant include caustic soda, sodium hypochlorite, and ammonium thiocyanate. The design and operation of the rhenium plant requires that the ammonium thiocyanate remain with the final product or be recycled (U.S. EPA 1988b, 1989d).

During the rhenium extraction process, liquor from the rhenium plant flows to the Sierrita tailings thickeners at about 5 gpm. The pH of this solution is approximately 11 (U.S. EPA 1989d).

In conjunction with the operation of the rhenium plant, a holding pond for storage of acidified liquor to be processed by the rhenium plant is located west of the Esperanza tailings impoundment. Fluids contained in the lined rhenium pond are part of the production circuit (U.S. EPA 1989d).

Twin Buttes Oxide Plant Wastes

Cyprus Sierrita Corporation-Twin Buttes began short-term operation of the Oxide plant at the Sierrita site in March 1989. The Oxide plant is comprised of a vat leaching circuit and an SX/EW plant. Discharge of tailings from the vat leaching circuit of the plant is piped to tailings slurry pipelines at the Sierrita operation for delivery to the Sierrita tailings impoundment. Oxide tailings from the Twin Buttes operation are from about 45 to 50 percent solids and have a pH of about 2.0. Average flow rate for tailings from the Twin Buttes Oxide plant is 1,800 gpm. Because the tailings discharged from the Sierrita operation are alkaline, tailings from the Twin Buttes Oxide plant are neutralized, and any dissolved constituents should be precipitated in the Sierrita tailings impoundment (U.S. EPA 1989d).

4. Cyprus Bagdad Copper Company; Bagdad Mine; Yavapai County, Arizona

The Bagdad Mine is located in western Yavapai County, 27 miles from Hillside and 120 miles northwest of Phoenix, Arizona. Its operating capacity is 95,000 metric tpy of copper. The ore body is a chalcocite-enriched zone in quartz monzonite porphyry that outcrops from an irregular stock. In 1948, mining operations changed from block-caving to open-pit production (U.S. DOI, Bureau of Mines 1965a; Niemi 1980). The mine-strip ratio in 1991 was 0.9 to 1, with an average ore grade of 0.44 percent copper. The concentrator capacity is 75,000 tpd since completion of the expansion in 1990 (Cyprus Minerals Company 1992).

The Bagdad operation consists of an open-pit copper-molybdenum mine, a 55,000-tpd concentrator, a dump leach operation, and an SX/EW plant. A $21-million expansion program underway was to have added a fifth grinding line to the existing mill and to have increased production capacity from 15 to 20 percent by mid-1990 (Beard 1990).

Extraction

Mining

Mining is conducted using electric shovels. Trucks are used to haul the ore to the primary crusher and dump. Bagdad uses an in-pit primary crusher and 6,400-foot-long conveyor system that transports the ore to the concentrator (Niemi 1980). The Bagdad open-pit mine generates about 20 million tons of overburden and 17 million tons of waste rock each year (for an accumulated lifetime volume of 424 million tons of waste material). The mine overburden and waste rock areas cover approximately 1,470 acres (Cyprus Minerals Company 1992).

The mine waste disposal areas do not have any type of impermeable liners or leachate collection systems. Storm water runoff from the waste disposal areas drains to the open pit where it is used as make-up process water (Cyprus Minerals Company 1992).

Leaching Operations

Sulfide ores that have a high-oxide content are placed in dumps and leached for 60 days before being sent to the concentrator (Beard 1990). Dump leach operations began in 1960. Cyprus is currently operating a number of leach dumps for sulfide ore that has a high-oxide content. The two main leach dumps are located along Copper Creek, and a third is located along Mineral Creek. Several other small dumps are located along Copper, Niagara, and Alum Creeks (see Figure 1-22)

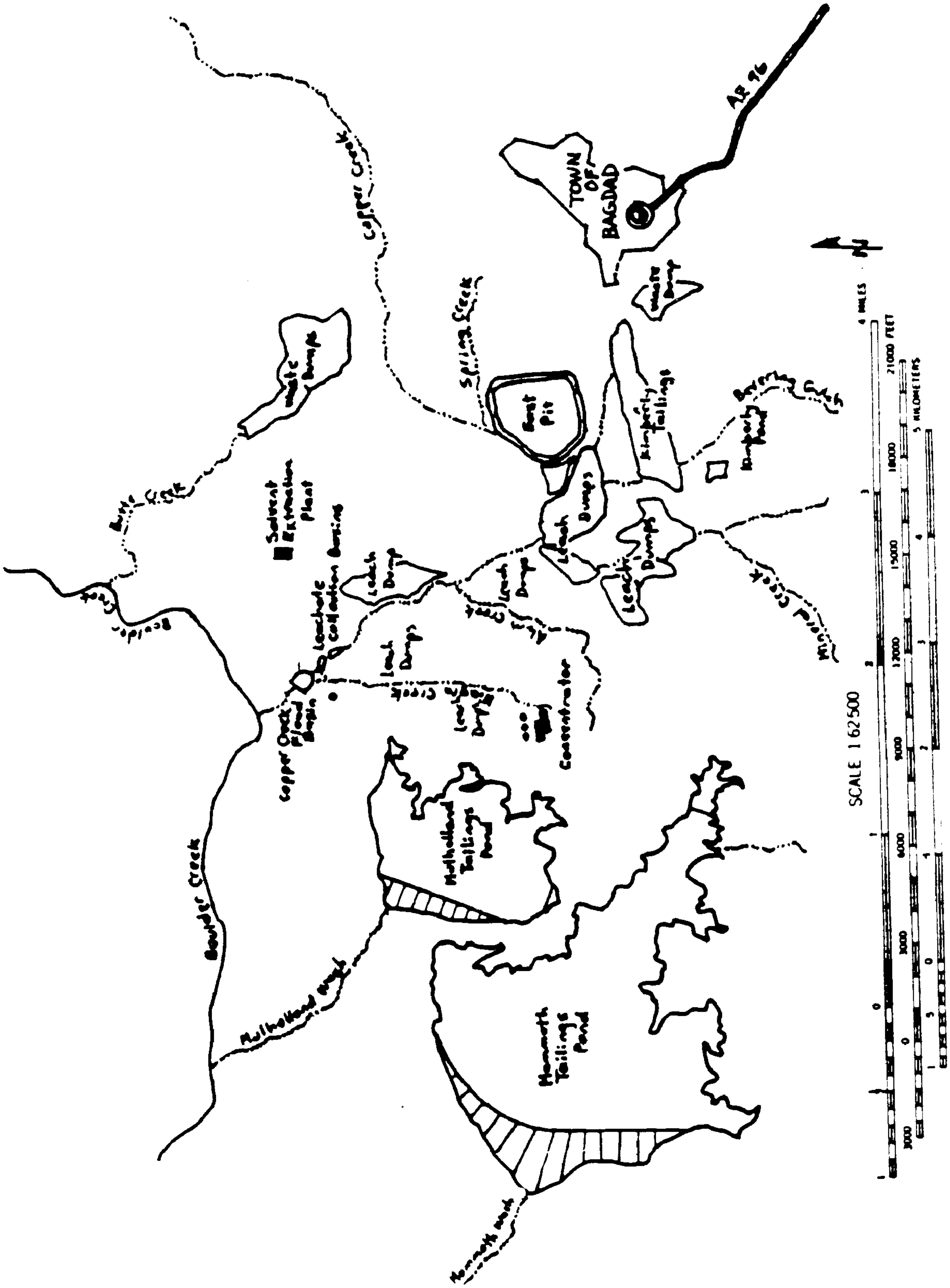

Figure 1-22. Locations of Bagdad Leach Dumps

(Source: U.S. EPA 1987)

(U.S. EPA 1987). The dumps contain over 600 million tons of oxide ore, and only 10 to 15 percent are active at any given time (U.S. EPA 1989e).

Low-grade, mine-run ore is used in the leach dumps. Ore having an oxide copper content of at least 0.07 percent, but not having sufficient copper to justify mining, is generally deposited on the leach dumps (Cyprus Minerals Company 1992). The dumps have been built directly on the existing topography, utilizing the natural drainage created by the contours of several canyons located on the property to divert and collect the PLS (U.S. EPA 1989e). "Leach dump site preparation has been minimal. Convenient, comparatively narrow, steep-walled canyons are selected. They are then subjected to critical examination for major fractures, etc." (Weiss 1985).

Haulage trucks carry the ore from the pit to a leaching area where it is dumped and spread by a bulldozer. Lift heights range from 40 to 300 feet depending on the particular topography of the land.

After a lift is competed, the surface is ripped to a depth of about 5 feet, and the solution distribution system is installed (U.S. EPA 1989e).

The rice paddy flooding method was first used at the leach dumps for solution introduction. Since that time, lixiviant is dispersed with a wiggler-type sprinkler system between 3,000 to 4,000 gpm. This reduces channeling and ensures uniform coverage. The lixiviant solution consists of dilute sulfuric acid (containing 8 g/l of sulfuric acid) having a pH of approximately 1.0. Initially, each lift is leached until the surface begins to pond due to a buildup of iron salt precipitates. After this period, the dumps are allowed to rest. The ratio between the leach period and the rest period is approximately 3 to 1.

Pregnant solution is collected at the base of each pile in a lined pond or large plastic-lined surge holding tanks. The pregnant solutions from the leach piles (except Allum Creek reservoir) are then combined in a series of ponds located in an ephemeral tributary to Copper Creek. The number, specific locations, and construction of these ponds are unclear from the available references. Pregnant solution from the Allum Creek reservoir is pumped to the top of the Copper Creek dump, through which it is allowed to percolate. The average concentration of copper in pregnant solution ranges from 1.07 to 1.89 g/l. The pregnant solution from the series of ponds is eventually metered out through Niagara dam into a trench and a collection reservoir. The dam is made of concrete and keyed into the bedrock of the surrounding hillside. In 1989, both the trench and the collection reservoir were lined with 100-mil polyethylene liner. The pregnant solution collected in the reservoir is then pumped to the SX/EW plant (Niemi 1980; Weiss 1985; U.S. EPA 1989e; Beard 1990).

The techniques used in 1989 to install the liner in the collection trench and pond were similar to those used in the lining of heap leach piles. The area chosen for the trench and pond was first excavated and rough graded. After the excavation and grading were completed, a fill subgrade material was hauled to the site and compacted in layers. After the subgrade material was in place and had been adequately compacted and finished, the liner was installed. The liner was cut and spread by hand in the trench and collection reservoir areas. The seams were then welded and tested to ensure their integrity. After the liner had been installed, a pumping system was installed to carry the pregnant liquor to the copper recovery plant (U.S. EPA 1989e).

After the copper has been recovered in the SX/EW plant, the barren solution is recycled to the leach dumps. Approximately 100 tpd of acid is added to this solution to reduce the pH. Mine water is used as makeup water. Between 500 and 800 gpm of water is produced in the mine pit. The mine water collected in the pit is used only in the mining operations (as makeup water) and is not discharged off the property (U.S. EPA 1989e). Cyprus has installed two wells as part of a ground water-monitoring system, and more are scheduled (Cyprus Minerals Company 1992).

Beneficiation

Milling

The beneficiation operation consists of a sulfide concentrator and a leach plant. Sulfide ore is transported 6,400 feet via conveyor belt to the coarse ore stockpile at the concentrator. At the concentrator, the ore is further crushed and ground by autogenous and ball mills. The concentrator flotation cells use sodium ethyl xanthate as a primary collector, Phillips Co. 400 as a secondary collector, and ore prep F-547 as frother. The pH is kept at 10.5 by applying lime. Column cells are utilized in the molybdenum flotation circuit. The primary depressant used in the molybdenum plant is sodium hydrosulfide (Cyprus Minerals Company 1992).

Tailings Disposal

There are two large tailings ponds (the Mulholland and Mammoth ponds) and a third small tailings pond (the Kimberly pond) at the Bagdad Mine (as shown in Figure 1-22) (U.S. EPA 1987). Approximately 25 million tpy of tailings are disposed of in the tailings ponds. The ponds contain about 270 million tons of tailings and cover approximately 1,200 acres. Tailings are piped by gravity to the disposal site, where they separated and dispersed by cyclones. Cyprus uses the center-line method of raised embankment berm erection. Barge-mounted pumps reclaim up to 21,000 gpm of water from the tailings impoundment (Cyprus Minerals Company 1992). The tailings ponds are not lined.

Leach Plant

Historically, a cementation plant was used at the Bagdad Mine between 1961 and 1970 (Cyprus Minerals Company 1992). Pregnant liquor was pumped from the holding tanks to precipitation cells filled with shredded scrap iron, which precipitated copper while solubilizing iron. A bleed stream was introduced to control the pH and prevent iron scaling in the pipes (Weiss 1985). As the quality of the scrap iron degraded and demand increased, undesirable insoluble materials, such as calcium and iron, required more cleaning by magnetic separators. Increased processing and cleaning costs necessitated introducing a new technology.

In 1970, Cyprus opened its SX operation. Cyprus' SX process produces 60,000 lbs of pure copper cathode per day (Cyprus Minerals Company 1992). Pregnant solution is passed through four extraction and stripping stages to recover the copper ion from the solution. The process utilizes a special reagent (LIX-64N) which has a high affinity for copper ion in a weak acid solution and a low affinity for other metal ions (Niemi 1980).

The reagent operates on hydrogen ion cycles, which, in general, proceed in the following manner. The reagent, carried in an organic medium, is intimately contacted with aqueous leach solutions in the extraction system. There, hydrogen ions are exchanged for copper ions. Sulfuric acid in the leach solution is regenerated while the copper is extracted. The organic medium containing the copper passes to the stripping system where it contacts aqueous copper sulfate in the presence of sulfuric acid; there, copper ions are exchanged for hydrogen ions. The reagent is regenerated and recycled to the extraction system. The enriched copper sulfate solution is essentially an impurity-free, concentrated electrolyte from which high-quality cathode copper is produced by electrowinning. The barren solutions are returned to the dumps after the copper has been extracted (Bagdad Copper Corp. undated).

Electrowinning

The electrowinning plant has two stages, a starter sheet and a commercial cathode production section. The starter sheet operation uses 48 stainless steel blank cathodes and 49 anodes in 6 cells. Commercial cathode production has 50 cells with 48 cathodes and 49 anodes per cell (Cyprus Minerals Company 1992).

Wastewater Management

The natural contours of the land divert the runoff from the surrounding hills around the mining and leaching operations. Precipitation falling within the mine area itself is collected in either the pregnant solution collection ponds or the mine pit. An overflow floodplain reservoir has been constructed to protect against a 100-year flood event. Runoff collected in the floodplain reservoir is pumped into the pregnant solution collection reservoir and used in the leaching circuit (U.S. EPA 1989e).

5. Magma Copper Company, Pinto Valley Mining Division

The Pinto Valley mine is an open-pit mine located adjacent to the Inspiration mine, near Globe in central Arizona. The Pinto Valley Division (PVD) consists of the Pinto Valley unit (Castle Dome and Copper Cities), the Miami unit, and the No. 2 Tailings Hydraulic Operation (Beard 1990).

The Pinto Valley ore body is fairly typical quartz monzonite porphyry-type deposits, which intrudes granite porphyry and is crosscut by diabase dikes. The ore body is bounded by major fault systems and is highly fractured. The host rock for the deposit at the Miami mine is Precambrian pinal schist, which is partially covered by the Gila Conglomerate. The principal copper mineral is chalcocite with minor amounts of chalcopyrite, bornite, covellite, malachite, azurite, chrysocolla, cuprite, and native copper. The mine produces both copper and molybdenum concentrates (Mining Magazine 1975; U.S. EPA 1989e).

Extraction

At the Pinto Valley unit, mining is accomplished with electric shovels, and trucks haul 64,500 tpd of ore to the concentrator (Magma 1992). The overall waste-to-ore ratio is 1.5 to 1. The total excavation is estimated to produce 500 million tons of waste and leach-grade material. The maximum pit size will be 6,000 feet long by 3,500 feet wide and 1,450 feet deep (Mining Magazine 1975). The Copper Cities unit consisted of an open pit operation and concentrator which were active between 1954 and 1975. All mining and milling operations ceased in late 1975. Initial open-pit mining began at the Pinto Valley site around 1972 (U.S. EPA 1989e). A dump leaching operation and a 7,000-gpm SX/EW plant are also located at the site (Magma 1992).

Solution Mining

Leaching at the Pinto Valley site consists of eleven waste dumps. The dumps currently contain approximately 297 million tons of leachable waste ore. About 28 million tons of leachable waste are being added to the dumps each year.

Active dump leaching operation began in 1981 when construction of the SX plant was completed. The operation currently covers an area of approximately 6,570 acres (470 acres of which are covered by leach dumps). Approximately 85,000 tons of copper are produced annually from the Pinto Valley operation (15 percent of which are produced from the leaching operation) (U.S. EPA 1989e).

Currently, only about 120 acres of the dumps are being leached at the Pinto Valley site. Trucks haul the material from the mine pit to the leach dump. The leach dumps at the Pinto Valley site have been constructed on existing topography with no prior subsurface preparation. After each lift is completed, the surface is ripped to a depth of approximately 3 to 4 feet using a cat ripper, and the distribution system is installed. The distribution system consists of 2-inch perforated Drisco pipe spread over the dump (U.S. EPA 1989e).

The leach solution applied to the Pinto Valley dumps contains approximately 2.25 g/l of sulfuric acid and has a pH ranging from 1.7 to 1.8. It is applied continuously until the surface of the dump begins to pond, indicating excess precipitation of iron salts. The pregnant leach liquor contains about 0.95 g/l of sulfuric acid and has a pH ranging from 2.0 to 2.1; it is collected in the drainage below the dumps. According to company personnel, the subsurface area on which the leaching operation is conducted consists of bedrock (U.S. EPA 1989e).

Copper Cities Unit

Pregnant liquor from the leach dumps at the Pinto Valley site is collected in an unlined reservoir behind Gold Gulch dam No. 1. An overflow catchment dam (Gold Gulch dam No. 2) has been constructed down the valley to retain any flows that may result from an upset condition. Both dams have a rock shell with a clay core and are key-cut grouted to bedrock. Pumps lift the solution through 1 mile of pipe to the SX/EW plant (U.S. EPA 1989e).

Miami Unit

When the dump leaching operation at the Copper Cities mine site was closed in 1982, Pinto Valley Copper Corporation constructed a system of diversion trenches to channel overflows from the leach pile collection sumps and storm water runoff onto the tailings pond for evaporation. The trench system was designed to handle flows resulting from a 100-year storm event and was lined with riprap to prevent erosion (U.S. EPA 1989e).

In situ stope leaching began on a small scale in 1942; full-scale leaching began when the underground mine was closed in 1959. The leach solution is percolated through the caved area by underground injection and surface spraying. The pregnant leach liquor is collected at the 1,000-foot haulage level and pumped to the surface (U.S. EPA 1989e).

The PLS contains 0.57 g/l of sulfuric acid and has a pH of 2.2. The raffinate from the SX plant contains 1.6 g/l of sulfuric acid and has a pH ranging from 1.7 to 1.8. The raffinate is recycled back to the caved area for distribution as part of the leach solution. The Miami mine's *in situ* stope leaching operation has a positive water balance. This indicates that the underground mine is acting as a sump, collecting water from surrounding areas and (at least in part) preventing the migrating of leachate away from the mined area (U.S. EPA 1989e).

In 1989, Pinto Valley began hydraulic remining of the No. 2 tailings dam sediments. These tails were slurred in an open launder system to a vat leaching operation that recovered copper from the tails (Beard 1990).

Hydraulic Mining

Magma has undertaken hydraulic remining of the No. 2 tailings pile, which contains approximately 38 million tons of abandoned tailings. Tailings are reclaimed in the area lying adjacent to the drainage way of the main street in the town of Miami. The hydraulically remined tailings are then reprocessed by vat leaching (McWaters 1990). Note, however, these are historical tailings and are not a result of flotation (U.S. DOI, Bureau of Mines 1992). The hydraulic mining operation uses up to four 4-inch hydraulic mining jet monitors, feeding two separate educator pump sets capable of pumping 523 gpm of water at 28 bars of pressure. The hydraulic monitors are automatically controlled. The hydraulic mining peptizes the tailings into a 32.4 percent solids slurry. Production from this operation is expected to be about 14 short tpd of copper (Magma 1992). Both the hydraulic mining and vat leaching operations take place in Miami, Arizona, approximately 65 miles from the San Manuel Mine (ADEQ 1992).

Beneficiation

Milling

Mined ore is loaded on 190-ton trucks and delivered to a primary crusher. Secondary and tertiary crushing are accomplished using separate cone crushers. Each crusher is equipped with belt conveyors and ore bins (Mining Magazine 1975).

The concentrator contains six ball mills with dedicated cyclones operating in a closed-circuit configuration. The flotation unit is a standard copper-molybdenum-sulfide-type, arranged in a two-staged circuit. The first stage is comprised of 6 rows of 14-cell rougher-scavenger trains with cyclone hydraulic separators. The second stage is comprised of two trains of four recleaner, six cleaner, and four scavenger cells with cyclones (Mining Magazine 1975). The concentrates and cathodes (from the SX/EW plant) are shipped to San Manuel, Arizona, for processing (Beard 1990).

The concentrator's milling capacity is 70,000 tpd. The addition of the 14 rougher cells in 1989 was expected to increase the copper metal recovery rate 2 percent; actual recovery rates are slightly greater than 90 percent (Magma 1992).

Tailings Disposal

Prior to 1975, tailings (from the copper-molybdenum rougher and cleaner flotation stages) were thickened in three 350-foot diameter thickeners. Thickener overflow was recycled into the mill water supply, while the thickened tailings were diverted to one of five tailings ponds. Water from the tailings disposal areas was also reclaimed for reuse in the mill (Mining Magazine 1975).

Because of the steep topography at Pinto Valley, five separate tailings ponds were built. Each starter dam formed a homogeneous embankment and was designed with a drainage blanket that was connected to a filter bed extending through the embankment for water removal. Any water that percolates through the drainage blanket is caught by a small dam downstream and is pumped back into the water system. The foundations of the dams were excavated to solid bedrock to form a strong and relatively incompressible base for the starter dams (Weiss 1985). Beginning in 1990, Pinto Valley began cycloning tailings for berm construction, replacing the previously used spigotting method.

Pinto Valley revegetated the Solitude tailings pond near Miami, Arizona, beginning in 1959. These efforts have involved spreading native soil over the top and side slopes of the 550-acre tailings pond and planting native plants. The soil was obtained from the surrounding hills to minimize haulage costs. The area surrounding the tailings pond was stripped, and vegetation and the surficial layer of dirt was excavated. A layer of this dirt (approximately 25 cm deep) was spread over the entire surface of the tailings pond. Available mining equipment was used to remove, haul, and spread the soil. The surface of the tailings pond and the area from which the dirt had been obtained were then seeded with a 10-seed mixture of native plants. The project required approximately 20 months to complete (U.S. EPA 1989e).

Vat Leaching

The remined tailings slurry is pumped to vat leaching operations (located 65 miles away in Miami, Arizona) in separate pipelines or from each educator sump. An agitated vat leach and SX is used to process the tailings. The vat leach system is capable of holding 323,000 ft^2 of material and is completely computerized. The process units are constructed of two Miami-type thickeners that were refurbished with HDPE linings.

The vessels, pumps, and plumbing are constructed of corrosion- and abrasion-resistant materials, such as stainless steel, polyethylene, and rubber (McWaters 1990).

The first step in the vat leaching process is to sort the remined tailings at a screening tower where 10+ mesh material is rejected. The undersize, screened material is initially piped to an agitated vessel where sulfuric acid is added to lower the pH to about 1.5; then, it is pumped to the first thickener (McWaters 1990).

Overflow from the first thickener is pumped to a second agitation vessel. There, it is washed with raffinate and sent to the second thickener. The overflow from the second thickener flows to a suction vessel, which feeds a group of water pumps that supply the hydraulic mining circuit. The circuit's underflow reports to a vessel where it is again washed with raffinate and pumped to the disposal-reclaim circuit. The leachate overflow from the No. 1 thickener is flocculated, clarified, and sent to a PLS pond, where it is mixed with leachate from another leaching circuit before flowing to the SX process (McWaters 1990).

Tailings from the vat leaching process are pumped in a 13-inch HDPE line to Copper Cities deep pit for disposal. The pipeline is located in a containment ditch. Also located in the containment ditch, adjacent to the slurry pipeline, is a 12- to 14-inch HDPE return-water pipeline. Both pipelines are continuously monitored by pressure and end flows to identify and contain leaks. The containment ditch leads to an HDPE-lined 333-square-foot containment pond (McWaters 1990).

Magma Copper Company has installed spill prevention measures at the vat leaching operations. According to Magma, its engineers over-designed the system to last longer than anticipated and installed leak detection systems (McWaters 1990). A barge with four centrifugal, elastomer-lined pumps transports the washed reprocessed tailings 4.5 miles from the process plant to an abandoned mine pit (the Copper Cities deep pit). According to Magma, this pit is geologically separated from the nearest known aquifer by the "relatively impervious" Miami fault system (McWaters 1990).

Leach Circuit SX/EW Plant

SX/EW Plant - PVD, Miami Unit

Copper is recovered from the leach solution at an SX/EW plant. The SX plant has been in operation since 1976 and is centrally located only a short distance from the heap leach pile. It originally consisted of two circuits of three extractor and two stripper cells. Each of these circuits was designed to handle 1,505 gpm of pregnant solution (McWaters 1990). The original SX/EW plant was designed and located so that it could be expanded at minimal cost, and, in 1989, it was expanded to handle 6,022 gpm. The solutions from *in situ* leaching (see below) are also fed to this plant, which has been expanded to a production capacity of 50,000 tpy of copper (Beard 1990).

Before the SX phase, pregnant solution is subjected to a flotation process to recover any entrained organic particles that may interfere in the electrowinning process. Clarity of the PLS is important because particles can form the nuclei for organic- and aqueous-phase emulsions and sludges (Beard 1990; McWaters 1990). After flotation/clarification, the PLS is fed into four asymmetrical SX circuits. The organic ion exchange reagent used to extract the copper consists of 7 percent LIX 984 by volume in a kerosene solution (Beard 1990; McWaters 1990).

In the extraction circuits, more than 90 percent of the copper is transferred in the organic phase. The loaded organic solution is then pumped through two circuits of two mixer-settler vessels in series. The copper is encouraged to transfer to the electrolyte by low pH and intimate mixing. The electrolyte is fifty times more concentrated than the original PLS. Before flowing to the electrowinning tankhouse, the electrolyte is passed through a bedded filter to remove particulates. The depleted PLS (raffinate) is predominantly recycled to the *in situ* leaching field (McWaters 1990).

The effluent from the SX circuit is of sufficient strength to be directly pumped through electrowinning cells. The electrolyte is warmed in heat exchangers where the heat sources are steam and the heat of the electrolysis process itself. The copper is then electrowon directly onto stainless steel cathodes rather than on the standard copper starter sheets (McWaters 1990).

As of 1987, the electrowinning tankhouse plant had 60 cells. Fifty-four cells were for commercial cathode production and six were dedicated to starter sheet production. More recently, the 60 cells were converted, and the facility was expanded with 42 new cells (McWaters 1990).

Wastewater Management

Diversion ditches and collection ponds have been constructed around the entire Copper Cities leach pile to catch any runoff and leachates. Overflow catchment dams have been constructed to retain any flow from these containment areas during upset conditions. Solutions collected in the ponds and catchment areas are diverted to the inactive tailings ponds, where the liquid is evaporated (U.S. EPA 1989e).

6. Cyprus Miami Mining Corporation, Cyprus Miami Mine and Smelter, Gila County, Arizona

The site is located 6 miles west of Globe, Arizona, between the towns of Claypool and Miami. The Inspiration operations consist of open-pit copper mines (formerly called Inspiration mines); leach dumps; a 24,000-tpd concentrator that is on stand-by status; an SX/EW plant; a 450,000-tpy electric furnace smelter and associated acid plant; an electrolytic refinery; and a 135,000-tpy rod plant (Beard 1990). The operations stretch from the Lower Oxide mine to the confluence of Miami wash, Bloody Tanks wash, and Russell gulch; a distance of over 7 miles. The Inspiration operations are part of a larger mining district near the Towns of Miami and Globe that includes Pinto Valley Copper Company's Oxide mine operations and Old Ranchers Exploration Bluebird mine (U.S. EPA 1987).

The site operation, a mine for leach operation, produces 110,000 short tpy of cathode copper, copper rod, and blister copper (U.S. DOI, Bureau of Mines 1992). The operation originally consisted of a fully integrated facility with mine, mill, concentrator, leach plant, and smelter (Weiss 1985). However, the pits are now only used as holding ponds for leaching solutions; the concentrator and tailings disposal ponds have been shut down since 1986 (U.S. EPA 1987; U.S. DOI, Bureau of Mines 1990a). Only leaching of oxide ore continues today (except for processing units).

The site and the larger mining district of which it is a part are extremely complex. There are numerous interrelated units under different ownership. Many changes in ownership and operational status have occurred, resulting in numerous inconsistencies among the available references.

Extraction

Mining

The mine was an underground block-caving operation, which was converted into two open-pit mines in 1948. During the active mining period, material was mined from several pits at the Inspiration site, including upper and lower Oxide pits, Barney north pit, Red Hill pit, Live Oak pit, Bluebird pit, Thorton pit, and Joe Bush pit (U.S. EPA 1987). Of these, only the Bluebird pit is active. The Bluebird pit was acquired from Ranchers Exploration and Development Corporation in July 1984. Approximately 80,000 tpd of ore is being mined at the Bluebird pit. The Live Oak pit is being dewatered (U.S. EPA 1989e).

The ore occurs as a complex mixture of disseminated chalcocite or oxidized copper minerals in an alteration zone surrounding the monzonite porphyry intrusive (U.S. DOI, Bureau of Mines 1965a). The ore was mined at a rate of 50,000 tpd with electric shovels and hauled by truck to primary crushers and waste dumps (Beard 1990).

The mine produced approximately 11 million tpy of waste rock, totaling 391 million tons for the life of the mine. This material was placed in dumps at the edge of the pit for permanent disposal. Figure 1-23

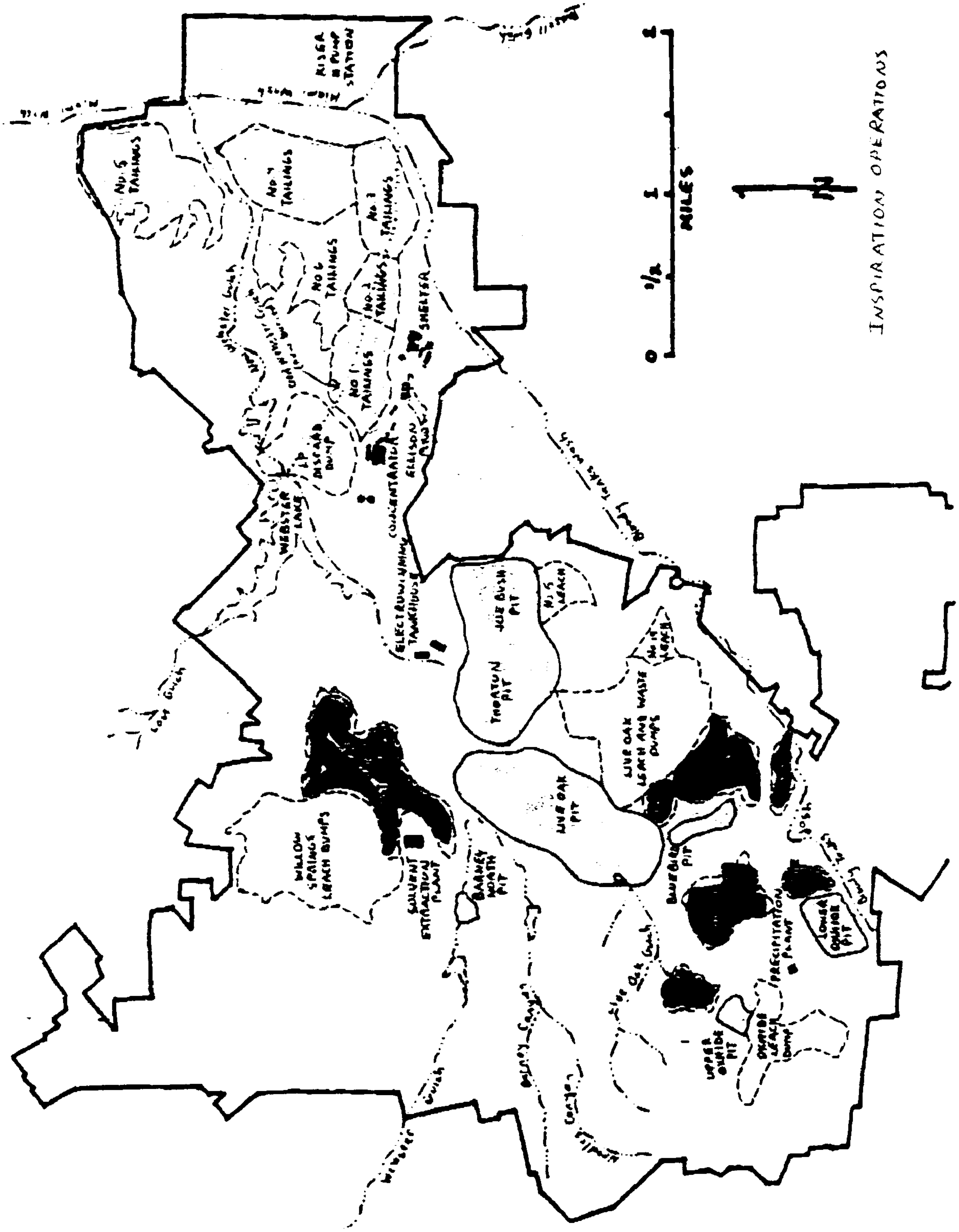

Figure 1-23. Locations of Cyprus Miami Mine and Smelter Waste Dumps

(Source: U.S. DOI, Bureau of Mines 1990) shows the location of the waste dumps. These dumps have no impermeable liners or leachate collection systems (U.S. DOI, Bureau of Mines 1990a).

Leach Dumps

Dump leach operations were started in 1955. The mine produced approximately 23 million tpy of leach material, totaling approximately 294 million tons during the life of the mine. This leach material covers 5.4 million square yards. Inspiration mine has five leach dump areas: the Willow

Springs leach dump, the Live Oak leach and waste dumps, the Nos. 5 and 19 leach dumps, and the Oxide leach dumps (U.S. EPA 1987; U.S. DOI, Bureau of Mines 1990a). Low-grade ore was sorted according to the primary mineralogy and transported to the appropriate leach dump site. Leach dumps Nos. 5, 9, and 27 are low-grade oxide ore dumps, and leach dumps Nos. 19, 28, 33, 34, and 35 are a mixture of oxide and sulfide low-grade ore dumps (U.S. EPA 1987).

Inspiration operates two separate leach circuits: a conventional dump leaching operation and a ferric cure leaching operation. Ore containing above 0.3 percent copper as chalcocite and oxides is delivered to the ferric cure circuit, while ore containing less than the 0.3 percent copper cutoff is delivered to the conventional leaching circuit. These circuits are operated in series (i.e., the PLS recovered from the conventional operation is used as the leaching solution for the ferric cure operation) (U.S. EPA 1989e).

The majority of the leach dumps in the old Inspiration property were built on the existing topography. The underlying surface was cleared of existing vegetation and graded to channel the PLS into the collection ponds located at the toe of the pile. The underlying surface of the old Bluebird leach dumps was also cleared of vegetation and dressed, and the soil was cemented and covered with dilute tar for curing and sealing (U.S. EPA 1989e).

New lifts of leach material are built on previously leached dump piles. Prior to the placement of a new lift, the surface of the dump is ripped to a depth of approximately 6 feet. The ore is then hauled to the pad by trucks and spread with bulldozers. After the lift has been completed, the surface of the lift is ripped and the solution distribution piping is laid (U.S. EPA 1989e).

The leaching solution distribution system consists of 2-inch piping perforated with 1/8 inch holes. The leaching solution contains approximately 5 to 15 g/l of sulfuric acid and has a pH of 1.0. It is applied to each lift for a period of up to 125 days at varying flow rates. A flow rate of approximately 15,000 gpm is maintained for the entire system (U.S. EPA 1989e).

Dumps and collection ponds have been constructed on a surface which Cyprus described as a tight formation of bedrock of relatively impermeable granite. To optimize drainage of leach liquors, the pads were constructed with compacted native soil and have collection ponds at their bases. A ground water monitoring-well system has also been installed around the periphery of the leach dumps (U.S. EPA 1986, 1987, 1989e; U.S. DOI, Bureau of Mines 1990a). No information is available on the analyses of samples collected from the monitoring-well system.

A stepped leaching process is used to enrich leach solutions. Lixiviant solutions from the barren pond are applied to the surface of leach dumps Nos. 5, 9, and 27 (the oxide waste dumps) in a conventional acidic leach-type operation. After the lixiviant percolates through these leach dumps, PLSs are collected, cured by adding acid, and applied to the "ferric cure" leach dumps Nos. 19, 28, 33, 34, and 35 (mixed oxide and sulfide ores). This procedure allows the "ferric ion" solution generated in the oxide leach circuit to react with the sulfide minerals, thus enhancing oxidization. Some of the higher-grade PLS is recycled to the "ferric cure" leach dumps. The remainder of the PLSs are collected in a system of small reservoirs and routed either to a precipitation plant located 4 miles southwest of the concentrator near the Oxide mine area or to a SX plant located 2.5 miles west of the concentrator near the Willow Springs leach dump (U.S. EPA 1987).

The leaching technique used in the ferric cure operations is unique because the leach pads are carefully constructed in uniform dimensions. The leach pads are generally rectangular, measuring approximately 250 feet wide by 600 feet long. A pad is stacked to a height of approximately 30 feet. After completion of the pad, the pile is cured. The cure solution contains 200 g/l of sulfuric acid and between 2 to 3 g/l of ferric iron. Sufficient cure solution is applied to the pad in two separate applications. The pad is then allowed to cure or rest for 15 days, after which it is rinsed with conventional leach solution for up to 120 days. It is estimated that, at the end of the leaching cycle, approximately 70 percent of the copper has been recovered (U.S. EPA 1989e). It is unclear whether the ferric cure operation is a separate heap-leach-type operation rather than a dump leach operation.

Diversion ditches have been dug around some of the dumps to divert runoff from the piles into collection ponds. In addition, diversion ditches have also been dug to divert surface runoff from outside the property away from the dumps (U.S. EPA 1989e). The leach solutions from each of the leaching circuits are collected in the ponds at the base of each dump. Most of these collection reservoirs are unlined. All of the retaining dams used to hold the pregnant solution are made of concrete with either clay or concrete cores. All of the dams have been keyed into the bedrock in the existing hillsides to prevent leakage. The pregnant solution collected in the ponds is pumped to an SX/EW plant for copper recovery. The SX/EW plant currently receives and processes approximately 4,500 gpm of pregnant liquor. The barren solution (or raffinate) produced by the SX/EW plant is then recycled into the conventional leaching circuit (U.S. EPA 1989e). Also, sulfuric acid (generated by roasting and converting of concentrates) is captured at the acid plant in the scrubber units and is added to the raffinate produced at the SX plant before it is recycled to the leach dumps (U.S. EPA 1987).

Beneficiation

Tailings Disposal

The Cyprus Miami Mine and Smelter operation's sulfide concentrator and crushing units were decommissioned in January 1986. At that time, Cyprus suspended operation of the six tailings ponds (Nos. 1 through 6). They are located east of the concentrator and on the northwest side of Bloody tanks and west of Miami wash (U.S. EPA 1987). No additional information is available on the operation of the flotation units, including the types of reagents used.

When operating, the mill used a system of sloughing tanks with thickeners to dewater the tails to between 22 and 31 percent solids. An upstream construction method was used for the tailings impoundments. They were constructed of a series of 40-foot raised embankment berms (Taggart 1945). Seventy-foot-high trestles were constructed on the inside berm around the tailings pond. The trestles supported an open-chute launder with automatic desliming cones spaced at 12-foot intervals along each trestle. These cones were used to dispense and segregate the tails at regular intervals around the periphery of the pond. The coarse sediments from the desliming cones created a series of piles at the base of the trestles which combined to form the new berm. When a new raise was completed, the old trestles were abandoned and new trestles were constructed about 80 feet behind the base of the new berm. About 50 percent of the water used in the concentrator was recovered from the tailings ponds by decant systems using pumps (Taggart 1945; U.S. EPA 1986). The tailings ponds contain a combined volume of approximately 480 million tons of tailings materials (U.S. EPA 1986; U.S. DOI, Bureau of Mines 1990a).

According to Cyprus, one of the tailings ponds is "closed" and dry. No additional information is available on the closure status of the other tailings ponds.

The tailings ponds were constructed without liners. Five ground water wells (located downgradient of the base of the tailing dam) are monitored on a quarterly basis (U.S. EPA 1986; U.S. DOI, Bureau of Mines 1990a). Additionally, a 25-well monitoring system was installed along the downgradient side of the property (U.S. DOI, Bureau of Mines 1990a). No information concerning the results of the quarterly monitoring was available.

Leach Circuit

PLS from the Oxide leach dumps, along with Mine Water Drainage (MWD) from the Live Oak pit, make up the influent to the precipitation plant. The effluent from the precipitation plant, known as "iron-launder off-solution" is combined with MWD from the lower Oxide pit and recycled back to the Oxide leach dumps. Excess barren leach solution is stored in the upper Oxide and Bluebird pits (U.S. EPA 1987).

In the precipitation process, PLS filters through surface impoundments containing submerged scrap iron. The copper ions in the solution replace the iron ions in the scrap to form copper flakes. The iron scrap is reused until it is consumed. Periodically, the ponds are hosed down to collect copper scale, which is allowed to dry and then shipped to a smelter (U.S. EPA 1987).

In the SX process, a chelating agent is added to the pregnant solution to form an uncharged extractable complex of copper. The uncharged complex then partitions into an organic solution with a low dielectric constant. Ionic compounds, such as dissolved iron, remain with the leach liquor. Usually, kerosene is the organic solution used. Through the process of complexing the copper and partitioning the complex into a relatively small volume of kerosene, the copper is concentrated. The remaining leach solution (raffinate) contains low concentrations of copper and low pH. The loaded organic solvent, containing the extracted copper, is directed to strippers that reverse the process and redissolve the copper in relatively concentrated sulfuric acid. The copper acid solution (known as electrolyte) proceeds to the tankhouse, where the process of electrowinning removes the copper from solution. Oxidized metal in solution is reduced (gains electrons) through the application of an electric current. The positively charged lead in the electrolyte, known as the "cathode," is usually made of a thin copper starter sheet. Ions of copper migrate through the electrolyte and plate on the starter sheet. This process produces nearly pure plates of copper which are known as "cathodes." The electrolyte recycles back through the strippers and the tankhouse (U.S. EPA 1987).

Water and Wastewater Management

The mine operates an integrated water and wastewater management system that consists of three principle circuits: the industrial/fresh-water circuit, the process-water-leaching circuit, and the process-wastewater circuit. A 1987 flow diagram of water and wastewater management (including all three circuits) at the Inspiration mine site is presented in Figure 1-24

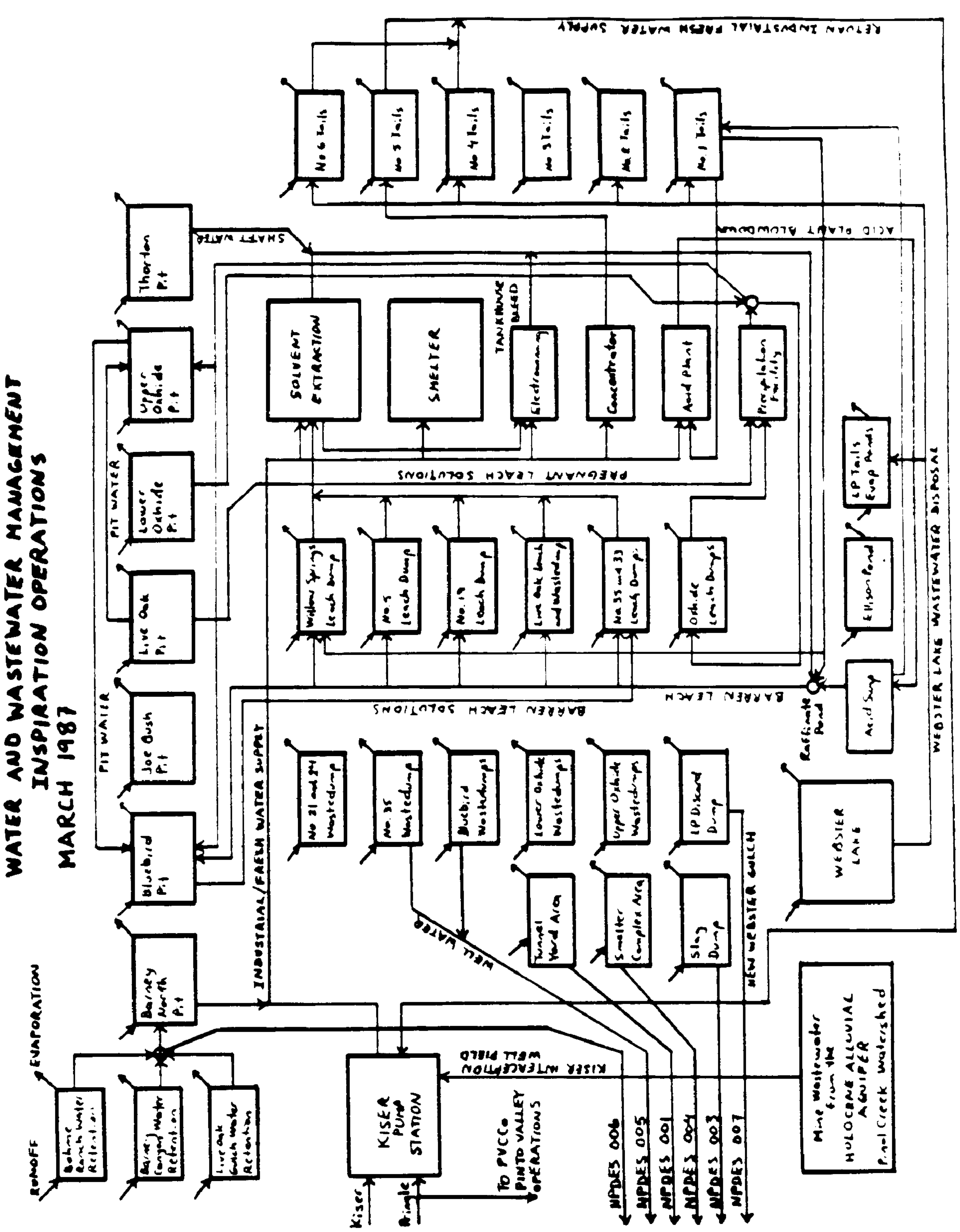

Figure 1-24. Water and Wastewater Management at Inspiration Operations

(Source: U.S. EPA 1987)

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In the past, the water and waste circuits were managed to maximize the efficient production of copper and to minimize the water and wastewater disposal costs. However, the mine now has altered the water and wastewater circuits to reduce the volume of process wastewaters by isolating this circuit from watershed runon and to decrease the generation of process wastewater by increasing reuse and evaporation (U.S. EPA 1987).

Industrial/Fresh-water Circuit

The industrial/fresh-water circuit consists of a fresh-water segment and an industrial-water segment. Both systems are recirculated through the No. 5 tailings pond and the Kiser water treatment plant and pump station. The fresh-water segment provides potable water for use as acid plant blowdown water, power plant cooling water, steam plant compressor water, slag dryer launder water, crusher dust suppression and lubrication water, makeup water for the SX/EW plant, and domestic water for housing and shops. The industrial-water segment provides water for use as anode casting, converter, and blister copper cooling water in the smelter; wash rack water; and truck fill water for dust suppression on roads. A large flow of water from the industrial-water segment is circulated through the inactive concentrator and tailings circuit to prevent tailings slurries from silting up the internal drainage systems (U.S. EPA 1987).

The fresh-water segment of this circuit is fed from three sources. First, relatively clean water is pumped out of the Gila Conglomerate formation from the Pringle well field. The well field is located midway between the headwaters of the perennial portion of Pinal Creek and the creek's confluence with the Salt River. Second, any seepage beneath the tailings ponds is intercepted and diverted to this segment. Third, well No. 62 of the Kiser well field extracts water from both the Gila Conglomerate formation and the overlying Holocene alluvium. This well is located near Russell gulch. All water from each of the three sources enters the fresh-water circuit at the Kiser water treatment plant and pump station, located near the Miami wash/Bloody tanks wash/Russell gulch confluence (U.S. EPA 1987).

The industrial-water segment of this circuit is fed by five sources. First, overflow of excess potable water from the fresh water segment is routed into the industrial water segment. Second, water is pumped out of the Gila Conglomerate formation by production well No. 20 of the Kiser well field. This well is located between the No. 3 tailings pond and the Miami wash. Third, contaminated wastewater in the Holocene alluvium is recovered by Kiser interception wells Nos. 1 and 2. Fourth, collected wastewater from the steam plant compressor and the slag dryer launder is recycled. Fifth, storm water runoff from the undisturbed areas west of the mining operations behind three fresh water retention dams (the Bohme ranch, Barney canyon, and Live Oak gulch) is collected and recycled. These three impoundments then feed the Barney north pit. The first three sources (production well No. 20, Kiser interception wells Nos. 1 and 2, and fresh-water overflow) enter the industrial-water circuit at the Kiser water treatment plant and pump station. The wastewater from the steam plant compressor and slag dryer launder drains into a portion of the industrial-water segment known as the anode sump. The storm water runoff in the Barney north pit is used in only the truck fill and wash rack water portion of the industrial-water segment (U.S. EPA 1987).

Water circulated through the inactive concentrator and tailings water (from the anode pond, tankhouse facility shops, and a storage tank) drain into tailings pond No. 5. Internal drains convey any seepage from the No. 5 decant pond along with additional seepage from the Nos. 4 through 6 decant ponds to the industrial/fresh-water circuit at the Kiser water treatment plant and pump station (U.S. EPA 1987).

To reduce the amount of water in the industrial/fresh-water circuit, Inspiration has applied for an NPDES permit to discharge water from the storm water retention dams into Bloody tanks wash. In addition, Cyprus Miami Mining sells Pringle well-field water to Pinto Valley. It is routed away from the industrial/fresh-water circuit by diversion through the Burch pump station to Pinto Valley's operation (U.S. EPA 1987).

Process Water Leaching Circuit

The process water leaching circuit is designed to facilitate leaching of copper from the dumps and provide for evaporation of process wastewater. The process water leaching circuit relies on the continuous recirculation of leach waters through the dumps. Inspiration replaces evaporation and bedrock infiltration losses at the leach dumps with rain-fall runoff from the dumps; drainage and pit water from the Bluebird, Live Oak, lower Oxide, upper Oxide, and Thorton pits; and supplements of process wastewater drawn from the top of the No. 1 tailings pond. The process water leaching circuit consists of two segments that are defined by the method of copper recovery (the leach segment feeding the precipitation plant and the main leach segment feeding the SX plant) (U.S. EPA 1987).

Evaporation losses in the SX leach segment are replaced with mine water from the Bluebird pit and process wastewater. The Bluebird pit supplies mine water that is used to supplement the lixiviant solution used at leach dumps Nos. 28, 34, and 35. Process wastewater feeds the raffinate pond. The process wastewater is made up of acid plant blowdown drawn from the acid sumps, wastewater mixtures drawn from the top of the tailings pond No. 1, shaft water from the Thorton pit area, and electrowinning tankhouse bleed. Excess raffinate is stored in the Bluebird pit (U.S. EPA 1987).

The surface impoundments and units integrated into the leach circuit include:

- Eleven surface-water-holding reservoirs: Nos. 19, 27, 28, 33, and 35; C; Live Oak; Barney canyon; Bohme ranch; on-pond collecting; and Davis canyon

- Eleven ponds: 001 NPDES; 003 NPDES; 004 NPDES; No. 4 decant; No. 5 decant; Main sewage disposal; Ellison; aqueous feed; raffinate; acid sump; and LP discard evaporation

- One basin: Honeyman overflow basin

- Seven water-holding pits: lower Oxide; upper Oxide; Bluebird; Live Oak; Thorton; Barney north; and Honeyman overflow basin (U.S. EPA 1987).

The locations of these units are shown in Figure 1-25

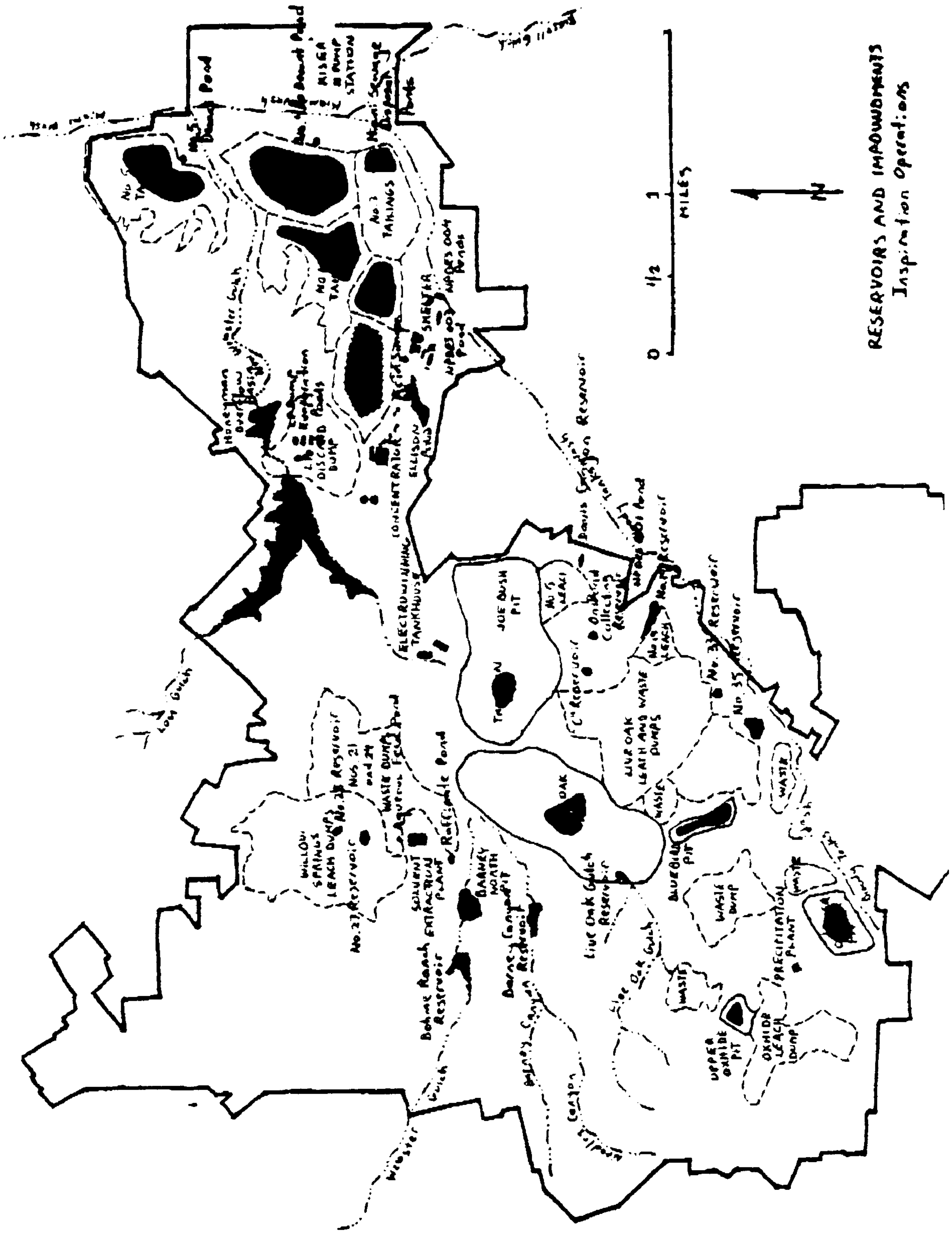


Figure 1-25. Reservoirs and Impoundments at Cyprus Miami Mine and Smelter Operations

(Source: U.S. EPA 1987)

. No information is available concerning the design of these units (including whether or not they are lined).

Process Wastewater Circuit

The process wastewater circuit handles three types of process wastewater: contaminated Webster Lake water, storm water runoff (except that from active leaching areas), and sewage from the Town of Miami (U.S. EPA 1987).

Since the formation of Webster Lake in 1941, Miami (formerly Inspiration) and Pinto Valley mines historically have used it as a storage and disposal reservoir for various process water leach solutions and process wastewaters. In the August 28, 1986, Cyprus Miami Mine and Smelter listed 15 separate current and past sources of wastewater discharge to Webster Lake. Among these sources are storm water runoff; vat leach iron-launder off-solution; *in situ* leach off-solution; Live Oak iron-launder off-solution; raffinate discard; Thorton shaft pumpage; black copper dump leach solution; tankhouse water treatment brine; secondary crusher dust control water; Webster east pumpage; Thorton pit water; slime pit iron-launder off-solution; and Copper Cities leach dump solutions. The other two sources were not identified. The combined wastewater feed into Webster Lake between 1941 and 1986 from these sources was over 2,485 million gallons (U.S. EPA 1987).

To assess the impact of the mine's operation on the surrounding ground water system, Cyprus Miami conducted a comprehensive water-quality study around several active and abandoned mines in Arizona's Miami/Globe area. This study required a monitoring network composed of 113 new wells.

This network included existing water-supply wells and wells converted to hydrologic monitoring nests of shallow small-diameter wells; wells drilled adjacent to deeper existing wells; and deep wells (U.S. EPA 1989e).

As a result of the study, the practice of disposing of waste solutions into Webster Lake was discontinued. In addition, Inspiration was required to drain Webster Lake by July 28, 1986. Prior to 1986, Webster Lake held an average of 1.46 billion gallons of process wastewater (U.S. EPA 1987).

To drain the lake, withdrawn lake water was applied to the LP evaporation ponds and the Nos. 1, 2, 4, and 6 tailings ponds for evaporation. Water from Webster Lake was also mixed with the process water leaching circuit by combining it with the acid sump surges and tailings pond No. 1 water, which feeds the raffinate pond. In addition, three fresh-water-retention structures were constructed to divert surface-water runoff from the watershed of the lake (U.S. EPA 1987).

Cyprus Miami is required to control surface-water runoff from areas not under leaching by retention and evaporation or by discharge through an NPDES-permitted outfall. Inspiration's NPDES Permit No. AZ0020508 authorizes the discharge of storm water runoff from waste dumps near the tunnel yard at Outfall 001. Storm water runoff from the slag dump south of the smelter yard is discharged at Outfall 003. Storm water runoff from the smelter yard area is released at Outfall 004. Runoff from waste dumps Nos. 21 and 24 and the lower and upper Oxide waste dumps are retained and evaporated. Wastewater from the Bluebird pit dewatering wells is discharged from Outfall 005. Storm water runoff, captured in the three new fresh-water-retention facilities west of the mining operations, is discharged at Outfall 006. Finally, seepage accumulating in the New Webster Gulch is released at Outfall 007. The Town of Miami pumps municipal sewage onto the top of the No. 3 tailings pond (U.S. EPA 1987).

7. ASARCO Inc.; Mission Mine; Pima County, Arizona

ASARCO's Mission Mine is located approximately 15 miles south-southwest of Tucson. The facility occupies approximately 23 square miles (see Figure 1-26

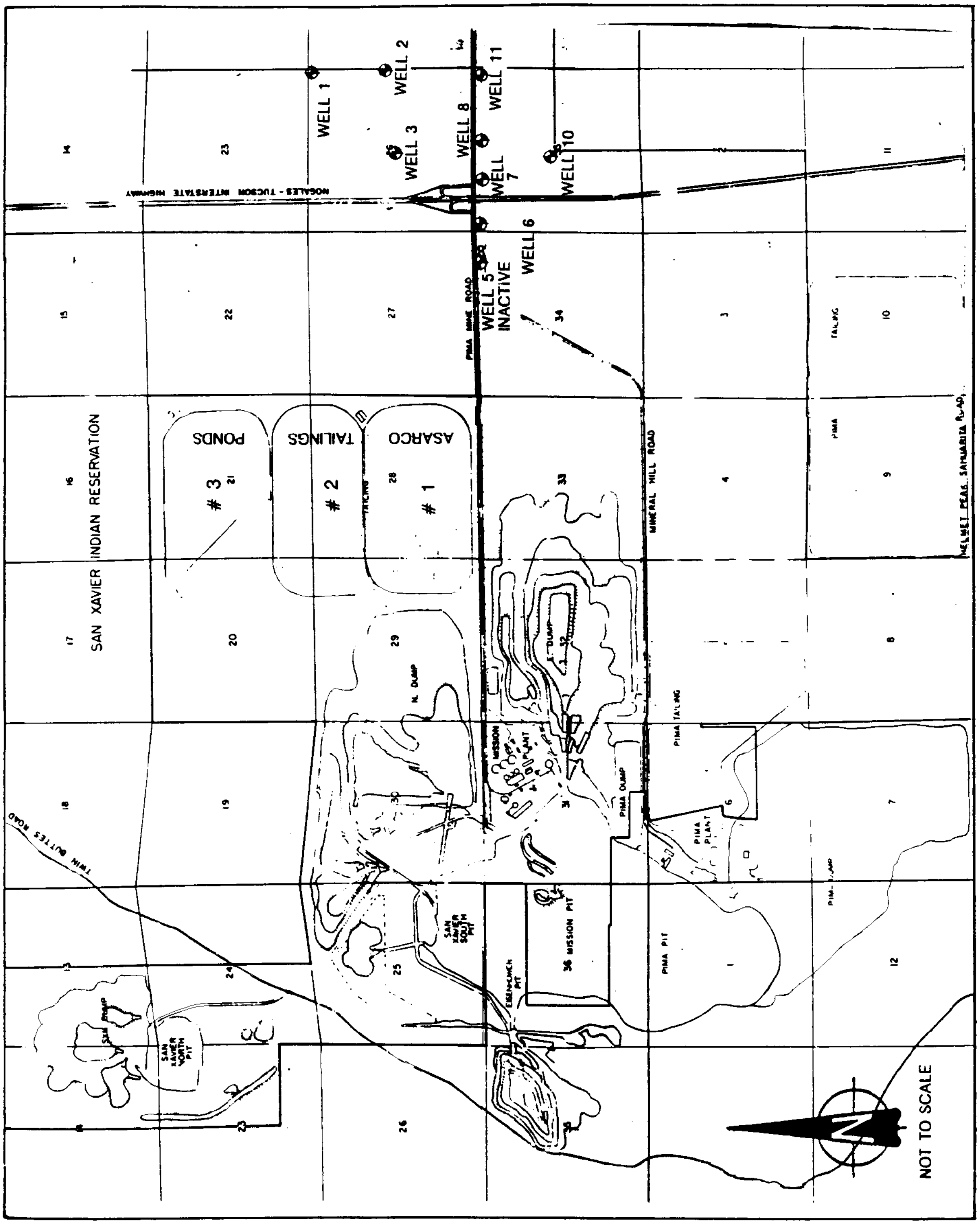


Figure 1-26. ASARCO Mission Complex Facility Map

(Source: U.S. EPA 1988c)

) (ASARCO 1992). Development of the Mission Mine began in the 1950s, with full-scale production beginning in the early 1960s. The adjacent Pima mine began operation in 1951 as an underground stoping operation and was converted to an open-pit operation in 1955. In 1982, while under different ownership, the mine was shut down (U.S. DOI, Bureau of Mines 1992). The Mission Mine expanded when it consolidated with the Pima mine in September 1985. At that time, the Pima concentrator was closed and dismantled (U.S. EPA 1988c). The Mission unit now consists of the Mission, Eisenhower, San Xavier, and Pima mines consolidated into one large open-pit mine referred to as the "Mission complex." Also included is the smaller San Xavier north pit (Beard 1990). In 1991, ASARCO completed a $100 million expansion of the Mission complex. The complex was expanded from 90,000 st of copper to 124,000 st. Reserves at the Mission complex are estimated at 600 million st, grading 0.681 copper (Mining Engineering 1991).

Geologically, the mine is located within a faulted complex of sedimentary, volcanic, and plutonic rocks along the eastern pediment of the low-lying Sierrita Mountains. During the Laramide Orogeny, the Paleozoic and Mesozoic rocks were intruded by the northwest-trending Laramide porphyrys. The area was segmented by the San Xavier low angle-thrust fault. Subsequently, the deposit was covered by alluvial fan deposits and volcanics (U.S. DOI, Bureau of Mines 1965a; Weiss 1985; U.S. EPA 1986, 1988c).

The ore body was mineralized by hydrothermal fluids resulting from the emplacement of a Laramide-age quartz monzonite porphyry. The copper porphyry sulfide ore occurs primarily in sedimentary rocks, dominated by carbonates. The principal sulfide mineral is chalcopyrite. Other minerals include pyrite, chalcocite, covellite, minor bornite, galena, sphalerite, pyrrhotite, and minor molybdenite. Silver content in the sulfide concentrate is approximately 0.11 oz/st. Other copper oxide minerals include malachite and azurite, which are copper carbonates (U.S. DOI, Bureau of Mines 1965a; Weiss 1985; U.S. EPA 1986, 1988c).

Extraction

The pit is relatively deep for an open-pit mine and has a high strip ratio of 2.5 to 3.0 at a cut-off grade of 0.30 percent copper (Weiss 1985).

Ore is extracted using conventional open-pit mining methods by drilling groups of blast holes 50 feet deep, then filling them with an ANFO blasting mixture. Electric shovels and scrapers load the blasted, fractured ore and/or waste rock into 170- and 200-ton electric drive dump trucks, which deliver the ore to three primary crushers. The waste rock and alluvium are subsequently transported to various dumps (Weiss 1985).

The Mission Complex produces approximately 150,000 tpd of mine waste rock, which is removed from the pit and placed in dumps for potential future leaching. Mine dumps are generally of a side-slope type. The mine waste dumps cover approximately 3,175 acres. Because mining has been completed in some areas of the pit, backfilling with waste rock is also practiced (Weiss 1985).

Vat Leaching

In the San Xavier north and south properties, the upper zone of the sulfide ore body is oxidized; its principal mineral is silicate chrysocolla. The oxide ore vat leach plant operated from 1972 through 1979, when the oxide ore reserves were depleted (U.S. EPA 1988c). After crushing, the ore was delivered to one of nine large concrete leach vats, where it was mixed with sulfuric acid. Copper was leached out of the ore into the sulfuric acid solution, forming a copper-rich pregnant solution. The pregnant solution was processed by cementation-precipitation methods in ponds filled with scrap iron. Copper precipitate (containing some iron) was recovered by spraying water over the scrap iron on a vibrating screen. The precipitate was then recovered in a sump and delivered to an offsite smelter, where the copper was processed. Scrap iron was reused until consumed. The iron-rich barren solution was sent to an evaporation pond located on the North Dump. This evaporation pond is presently closed and covered by waste rock (U.S. EPA 1986, 1988c).

Beneficiation

Mission Mill

The sulfide plant began operation in 1961 with four rod-and-ball mills (U.S. EPA 1988c). The sulfide ore currently undergoes secondary and tertiary stage crushing to about 0.75-inch in diameter, and it is then stockpiled. At the mill, water is mixed with the ore to form a slurry, which is ground by six rod-and-ball mills and two single-stage ball mills. The slurry is classified and the fines (about 0.008 inches in diameter) are diverted to the concentrator flotation cells (U.S. EPA 1988c; ASARCO 1992).

The concentrator is a double-circuit, sulfide copper-molybdenite system. In the primary flotation cells, low-pressure air is introduced, forming a froth and causing the copper and molybdenum-sulfide minerals to float to the surface as froth or sulfide rougher concentrate. Frothing and collector reagents are added to the slurry to facilitate the separation of the sulfide-bearing minerals in the flotation cells (Table 1-12). The finely ground waste material that remains in the bottom of the flotation cells (tails) is sent to the tailings ponds. The sulfide concentrate is reground and floated two more times in the cleaner-recleaner flotation cells. Remaining tails are also sent to the tailings ponds. Before the molybdenum plant closed, the sulfide concentrate was further processed to recover molybdenite. Additional reagents were added to float the molybdenum sulfides. The molybdenite flotation cells included a rougher section and seven cleaner-recleaner sections (U.S. EPA 1988c).

Table 1-12. Reagents and Ore Used at Mission Mine

Reagents	Usage (in pounds per ton of ore)
Lime	2.0
Pine oil	0.01
Potassium amyl xanthate	0.007
Dithiophosphate	0.006
MIBC	0.05

(Source: U.S. EPA 1988c)

Prior to 1988, the concentrator's capacity was about 28,000 tpd (ASARCO 1992). During 1988, the concentrator's capacity was increased to 40,000 tpd. This was accomplished by lengthening the 10.5-foot (diameter) ball mills from 15 to 18 feet and installing 2 new ball mills (salvaged from a Sacaton, Arizona, mill); adding additional rougher flotation capacity in 1991; and installing six 8-by-52-foot column flotation cells for cleaners (Beard 1990; ASARCO 1992).

Mission Tailings Disposal

Waste tails from the concentration processes are thickened to approximately 50 percent solids in the four thickening units and transported by gravity in a slurry line to three tailings ponds. Water reclaimed from the thickeners and tailings ponds is recycled as process water for the sulfide plant (U.S. EPA 1986, 1988c).

The three tailings ponds are located on San Xavier Indian Reservation land. The combined area of these ponds is 1,500 acres, and the total accumulated volume of tailings disposed of is approximately 250 million tons (U.S. EPA 1988c).

The tailings impoundments were constructed with compacted alluvial starter dams. Berms are constructed with a dragline. Sixty- to 80-foot center decant towers are used for water recovery. The tailings impoundments do not have liners and are underlaid by native soil comprised of sand and gravel. The tailings areas were preslimed before deposition (ASARCO 1992). In 1991, 208 gallons of water per ton of ore was required. One third of this was lost to seepage, which percolates through the alluvium and reaches ground water (about 250 feet below the surface). Ground water may have elevated sulfate and TDS levels (U.S. EPA 1988c). The remainder was lost to evaporation and containment in the dam (ASARCO 1992).

A U.S. DOI, Bureau of Mines, study found that the overall permeability of the tailing ponds was 2×10^{-6} cm/sec (U.S. EPA 1988c). In addition, Mission pit mine water, which is produced at about 200 gpm, is pumped to mill reclaim tanks. No other information was available on the generation and management of mine water. Runoff from surrounding areas is diverted around the tailing impoundments (U.S. EPA 1988c).

According to the Inspection Report, ASARCO Inc.'s Mission Mine had ground water near its tailings pond sampled. Previous analyses by the Pima Association of Governments for inorganics documented a release to the environment, although the Target Compound List was not analyzed. The Pima Association of Governments documented elevated levels of sulfates and TDS immediately downgradient from the tailings ponds. However, elevated levels of TDS or sulfates above enforceable EPA secondary drinking-water standards were not present in the nearest domestic water well located 1 mile downgradient from ASARCO's tailings impoundments. Tailings impoundments were moistened to control dust emission (U.S. EPA 1987).

FIT concluded that no additional sampling was required under CERCLA, as prior sample data and investigations indicated that there were no hazardous substances present in the ground water and no potential public health threat (U.S. EPA 1987).

South Mill Tailings Disposal

When the Pima concentrator was active, tailings were disposed of in upper and lower tailings ponds. When one of the ponds was full, the tails were routed to other ponds to allow the berms to be raised and dried while the other pond was filling. Spigotting was used to deliver nonsegregated tailings to the ponds; this supplied the sediment for the next level of dam construction to be built utilizing the upstream method (Weiss 1985). Periodically, coarse materials were raked back onto the berm by shovels or a drag line to form 15-foot dikes (Weiss 1985; U.S. EPA 1986).

About 75 percent of the water in the Pima tailings pond was recycled to a reservoir and subsequently returned to the process (U.S. EPA 1986). The reservoir was constructed with an 8-inch reinforced concrete bottom with 6-inch gunite walls. The reservoir was divided in half by a spillway which allowed settling in the feed compartment prior to overflow to the discharge side of the reservoir pond. The water level was kept constant by a probe, which regulated the inflow of fresh water. Sludges, which occasionally accumulated in the feed compartment, were pumped out to the tailings pond by a system of three stationary pumps (Weiss 1985).

The Pima tailings ponds are now completely capped with alluvial material to prevent blowing dust; they are well-stabilized with vegetative material (Weiss 1985).

APPENDIX 1-C

NPL SITE SUMMARIES RELATED TO COPPER MINING ACTIVITIES

1. Silver Bow Creek - Part of the Clark Fork Superfund Sites
2. Milltown Reservoir - Part of the Clark Fork Superfund Sites
3. Celtor Chemical Works, Humboldt County, California
4. Torch Lake, Houghton County, Michigan

1. Silver Bow Creek - Part of the Clark Fork Superfund Sites

Site Overview

The Silver Bow Creek site is one of four separate but contiguous Superfund Sites located near the City of Butte, along the course of the Clark Fork River in southwestern Montana. The four sites, known collectively as the Clark Fork Superfund Sites, are the Anaconda Smelter site, the Milltown Reservoir site, the Montana Pole site, and the Silver Bow Creek/Butte Area site. All four sites have the potential to contaminate Silver Bow Creek and/or the Clark Fork River. Also, Milltown Reservoir has the potential to contaminate the sole-source aquifer below Missoula. The Superfund effort in the Clark Fork Basin encompasses the largest geographic area of all Superfund assignments in the United States. Except for the Montana Pole site, contamination at the sites is primarily mining wastes and heavy metal-laden soils and water. The Montana Pole site, which lies adjacent to the Silver Bow/Butte Area site, is contaminated with wood-treating wastes, unrelated to mining activity (U.S. EPA 1991).

The Silver Bow Creek/Butte Area Superfund Site is the largest and most complex of the four sites. Silver Bow Creek has historically received discharge from mining, smelting, wood treating, and other industrial sources for over 110 years. The Silver Bow Creek/Butte Area site includes the Cities of Butte and Walkerville (population 38,000), the Berkeley Pit (a nonoperating open-pit copper mine); numerous underground mine works (operated by New Butte Mining, Inc.); the Continental Pit (operated by Montana Resources); Silver Bow Creek; Warm Springs Ponds (mine tailings); and Rocker Timber Framing and Treating Plant. The approximate size of the Silver Bow Creek/Butte Area site is 450 acres. The Silver Bow Creek site was added to the NPL in September 1983. Originally, the site encompassed the Silver Bow Creek floodplain from Butte (downstream) to Warm Springs Ponds. Remedial Investigations were initiated in this area in 1985. In November 1985, the site boundaries were expanded to include Butte (U.S. EPA 1991).

Operating History

In the years following the discovery of gold (in 1864), the Butte area became an internationally recognized mining center with over 300 combined copper and silver mines and 8 smelters in operation by 1884. The Butte area has been mined almost continuously for 110 years. Most of the ore mined in Butte was shipped 26 miles west to the smelting complex in Anaconda, Montana (a separate Superfund Site); however, ore was also smelted in any of eight smelters in the Butte area. Smelting continued in the Butte area until the Washoe Smelter became operational in Anaconda in 1902. By the 1950s, the Anaconda Company (purchased by Atlantic Richfield Company in 1979) had consolidated all mining activity in the area (U.S. EPA 1991).

Copper, silver, gold, zinc, lead, manganese, and molybdenum have been mined by both underground (vein) mines and open-pit mines in the Butte area. Major underground mining activity took place from the late 1880s through 1960. Over 3,500 miles of underground workings exist in the area; some of the vein mines reached over 5,000 feet in depth. The Berkeley Pit, an open-pit mine, operated from 1955 to 1982. It is estimated that over 3,500 miles of underground mine workings are interconnected with the Berkeley Pit. The pit is over 1 mile deep and 1.5 miles wide at the rim. Mining companies installed a pumping system to dewater the underground mines and the Berkeley Pit during active mining. In the 1950s, bulkheads were installed underground to inhibit the flow of water between mines and the pits and create two underground flow systems, the east camp (includes the Berkeley Pit) and the west camp. These bulkheads were installed to improve the efficiency of pumping operations (U.S. EPA).

In 1964, a mill was constructed in Butte to concentrate the copper sulphide ore from the Butte mines. High-grade ore was processed through the mill and smelter, while lower-grade ores were leached with acid water from the mines in large leach dumps located near the tailings disposal area. The mill tailings were impounded behind a 2-mile-long dam northeast of the mining operation (Yankee Doodle Tailings Pond). Prior to 1911, when pollution control measures were first initiated, all mining, milling, and smelting wastes were discharged directly to Silver Bow Creek (U.S. EPA 1991).

The first pollution control measures consisted of ponds created by dams built to trap and settle the mining wastes (sediments, tailings, and sludges). In 1911, a 20-foot high dam was erected on Silver Bow Creek, creating Warm Springs Pond 1. Another dam, 18 feet high, was erected on the creek in 1916, creating Warm Springs Pond 2. (This dam was extended to a height of 23 feet.) A third dam, 28-feet high (built between 1954 and 1959), was primarily for sediment control. This dam was eventually raised to 33 feet. In 1967, Pond 3 was converted to treat mill losses, precipitation plant spent solution from Butte operations, and overflow from the Opportunity Ponds. Treatment consisted of adding a lime/water suspension to raise the Ph of the surface water in Silver Bow Creek and precipitate heavy metals in Pond 3. The three ponds are currently used to physically, chemically, and biologically treat Silver Bow Creek surface water through sedimentation and chemical and biological precipitation of heavy metals (U.S. EPA 1991).

Mining activity in the Butte area continued until 1982, when the Berkeley Pit was closed. At this time, the pumps dewatering the mine were shut down and the underground mines began to flood. As the water levels reached the bottom of the Berkeley pit, it began to fill. In 1986, mining activity resumed, although on a smaller scale. The Continental Pit, operated by Montana Resources, produces approximately 50,000 tons per day of copper/molybdenum ore; New Butte Mining, through its underground operation, produces approximately 500 to 1,000 tons per day of silver, lead, and zinc ore. Montana Resources operates an onsite mill to concentrate its ore, discharging the tailings to the Yankee Doodle Tailings Pond area; New Butte Mining ore is shipped offsite for milling and smelting (U.S. EPA 1991).

Environmental Damages and Risks

The wastes generated by mining, milling, and smelting activities are sources of contamination for soils, surface water, and ground water. Contamination is occurring through blowing dust, contaminated runoff and contaminants leaching through the soil into the ground water. Investigations into the environmental problems associated with mining activity in the Upper Clark Fork area were conducted first by the Potentially Responsible Party (PRP) (Anaconda Minerals Company) from 1966 to 1982. EPA initiated the Remedial Investigation/Feasibility Study process in 1983. An Initial Remedial Investigation for the Silver Bow Creek site prior to inclusion of the Butte area was completed in 1987 (U.S. EPA 1991).

EPA established priorities to ensure the most serious problems were dealt with first (i.e., areas involving potential human health risks were given a higher priority than environmental risks). The four Operable Units at the Silver Bow Creek/Butte Area Superfund Site which are considered high priorities are: (1) Warm Springs Ponds; (2) Rocker Timber Plant; (3) Butte Mine Flooding; and (4) Butte Priority Soils. The Streamside Tailings Operable Unit is considered an intermediate priority (U.S. EPA 1991).

Ground water infiltration into underground mines and the Berkeley Pit could potentially contaminant the shallow ground water aquifer and surface water if the water in Berkeley Pit rises beyond 5,410 feet. As of February 27, 1990, the water level was 4,975 feet, and has not, therefore, reached the critical level. The contaminants of concern are arsenic, cadmium, lead, copper, zinc, iron, manganese, and sulfates (U.S. EPA 1991). Wells for domestic-water consumption are located in the vicinity of the Silver Bow Creek site and draw water from the shallow aquifer.

Surface water in Silver Bow Creek was sampled to determine levels of heavy metals and the results were reported in the 1987 Remedial Investigation. For the protection of aquatic life, the concentrations of total recoverable arsenic, cadmium, copper, lead, and zinc in surface water should not exceed specific criteria.

When these heavy metals were measured, concentrations did exceed the standards for protection of aquatic life in Silver Bow Creek (U.S. EPA 1991).

The West Camp/Travona underground mine-flooding discharges could contaminate Silver Bow Creek through direct discharge of ground water into Missoula Gulch, which joins Silver Bow Creek. When pumps for the West Camp mines were shut off in 1965, ground water began to flood basements in the residential areas south of the mine shafts. An intercept well was drilled in 1965. From 1965 to 1969, water flowed from this well into Missoula Gulch, and then, into Silver Bow Creek (U.S. EPA 1991).

Agricultural soils and crops were also affected by the mine wastes from the Silver Bow Creek site. Circumstantial evidence exists that approximately 5,400 acres of land have been contaminated by heavy metals to varying degrees, by using Silver Bow Creek or the Upper Clark Fork River water for irrigation (U.S. EPA 1991).

Fish and water fowl were also studied during the 1987 Phase I Remedial Investigation. There is evidence that fish, particularly Rainbow Trout, are receptors of heavy metals within the study area. However, it was found that arsenic concentrations in fish tissue were below U.S. Department of Agriculture (USDA) food standards (U.S. EPA 1991).

2. Milltown Reservoir - Part of the Clark Fork Superfund Sites

Operating History

The Milltown Reservoir Superfund Site is located in Milltown Valley, 5 miles east of Missoula, Montana. The Milltown dam was built in 1906 and 1907 below the confluence of the Clark Fork and Blackfoot Rivers to provide hydroelectric power. The Towns of Milltown and Bonner are the main population centers in the study area. The Milltown Reservoir Site is one of four Superfund Sites in the Clark Fork River Basin. The three other sites, located upstream of Milltown Reservoir along the Clark Fork River, are the Anaconda Smelter site, the Silver Bow Creek/Butte Area site, and the Montana Pole site.

Although mining, milling, and processing activities were never conducted at this site, the reservoir has accumulated large volumes of river-borne sediments from upstream mining areas of Anaconda and Butte. Sedimentation from mining-related activities has been determined to be the source of both surface- and ground water contamination in the area. Mining operations in the Clark River Basin began with the 1864 gold discovery in Butte. Mining wastes from these areas were discharged directly into tributaries of the Clark Fork River. These wastes, containing arsenic, cadmium, copper, iron, lead, and zinc, were added to the normal sediment load (U.S. EPA 1991).

The Milltown Reservoir is subject to considerable sediment accumulation from both the Clark Fork River and Blackfoot River watersheds. The Clark Fork and Blackfoot Rivers drain approximately 3,710 square miles and 2,290 square miles, respectively. In 1984, Woessner, et al., estimated that the reservoir contains 120 million cubic feet of sediment. Assuming a density of 1.8 grams per cubic centimeter (g/cc), this translates into approximately 6.5 million tons of sediment (U.S. EPA 1991).

Environmental Damages and Risks

Concern arose at the site in May 1981, when arsenic was found in four community supply wells at concentrations ranging from 0.54 to 0.90 mg/l. In August 1981, residents were advised not use water from these wells for potable purposes. In 1983, EPA and MDHES initiated a Remedial Investigation to determine the environmental characteristics and the type and extent of contamination in the Milltown area. Testing conducted during this investigation indicated that contamination appeared to be hydraulically confined to the uppermost aquifer in the present area (U.S. EPA 1991).

In 1983, vegetables from two gardens in Milltown were analyzed for arsenic. Tests by MDHES laboratory showed spinach to have an arsenic level of 2.66 ppm, lettuce had a level of 1.41 ppm, and two rhubarb plants had arsenic levels of 1.1 and 0.2 ppm, respectively. Levels of arsenic and copper in two plant species growing in the Milltown Reservoir were studied and were compared with levels of arsenic and copper in the same species located in the Blackfoot River (representative of background concentrations). The study found that arsenic and copper levels for both species were greater in the Reservoir samples, with the greatest levels occurring in the east section of the Reservoir (in the roots of the plants) (U.S. EPA 1991).

The Milltown Reservoir was placed on the NPL in 1983. The Remedial Investigation/Feasibility Study began in 1983 for the Water Supply Operable Unit. In 1984, an interim ROD described the two selected actions: abandonment of the existing ground water supply and replacement and relocation of water supply and transmission facilities. The actions were funded and completed in 1985. In 1985, a supplemental ROD described two additional measures: replacement of household water-supply equipment (as needed to reduce contamination) and on going sampling at residences (U.S. EPA 1991).

Additional studies were completed to determine if releases of hazardous substances, pollutants, or contaminants have occurred, or have the potential to occur, downstream from the Reservoir. ARCO had started working on the Remedial Investigation/Feasibility Study for the Milltown Reservoir/Sediments Operable Unit in 1990. The main objectives of the Feasibility Study include clean-up or control of the following: contaminated ground water; submerged contaminated reservoir sediments; and contaminated soils and exposed sediments (U.S. EPA 1991).

In addition, EPA will conduct an Endangerment Assessment to evaluate any present or future risks that the sediments pose for human health and the environment. Work groups are currently evaluating Risk Assessment work plans to assess the effects of contamination on public health, fisheries, and wetlands, and continued releases from the Reservoir (U.S. EPA 1991).

3. <u>Celtor Chemical Works, Humboldt County, California</u>

Operating History

The Celtor Chemical Works site covers 2.5 acres and is located in Humboldt County, California, in the Klamath mountain range. The site is located at the north end of the Hoopa Indian Reservation, several hundred feet from the Trinity River. There are approximately 900 residents within 3 miles of the site. Local residents use the area for agriculture, fishing, and grazing their domestic animals.

The Celtor Chemical Works mill began operation in 1958. Sulfide ores were mined at the Copper Bluff mine and shipped to the mill. Copper, zinc, and other precious metals were extracted at the Celtor mill. Tailings were then either stockpiled or (presumably) flushed down a gully to the Trinity River.

After the facility ceased operation in the early 1960s, an abandoned tailings pile washed into the Trinity River during a heavy flood in 1964. The State of California Department of Health Services also noted that other tailings may have caused acidic surface-water runoff in the area and high heavy metals concentrations in area soils (U.S. EPA 1991).

Environmental Damages and Risks

The site was proposed for industries on NPL on December 30, 1983, and the final Remedial Investigation found that the Celtor Chemical Works site poses a significant threat to human health and the environment due to elevated levels (in excess of CAM TTLCs, DWSs, and AWQCFALs) of arsenic, cadmium, copper, lead, and zinc in soil and surface-water samples.

Direct contact with contaminated water, especially through ingestion of more than 2 liters per day, could cause human health problems. Ingestion of contaminated soils is also thought to be a potential human health hazard. Contamination may be responsible for the defoliation of lands adjacent to the site due to runoff. The State of California issued citations to the Celtor Chemical Works due to fishkills that were most likely caused by runoff from the tailings located at the site. As a result of activities at the Celtor Mill, the area is no longer suitable for agriculture, and the nearby Trinity River can be used only for limited recreational fishing (U.S. EPA 1991).

4. <u>Torch Lake, Houghton County, Michigan</u>

Operating History

Torch Lake is located on the Keweenaw Peninsula of Upper Michigan. The surface area of Torch lake is 2,717 acres; it has a mean depth of 56 feet and a maximum depth of 115 feet. The area of the Torch Lake watershed is 77 square miles. Two small communities with populations of approximately 1,000 each (Linden and Hubbell) are located on the west side of Torch Lake.

For more than 100 years, Torch Lake was the center of Michigan's copper mining, smelting, and milling activities. Mining began in the 1860s in an elemental copper belt extending from the northern tip of Keweenaw Peninsula 100 miles to the southwest. Over 10.5 billion pounds of copper were beneficiated in the Torch Lake area, and an estimated 200 million tons of tailings were pumped into Torch Lake and surrounding properties between 1868 and 1968. The tailings reduced the lake's volume by 20 percent. Mining activities in the Torch Lake area peaked in the early 1900s (U.S. EPA 1991).

Beginning in 1916, technological innovations allowed for the recovery of copper from tailings previously discarded in Torch Lake. The submerged tailings were collected, screened, recrushed, and gravity-separated at one of three reclamation plants. These plants included Calumet and Hecla (opened in 1916), Tamarack (1925), and Quincy (1943). In the 1970s, copper recovery plants began operating in the Torch Lake area. The only discharge to Torch Lake from the copper recovery plants was noncontact cooling water. By 1986, only one small copper recovery plant was still operating.

Environmental Damages and Risks

By the 1970s, there was concern over the environmental health of Torch Lake because of the century of mining waste deposition into it. Copper ore tailings are present in and around Torch Lake and other areas of the Keweenaw Peninsula. The sources of contamination include tailings and associated debris and flotation chemical drums in the tailings, drums in Torch Lake, and industrial chemicals.

Torch Lake was listed on NPL in June 1988. In 1983, the Michigan Department of Public Health (MDPH) issued a fish consumption advisory on all sauger and walleye caught in Torch Lake. In a study of heavy metal concentrations in Torch Lake sediments and mining wastes, it was concluded that the water in Torch Lake is not directly contaminated with heavy metals, although the tailings are directly contaminated with arsenic, chromium, copper, lead, tin, and zinc. Furthermore, heavy metals may be entrained in wind currents, but they do not represent a serious human-health risk.

Over 96 percent of the copper input is from surface runoff, 3 percent is from precipitation, and 1 percent is from ground water inflow. Copper loss occurs by outflow into Portage Lake. The budget indicates an annual net loss of dissolved copper. However, copper concentrations have been relatively stable for the past 14 years. Therefore, precipitation, complexation, dissolution, absorption, and diffusion control dissolved copper concentrations.

In a 1988 health assessment for Torch Lake, the Agency for Toxic Substances and Disease Registry (ATSDR) concluded that the site is a potential public health concern because of possible exposure to unknown etiological agents that may create adverse health effects over time. Although Torch Lake is currently contaminated with mine tailings, there are no known health effects linked to this contamination (U.S. EPA 1991).

APPENDIX 1-D

304(l) SITE SUMMARIES RELATED TO COPPER MINING ACTIVITIES

1. Anaconda Minerals
2. Ferri Haggerty Mine
3. Kennecott-Utah Copper Division

1. Anaconda Minerals

The Anaconda Minerals Company operates a treatment system, including settling ponds, to collect nonpoint source runoff from tailings previously generated by an inactive copper smelter near Butte, Montana. One settling pond discharges to Silver Bow Creek. The discharge from this pond contains high concentrations of copper, zinc, and arsenic exceeding applicable water-quality standards. Additional discharges of contaminated runoff from the Anaconda site have occurred during precipitation events, when the volume of runoff exceeds the capacity of the treatment system. When capacity is exceeded, runoff is discharged directly into Silver Bow Creek (U.S. EPA 1990b).

Downstream of the Anaconda site, numerous other nonpoint source dischargers also contribute high metal loads. The contamination from the Anaconda site and the nonpoint discharges from other mine sites have caused extensive degradation of Silver Bow Creek and have led to its classification as an NPL site (U.S. EPA 1990b).

2. Ferri Haggerty Mine

The Ferri Haggerty Mine, an active copper mine, discharges mine water into surrounding water bodies. The discharges from the mine have caused exceedances of applicable water-quality standards for copper. Discharges from the site have been found to be toxic to aquatic life. To control the levels of copper in the discharge, the owner/operator has developed a three-step plan to ensure compliance. The plan includes passive ion exchange, flow management within the mine workings, and wetlands treatment (U.S. EPA 1990b).

3. Kennecott-Utah Copper Division

Kennecott-Utah Copper Division operates a copper mine near the Great Salt Lake. The facility discharges contaminated runoff from the tailings area to a ditch (known as C-7). In turn, the C-7 ditch discharges into the Great Salt Lake. The discharges have been shown to be toxic to aquatic life. According to the facility, this is primarily due to arsenic. Kennecott has spent over $10 million to reduce arsenic levels in its discharges.

APPENDIX 1-E

ACRONYM LIST

ACRONYM LIST

AAC	Arizona Administrative Code
AEQA	Arizona Environmental Quality Act
AMD	Acid Mine Drainage
amps/m	amps per meter
ANFO	Ammonium Nitrate and Fuel Oil
AOC	Area of Concern
APP	Aquifer Protection Permit
ARCO	Atlantic Richfield Corporation
ATSDR	Agency for Toxic Substances and Disease Registry
AWQCFAL	Ambient Water Quality Criteria for Freshwater Aquatic Life
AZMMR	Arizona Department of Mines and Mineral Resources
BADCT	Best Available Demonstrated Control Technology
BMP	Best Management Practices
CAM	California Assessment Method
CDC	Centers for Disease Control
CERCLA	Comprehensive Environmental Response, Compensation, and Liability Act
CFR	Code of Federal Regulations
cm/s	centimeters per second
cm	centimeter
CMP	Corrugated Metal Pipe
CWA	Clean Water Act
DEQ	Department of Environmental Quality
DWS	Drinking Water Standards
EP	Extraction Procedure
FIT	Field Investigation Team
ft/sec	feet per second
g/l	grams per liter
gpm	gallons per minute
gpm/ft^2	gallons per minute per square foot
GPR	Ground Penetrating Radar
HDPE	High-Density Polyethylene
ICS	Individual Control Strategy
IJC	International Joint Commission
in/ft	inches per foot
IPCC	In-pit Crusher and Conveyer
km^3	kilograms per cubic meter
l/m^2	liters per square meter
lbs	pounds
LC$_{50}$	Lethal Concentration (50% mortality)
LD$_{50}$	Lethal Dose (50% mortality)
m^3	cubic meter
MCL	Maximum Contaminant Level
MDNR	Michigan Department of Natural Resources
MDPH	Michigan Department of Public Health
mg/kg	milligrams per kilogram
mg/l	milligrams per liter
MGD	Million Gallons Per Day

ACRONYM LIST (Continued)

MIBC	methyl isobutyl carbinol
ml	milliliter
mm	millimeter
mph	miles per hour
MSHA	Mine Safety and Health Administration
MTU	Michigan Technical University
MWD	Mine Water Drainage
MWHP	Mine Water Holding Ponds
NIPDWS	National Interim Primary Drinking Water Standards
NOD	Notice of Disposal
NPDES	National Pollutant Discharge Elimination System
NPL	National Priorities List
NPSP	Nonpoint Source Discharge Permit
PCB	Polychlorinated Biphenyl
PD	Phelps Dodge Corporation
PDWS	Primary Drinking Water Standards
PLS	Pregnant Leach Solution
ppm	parts per million
PVC	Polyvinyl Chloride
RAP	Remedial Action Plan
RCRA	Resource Conservation and Recovery Act
RI/FS	Remedial Investigation/Feasibility Study
RI	Remedial Investigation
ROD	Record of Decision
RPM	Remedial Project Manager
SAIC	Science Applications International Corporation
SIC	Standard Industrial Classification
SMCL	Secondary Maximum Contaminant Level
SME	Society of Mining Engineers
STEL	Short Term Exposure Limit
SX/EW	Solvent Extraction/Electrowinning
TAT	Technical Assistance Team
TCE	Trichloroethylene (Trichloroethene)
TCLP	Toxicity Characteristic Leachate Procedure
TDS	Total Dissolved Solids
tpd	tons per day
tph	tons per hour
tpy	tons per year
TSP	Total Suspended Particulates
TTLC	Total Threshold Limit Concentration
TWA	Time Weighted Average
USBM	U.S. Bureau of Mines
USC	United States Code
U.S. EPA	U.S. Environmental Protection Agency
USGS	U.S. Geological Survey
	micrograms per kilogram
	micrograms per liter
^{3}	micrograms per cubic meter
	micrometer

MINE SITE VISIT:

ASARCO/TROY MINE

U.S. Environmental Protection Agency
Office of Solid Waste
401 M Street SW
Washington, DC 20460

2.0 MINE SITE VISIT: ASARCO/TROY MINE

2.1 INTRODUCTION

2.1.1 Background

EPA has initiated several information gathering activities to characterize mining wastes and waste management practices. As part of these ongoing efforts, EPA is gathering data by conducting visits to mine sites to study waste generation and management practices. As one of several site visits, EPA visited ASARCO's Troy Mine near Troy, Montana, on July 23 and 24, 1991.

Sites to be visited were selected to represent both an array of mining industry sectors and different regional geographies. All site visits have been conducted pursuant to RCRA Sections 3001 and 3007 information collection authorities. When sites have been on Federal land, EPA has invited representatives of the land management agencies (Forest Service and/or Bureau of Land Management) to participate. State agency representatives and EPA regional personnel have also been invited to participate in each site visit.

For each site, EPA has collected waste generation and management information using a three-step approach: (1) contacting the facility by telephone to obtain initial information, (2) contacting State regulatory agencies by telephone to get further information, and (3) conducting the actual site visit. Information collected prior to the site visit is then reviewed and confirmed at the site.

The site visit reports describe mine processes, mine waste generation and management practices, and regulatory status on a site-specific basis and are based on information gathered from State and Federal agency files as well as on observations made during the site visit. The site visit report on ASARCO's Troy Mine was prepared from a variety of information sources including the ASARCO files from the Montana Department of State Lands (MDSL) and the Montana Department of Health and Environmental Sciences (MDHES), information collected during telephone conversations, the site visit, and other published information. The following individuals participated in the ASARCO/Troy Mine site visit on July 23 and 24, 1991:

ASARCO/Troy Mine	
Doug Miller, Mine Manager	(406) 295-5882
Larry Erickson, Senior Mine Engineer	(406) 295-5882
U.S. Environmental Protection Agency	
Stephen Hoffman, Chief, Mine Waste Section	(703) 308-8413
Van Housman, Chemical Engineer	(703) 308-8419
Jeff Bryan, EPA Region VIII	(406) 449-5486
Orville Kiehn, EPA Region VIII, Mining Engineer	(303) 294-7535

Science Applications International Corporation

Ingrid Rosencrantz, Environmental Scientist	(703) 734-2508
Laurie Lamb, Geologist	(303) 292-2074

U.S. Forest Service

Norm Yogerst, Soil Scientist	(406) 329-3634
Mark Mason	(406) 295-4693
Richard Stearns	(406) 293-6211

Montana Department of State Lands

Sandi Olsen, Director, Division of Reclamation	(406) 444-2074

Participants in the site visit were provided opportunity to comment on a draft of this report. Comments were submitted by ASARCO and EPA Region VIII. ASARCO's comments are presented in Appendix 2-A. EPA's responses to ASARCO's comments are presented in Appendix 2-B.

2.1.2 General Facility Description

ASARCO, Inc. (ASARCO) mines copper/silver ore from the underground Troy mine located in the Cabinet Mountains in the northwestern corner of Montana (see Figure 2-1

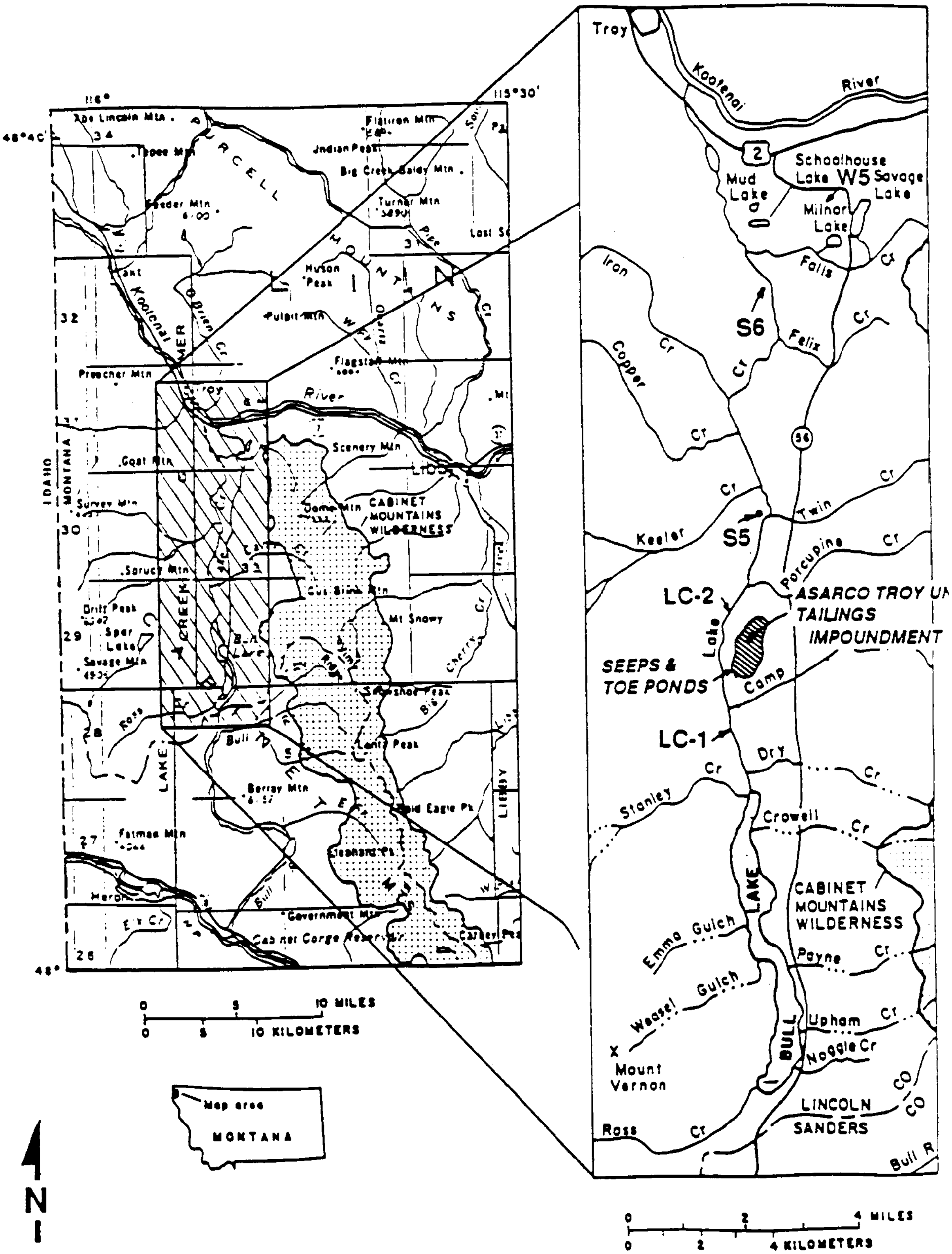


Figure 2-1. General Project Location Map

(Source: Levings et al., 1984)

). The mine is located in Lincoln County, approximately 15 miles south of Troy, Montana, the closest town with a variety of public services. Libby, Montana, is located approximately 20 miles northeast of the mine. The mine access and haulage road turnoff is located on the west side of Highway 56 approximately 12 miles south of Highway 2. The mine is seven miles from the mine access and haulage road turnoff.

The relatively flat lying stratabound ore body lies under Mt. Vernon at an elevation of approximately 4,400 ft. above sea level. Mine portals and facility buildings are perched on the steep northern slopes of Mt. Vernon (see Figure 2-2

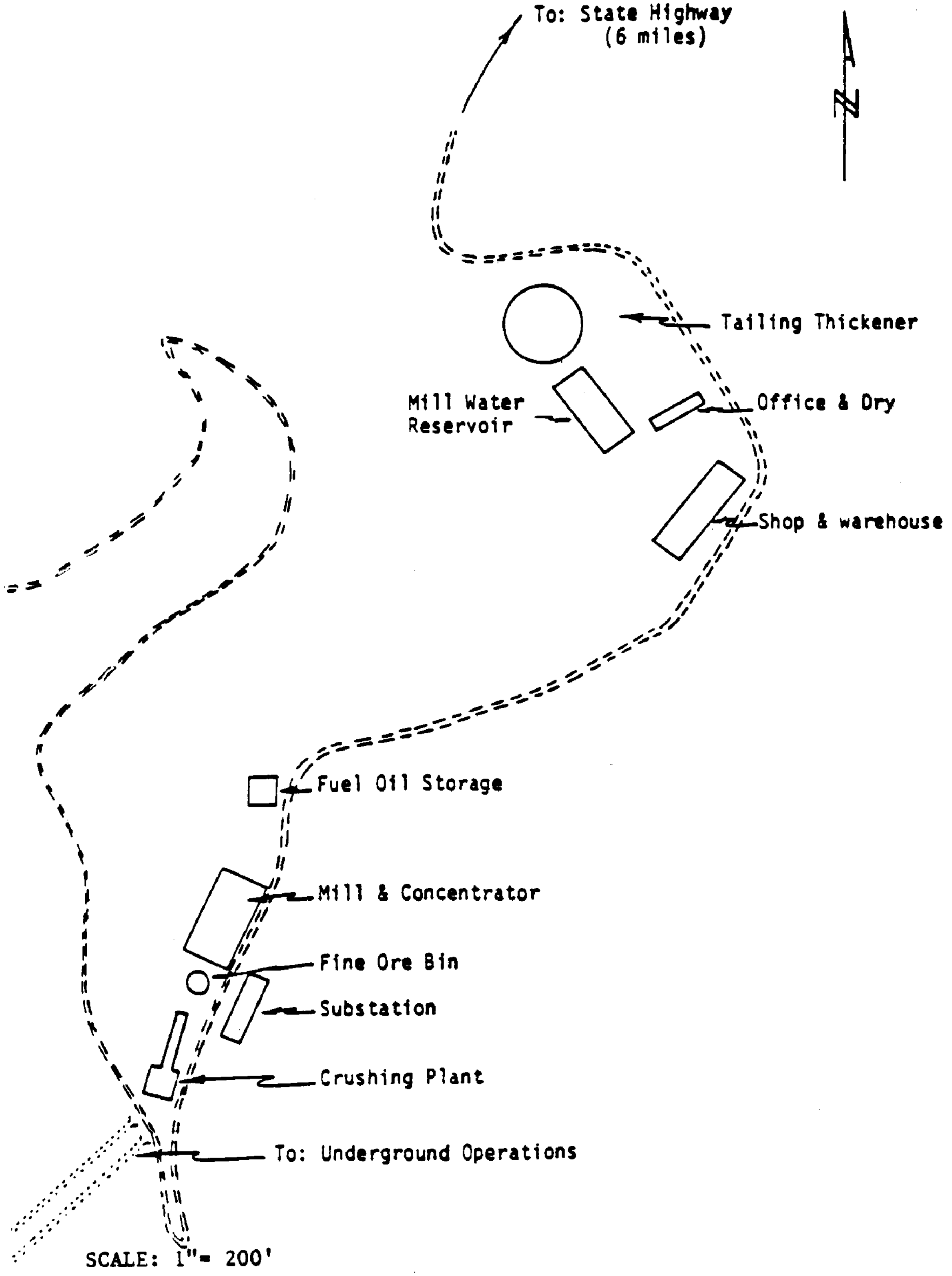


Figure 2-2. Facility Location Map, ASARCO Troy Mine

(Source: From 1979 MDHES Air Quality Bureau Construction Permit Application) according to ASARCO, the fuel storage tank is actually located south of the shop rather than as shown in Figure 2-2). Vehicle access to the underground room and pillar workings is through two main adits. Extracted ore is crushed underground and conveyed to the surface mill for secondary crushing, grinding, and concentration of the silver-bearing and non-silver bearing copper minerals. Tailings generated through beneficiation of the ore are disposed of in a tailings impoundment located approximately six miles north of the mill in Lake Creek Valley. Waste rock generated during the active phase of mining is returned to the mine or used as road or base for facility construction. The copper/silver concentrate is shipped out of State for processing.

The active facility is located on approximately 350 acres of United States Forest Service (USFS) and privately owned lands. The USFS lands are part of the Kootenai National Forest. According to the mine manager, the mine is located on patented claims, the mill is located on unpatented claims, and the tailings impoundment is located on land owned by Kennecott Minerals and leased to ASARCO. The mine holds unpatented and patented claims on an additional 2,400 acres of Forest Service land that are currently inactive or reserved for potential future uses. (MDSL and USFS, 1978)

2.1.3 Environmental Setting

Being located in northwestern Montana, the mine is usually under the influence of the Pacific maritime air mass providing a moist, temperate climate. However, cold arctic air masses occasionally spill over the Continental Divide, contributing to the extreme variation in temperatures on a seasonal basis. Temperature at Troy ranges from -25

vary greatly depending upon the elevation. The Draft Environmental Impact Statement (DEIS) for the Troy Project states that the annual precipitation on Mt. Vernon is over 100 inches, while the area surrounding the tailings impoundment receives approximately 35 inches of precipitation per year. Sixty-five percent of the precipitation occurs in the form of snow between November and March. In 1978, gross evaporation from planned reservoirs and evaporation ponds at the site was estimated at 28-30 inches of water per year. The primary wind direction is westerly; however, during the winter an occasional cold arctic air mass results in strong northeasterly and easterly winds. In the fall and winter, Lake Creek Valley frequently experiences low level inversions during periods of high pressure that tend to trap air pollutants in the valley until a front moves through the area. (MDSL and USFS, 1978)

2.1.3.1 Surface Water

The mine complex lies in portions of both the Stanley Creek and Lake Creek drainages. Stanley Creek drains the northern slope of Mt. Vernon, captures the flow of Fairway Creek downstream of the mill, and empties into the northern end of Bull Lake. Lake Creek drains from the northern end of Bull Lake, acquiring flow from several tributary streams before flowing into the Kootenai River 18 miles north of Bull Lake (see Figure 2-1). The Draft Environmental Impact Statement describes a marshy area along the southeastern border of the proposed tailings impoundment; however, a separate section of the document states that no wetlands exist in the mine study area. (MDSL and USFS, 1978)

2.1.3.2 Geology

Faulting, glaciation and the erosive force of water have influenced the topography and geology of the area around the Troy mine. The Lake Creek-Bull River Valley is a down-dropped block bordered by two major north-south trending faults. The deep valley was in-filled primarily with fine-grained glacio-lacustrine and glacio-fluvial sediments during Pleistocene glaciation. (MDSL and USFS, 1978)

The Cabinet Mountains are composed of partially metamorphosed, folded, and faulted argillites, quartzites, and carbonates of the Precambrian Belt Series, intruded by Jurassic and Cretaceous age granitic plutons. These well indurated sedimentary rocks have very low primary porosity and permeability although fractures and faults within these units can allow the movement of water. The reviewed documents (see References) briefly discuss several faults in and around the mine area; however, extensive information on fracture patterns and faulting for the site was not available. (MDSL and USFS, 1978)

Mining is occurring beneath Mount Vernon in the upper section of the Precambrian Revett quartzite in disseminated mineralization consisting of the copper minerals chalcocite, bornite, and chalcopyrite and small amounts of copper oxides and native silver. The ore body is approximately 7,400 feet long, 1,800 feet wide, and 60 feet thick. It trends north 5 degrees west and dips 0 to 15 degrees to the southwest. The eastern boundary of the ore zone is cut by a fault striking north-northwest. The ore body is cut by an east-west reverse fault dipping 70
and the upper south block. (MDSL and USFS, 1978)

Geophysical investigations and borings indicate that the thick Lake Valley deposits overlying bedrock include coarse materials (sands, gravels, and cobbles) interlayered with the thick layers of silts and fine sandy silts deposited by glaciation. In addition, Lake Creek has deposited clays and silty alluvial materials in its floodplain. The sediments of Lake Creek Valley have been estimated to be over 400 feet thick. (MDSL and USFS, 1978)

The DEIS states that the mine is located in Zone 2 of seismic risk on the seismic risk map of the United States. Zone 2 indicates areas where moderate seismic damage should be expected as determined through historic records. (MDSL and USFS, 1978)

2.1.3.3 Hydrogeology

The reviewed literature does not define specific aquifers, but discusses ground water in relation to the lithology in which it is found. Ground water occurs in joints, fractures, and faults of the Belt Series bedrock. These features control the movement of ground water and yields to wells completed in bedrock. Alluvial deposits of Recent age are present in the Lake Creek floodplain. These deposits may yield small to moderate quantities of water. In addition to rainfall and snowmelt infiltration, the alluvial aquifer may be recharged by Lake Creek. In the summer, ground water provides baseflow to Lake Creek. (MDSL and USFS, 1978)

No detailed hydrological work was conducted in the mine area prior to mine construction and development. Small quantities of ground water were encountered in colluvium and landslide debris during drilling at one of the proposed adits. Rapid infiltration of surface water into the Stanley Creek channel sediments suggests significant ground-water supplies in the Stanley Creek alluvium. The facility maintains two mill supply water wells in lower Stanley Creek between the mine administration building and the tailings impoundment. Each well is approximately 300 to 400 feet deep. Facility representatives were unable to provide information about the yields of the wells or the volume of water provided to the mill by these wells. Three domestic water wells are also present in lower Stanley Creek. Another well, located 400 yards from the site, supplies potable water to the facility. According to ASARCO, the domestic wells are completed in bedrock, but the lithology within which the other wells are completed remains unidentified. Hydrogeological investigations in the area of the tailings impoundment have been largely unsuccessful in determining the hydrogeology of that area. (MDSL and USFS, 1978)

Most of the reports conclude that the hydrogeology of the tailings area is "complex" and that additional detailed hydrogeological investigations would produce a better understanding of the site hydrogeology. Prior to construction, ASARCO's contractors conducted an investigation of the tailings disposal area to provide information for use in the design of the unit. Two deep test wells and 12 test holes were drilled with a total of 25 permeability tests conducted in 11 of the holes (the type of permeability tests were not clear). The DEIS stated that no hydraulic testing of the aquifer was performed due to the presence of fine-grained materials. The two deep test wells were completed on the east side of the proposed impoundment. Test well 1 was completed at 283 feet below ground surface in a coarse-grained zone. The upper 167 feet of the well encountered fine-grained material. When the screened interval was placed in the fine-grained zone, the well collapsed and filled with sediment to the 190 feet level. The DEIS reported water levels in this well of 75.3 and 62.9 feet below the casing top. The 62.9 feet level was interpreted as below the level of Lake Creek at the time of measurement. The other test well was drilled through silts and sands before encountering coarse-grained materials at 165 feet. Test holes completed along the proposed tailings dam axis found medium dense silts, sandy silts, and silty sands down to 65 feet deep. In the southern part of the tailings impoundment site, medium dense to dense gravelly sand, sandy gravel, and cobbly gravel was encountered. Water levels in these test holes ranged from 9 to 65 feet below ground surface four days after drilling. The permeabilities calculated by the contractor for the materials below the proposed tailings impoundment range from 8 to 8100 ft. per year. The DEIS cautioned that the calculated figures could vary as much as two orders of magnitude from actual conditions. (MDSL and USFS, 1978; and ASARCO, 1990)

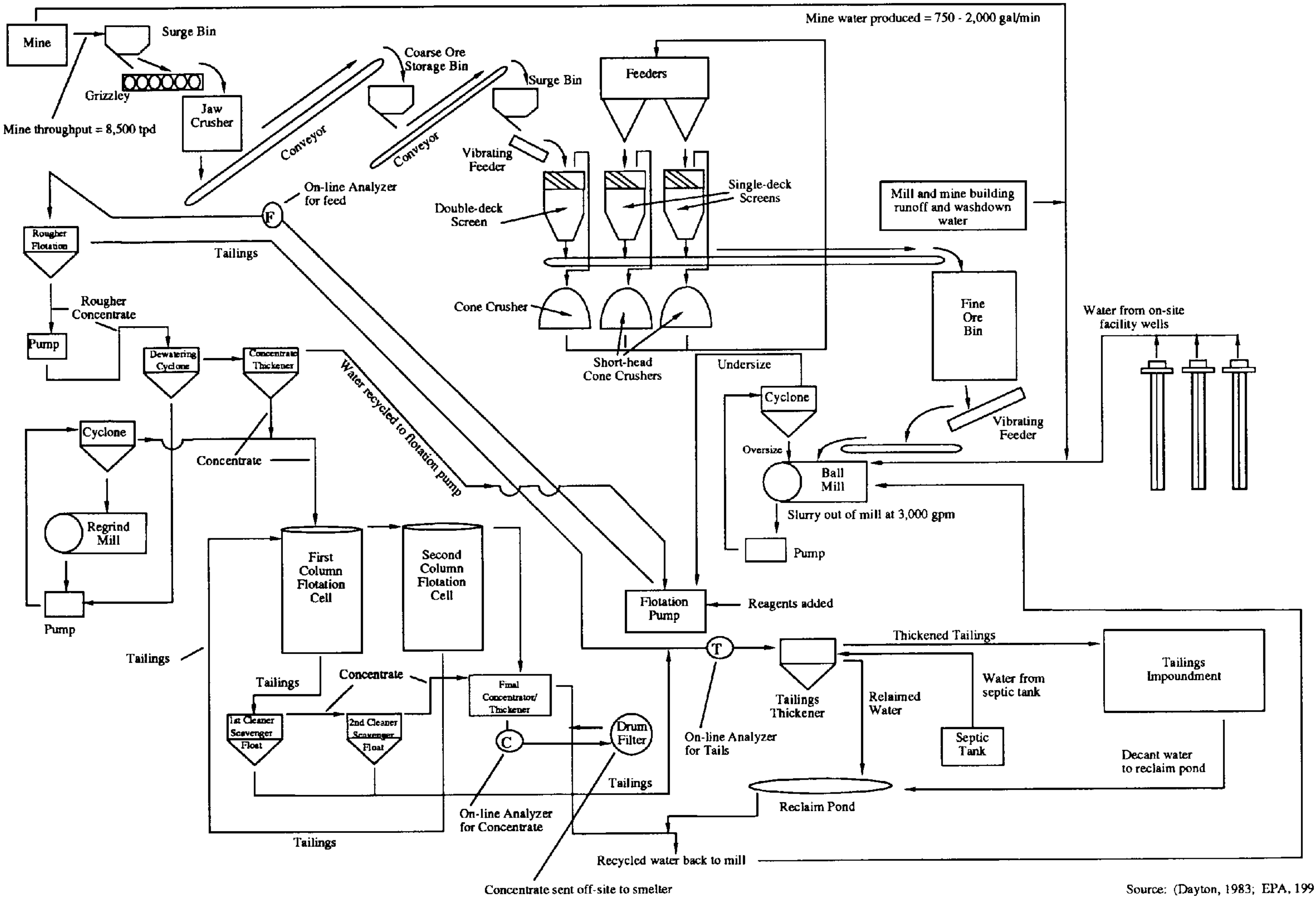
In addition, according to ASARCO, four ground water test wells are located on the west side of the impoundment. No information was available concerning the completion, date of installation, or monitoring of these wells.

The DEIS stated that a coherent picture of the piezometric surface was not provided by the data collected at that time. Low permeability, fine-grained materials were considered to be the reason for the inconclusive testing. The DEIS assumed, for the purposes of that document, that ground water flows toward Lake Creek as suggested by the geophysical survey, water-level data, and the general hydrogeologic setting of the pond. The original hydrogeological investigation was not available for review and inclusion in this report and the available documents did not provide information concerning aquifer properties such as hydraulic conductivity, gradient, transmissivity, and storativity. (MDSL and USFS, 1978)

2.2 FACILITY OPERATIONS

The Troy copper/silver ore body was delineated in 1964 as a result of an exploratory drilling program. Development and construction of the mine began in 1979, with full production following in 1982. At the Troy mine, room and pillar methods are used to mine ore from the underground workings. Ore is hauled by trucks to the underground crusher and then conveyed to the mill. Milling unit operations include crushing, grinding, concentration by flotation, regrinding coarse concentrate fractions, and dewatering concentrate.

The flotation operation used at the mill is a physical separation and concentration operation that allows the valuable minerals to float to the top and over the edge of the flotation cell in a froth. Chemicals are added to the slurry entering the flotation cells to facilitate the separation process. The product of the flotation cell, the froth, is dewatered in a concentrate thickener and further dewatered on a drum filter. The waste product of flotation, gangue minerals known as tailings, flow from the bottom of the flotation cell to a tailings thickener for dewatering. Water from the concentrate thickener, filter drum, and tailings thickener are recycled back to the ball mill. A control room in the grinding and concentrating mill building monitors and controls the operations. The final product of flotation and filtration is a concentrate of silver and copper minerals. Figure 2-3



Source: (Dayton, 1983; EPA, 1991)

illustrates the mining and milling operations, from the input of raw materials to the output of products and wastes.

Approximately 350 acres are disturbed by facility buildings, roads, and operations. The mine employs approximately 320 people and operates 24 hours a day seven days a week. Mine and mill employees work 12-hour shifts. Ore is extracted from the mine at a rate of 8,500 tons per day, with yearly production at approximately 3 million tons per year. Seventy-five percent of the 64 million ton ore reserve is recoverable and 25 percent will remain in the supporting pillars. The ore averages 0.76 percent copper and 1.58 ounces of silver per ton. Approximately 80 tons of ore are required to extract one ton of concentrate. The final concentrate, averaging approximately 100 ounces of silver per ton (0.3 percent) and 45 percent copper, is trucked to Troy where it is loaded into railcars for shipment to a smelter. The mine life is estimated to be 15 years from the start of full production in 1982. According to the mine manager, the current reserves should last another 6 to 7 years at the current rate of production. Exploration to extend the reserves is ongoing. (Dayton, 1983; and EPA, 1991.) A detailed explanation of the mining and milling operations, including the products and wastes generated by each state of the operation, is discussed below.

2.2.1 Mining Operations

The underground mining operations are accessed by vehicles through two main adits, the service adit and the main haulage adit. These two adits and the south and conveyor adits function as air intake adits. The EPA team entered and exited the mine via the service adit. Two additional adits (the east and west adits) function as air exhaust adits. Rock removed during adit excavation was used as fill to construct patios for the surface buildings. (Dayton, 1983; and EPA, 1991)

Figure 2-3. ASARCO Troy Flow Diagram

(Source: Dayton, 1983 and EPA, 1991)

The competent stratabound deposit is mined using headings and benching in a room-and-pillar pattern. Top slice headings are driven 20 feet high by 45 feet wide using mobile electro-hydraulic rigs to drill horizontal rounds. The lower 40 foot benches are drilled vertically using electro-hydraulic rigs. The 45 feet deep holes are loaded with an emulsified slurry (ANFO). After blasting, Caterpillar 988 and 980 loaders muck the newly blasted faces and load the ore into diesel-powered, articulated haulers. The ore is transported to a surge bin at the underground primary crusher. (Dayton, 1983)

Mine water collected from areas throughout the mine flows to two sumps in the mine. Water is pumped from the sumps through a secondary pumping station to the surface mill for use in the grinding and flotation circuits. Flow rate into the sumps ranges from 750 to 2000 gallons per minute (gpm) depending upon the season. In the event of excess water, the surplus is pumped directly to the tailings thickener. Although the DEIS suggested that an oil/water separator or skimmer was to be used at the sumps, according to the mine manager, none existed at the time of the site visit. (MDSL and USFS, 1978)

Ore is fed from the underground ore bin along a vibratory feeder to the Allis-Chalmers 48 x 60 inch, 250 HP jaw crusher. The ore is crushed to minus 6 inches before it drops into the coarse ore storage bin below the primary crusher. A conveyor transports ore from the coarse ore storage bin through the 4,400 foot long adit decline to the surface mill. The EPA team viewed the primary ore crusher on Wednesday July 24, 1991, when it was shut down for weekly maintenance. At the time of EPA's visit, water was in use for cleaning the primary crusher. Particulate emissions from the jaw crusher are controlled by a baghouse.

There are several maintenance and chemical storage (oil, gas, lubricates, etc.) facilities located underground. The EPA team visited an equipment maintenance area. Tools, welding tanks, and spare parts were observed in this area. A 10,000 gallon diesel fuel tank is stored in a topslice heading down grade from the underground maintenance facility. Used oils generated underground are stored in surface tanks. Drums were observed as the EPA team toured the mine.

2.2.2 Milling Operations

Primary crushed ore is transported on a conveyor to the surface secondary crusher building. One Symons standard cone crusher and two Symons short head crushers perform secondary crushing. According to the mill superintendent, throughput of ore is 600 tons/hour. Ore is initially fed to the Symons standard cone crusher along vibrating feeders to a vibrating double deck screen. Fines falling through the double deck screen (minus 1/2 inch) are transported by belt conveyor to the fine ore bin. Oversized material is fed to the cone crusher for size reduction. The product of the cone crusher is transferred to single deck screens (with slots minus 1/2 or 5/8 inch by 3 inches) for sizing. Material falling through the single deck screen is conveyed to a fine ore surge bin. Coarse materials remaining on the screen are fed into the short head cone crushers. As crushed ore exits the short head crushers it is conveyed back to the single-deck screens for a repeat of the sizing and crushing operations. The moisture content of coarse ore entering the secondary crushing stage is 1.8 percent. No water is added to the ore during the secondary crushing stage. (Dayton,

1983) The secondary crushers were down for maintenance during EPA's Tuesday, July 23, 1991, visit to the mill area.

The secondary crushing building is equipped with a wet scrubber to treat emissions resulting from the transfer points in the crushing circuit. A hooded vacuum system pulls the dust up into the scrubber. Slurry produced in the scrubber is pumped to the ball mill discharge sump and joins the ball mill discharge for additional communition circuit processing. (EPA, 1991)

Fine ore is conveyed from the fine ore surge bins to a transfer tower to facilitate transport to the fine ore bin. The transfer tower is equipped with a baghouse to control emissions. The collected dust is returned to the belt and transported to the fine ore bin. The mill superintendent stated that the bags are disposed of "at the end of their life."

The fine ore bin receives materials crushed and sized to minus 3/4 inch. Six vibrating feeders transports fine ore from the bin to a conveyor system that feeds two KVS ball mills. Forty percent of the ball mill volume is filled with steel balls of a 3 1/2 inch top size. The balls are added continuously during operation of the mills due to steel abrasion of approximately 2 pounds of steel balls per ton of ore. Ball mill makeup water is supplied by a variety of sources depending on seasonal availability. These sources include mine water, tailings decant water, tailings thickener overflow, secondary crushing wet scrubber slurry, mill patio area runoff, and well water. The mill superintendent could not provide an estimate of the water flow rate entering each ball mill. ASARCO's 1978 plan of operations estimated that 4,313 GPM of process water would be required in the milling circuit (the amount added in the ball mills is not specified). The mill manager stated that approximately 3,000 gpm of ground ore slurry leaves the ball mills and is pumped to a cyclone for sizing. The 35-40 percent pulp density overflow from the cyclone reports to the three rougher flotation cells and the underflow is recycled back to the ball mills for further grinding.

The mill control room is located between the grinding circuit and the flotation circuit. An on-stream analyzer provides continuous metals content analysis of the grinding circuit feed, concentrate and tailings streams. During the mill tour, drums of reagents were observed in storage beneath the concentration circuit.

Potassium amyl xanthate, a collector, and an 80/20 mixture of pine oil and Dowfroth 250 (a frothing agent) are added to the cyclone overflow before it enters the flotation circuit. Collectors coat the target minerals and make them hydrophobic. Frothers stabilize air bubbles in the flotation unit, allowing the target minerals to cling to the bubbles and rise to the top of the flotation unit. The 80/20 frother mixture is added to the slurry at a rate of 0.02 pound per ton of feed and the xanthate at 0.03 pound per ton of feed.

The chemically fortified ore slurry is pumped to one of three banks of seven 500 Wemco 144 flotation cells. Air is injected from the bottom of the cell into the slurry. The ore mineral particles attach to the air bubbles and rise to the surface. A bubbling froth is created at the surface of the cell as the minerals float to the top of the cell. This surface froth of air bubbles, water and ore minerals overflows the cell into a collection launder.

The froth from the first three cells of each bank is collected and pumped to the first column cell. The froth from the remaining four cells of each bank is collected and pumped to cyclones for size classification.

Ore froth overflow (fine material) from the cyclone flows to a middlings thickener. The cyclone underflow (coarse material) flows to the regrind ball mill. The valueless gangue minerals (e.g., quartz) settle and flow out the bottom of the flotation cells to the tailings thickener.

The first column flotation cell is used to "clean" (remove unvaluable minerals) the froth from the first three Wemco cells. The froth from the first column cell reports to the final concentrate thickener. The tailing or underflow of the first column cell is pumped to a cyclone for size classification.

The thickened middlings concentrate is sent to two banks of first cleaner flotation cells. Concentrate overflow from the first cleaner cells is sent directly to the second column cell. The tailings are sent to the cleaner scavenger cells. Second column cell concentrate overflow goes directly to the concentrate thickener and the tailings report back to the first cleaner flotation.

The cleaner scavenger froth concentrate is pumped to a cyclone for size classification. The cleaner scavenger tailing flows to the tailings thickener.

A 75-foot diameter final concentrate thickener dewaters the concentrate with the aid of an anionic flocculent added at the rate of 0.015 lbs./ton of feed. The thickened concentrate is further dewatered on a drum filter and then stored in a bin; the water is recycled to the ball mills. The final moisture content of the concentrate averages 12%. The final concentrate averages 45% copper and 100 ounces of silver per ton. Approximately 80 tons of ore are needed to produce one ton of concentrate.

There is an equipment and vehicle maintenance and reconditioning building located north and downhill of the mill buildings. The EPA team did not visit this area. There is also an onsite laboratory that the EPA team did not visit.

2.3 WASTE MANAGEMENT

2.3.1 Types of Waste

Wastes produced at the ASARCO/Troy mine are generated as a direct result of mining and milling of ore. Wastes generated at the Troy mine include mill tailings, tailings pipeline pigging waste, and waste rock. Ground water seeping into the Troy underground mine workings becomes mine drainage; however, according to the mine manager, it is currently used in the mill process or bypasses the mill process and flows with the mill tailings to the tailings thickener. Mine water has never been discharged to surface water or ground water.

Mill tailings are the largest volume solid waste generated at the Troy mill and disposed of in the tailings impoundment. Tailings pipeline pigging waste is produced infrequently. Exact volumes of pigging waste were not provided by the facility. Waste rock (non-ore bearing rock removed to access the ore) is currently removed and disposed of in the underground workings. During development and construction of the mine complex, the removed waste rock was used as fill to create a surface area large enough to accommodate the mill and associated buildings.

Several wastes are generated at the mine that are not uniquely associated with mining and milling operations. According to the mine manager, waste halogenated solvents, used in the cleaning of parts and equipment, are the primary hazardous wastes produced at the facility. A few PCB-containing transformers or capacitors are still in use in the mine. Wet chemistry lab wastes are produced on site from the analysis of mine and mill products (ore, concentrate and tailings) and are placed in the flotation circuit for concentration. Waste hydraulic and motor oil, considered nonhazardous by Montana State law, are produced from their use in mine vehicles and equipment. The waste oils are collected by a waste oil recycler and transported offsite. The mine also generates garbage and sewage/sanitary wastewaters. Garbage is transported to a local landfill and sanitary waste water is disposed of in the tailings impoundment via the tailings thickener.

2.3.2 Solid Waste Management Units

2.3.2.1 Tailings Thickener

Tailings leaving the flotation circuit flow to a 250-feet diameter tailings thickener, located downhill and north of the mill buildings (see Figure 2-2). According to the mine manager, the tailings thickener also receives wastewater from the septic system and any excess drainage from the mine water supply system that is not required for milling operations. The thickener settles tailings to a specific density using a cationic-anionic flocculent at a rate of 0.01 pounds per ton of feed. Overflow water drains to the partially fenced, lined (1/16-inch Hypalon), tailings thickener reclaim pond. The reclaim water is used as needed in the mill circuit. According to ASARCO, in the event of a reclaim pond surplus, an overflow box located on the east side of the reclaim pond would convey the excess water to the tailings impoundment via the tailings pipeline. North and downhill of the tailings thickener and thickener reclaim pond is an unlined basin used as the emergency overflow pond. According to the mill superintendent there are pumps available to remove water from the emergency overflow basin. None were noted by the EPA team. The mine manager stated that the pond is used infrequently; however, the last use of the emergency dump pond was within the two weeks prior to EPA's visit. All three units are located on the cut and fill slope constructed during mine development.

The mine manager stated that the quality of the thickener reclaim water is not monitored and there are no ground-water monitoring wells in the area. The surface-water-quality and benthic macroinvertebrate populations of Stanley Creek, located below the thickener and the two ponds, are monitored downstream of the plant site. No releases from these units have been documented in the State files reviewed by EPA or in conversations with facility personnel.

2.3.2.2 Tailings Pipeline

Two eight-inch schedule 40 steel pipes convey, by gravity flow, thickened tailings seven miles to the tailings impoundment. According to the mine manager, the pipe interiors are coated with urethane. Gravity flow of tailings through the pipeline ranges from 7 to 12 feet/second, depending upon tonnage. No drop boxes are used on the pipeline.

Tailings pipeline inspections are no longer conducted on a formal basis by the mine staff. However, according to the mine manager the pipeline's primarily above ground location along the mine access road allows for daily observation by the mine maintenance staff. According to the mine manager, a break in the pipe would be detected immediately in the mill control system due to the loss of pipeline pressure. It is unclear if the facility has a pressure loss detection system at the end of the pipe which would be necessary to signal a leak in a pipeline under gravity flow. The pipeline required pigging after six years of operation and then once a year for the past two or three years. Pigging wastes were disposed of in the tailings impoundment.

MDSL and MDHES documents indicate that the pipeline has released tailings into the environment in two separate events. In July 1981, tailings from a break in the tailings pipeline indirectly entered Stanley Creek when they leaked from the emergency storage impoundment, where they had been diverted. In June 1984, a tailings pipeline break released tailings into Lake Creek.

2.3.2.3 Tailings Impoundment

The 320 acre tailings impoundment is used for the management of the estimated 2,930,000 tons of mill tailings produced each year. It is located in the Lake Creek Valley, on a terrace east of Lake Creek and is dammed on three sides; north, west, and south. The eastern boundary of the tailings impoundment is located just west of a bedrock hill, which will eventually mark the eastern boundary of the tailings impoundment. Four toe ponds (two north toe ponds, toe pond number two, and south toe pond) are located along the toe of the west dam, as shown in Figures 2-4 and 2-5 (locations NTP, TP2, STP on Figure 2-4 and as labeled on Figure 2-5—it should be noted that Figure 2-5 is not drawn to scale; the toe ponds are shown greatly enlarged for illustrative purposes).

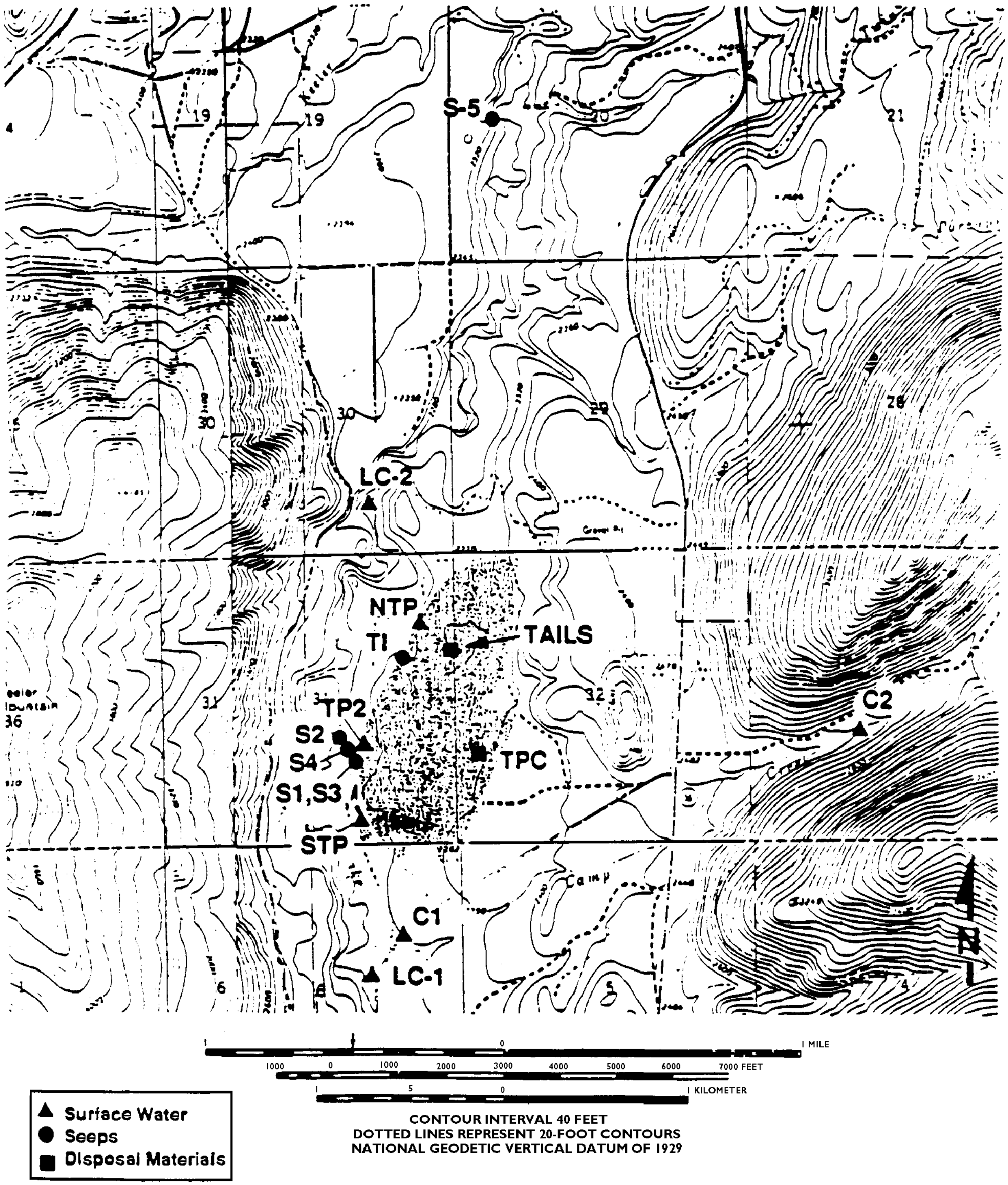


Figure 2-4. Location Map: Sampling Stations
Base: USGS 7.5' Quadrangles: Crowell Mountain and Spar Lake 1963 (PR 1923)

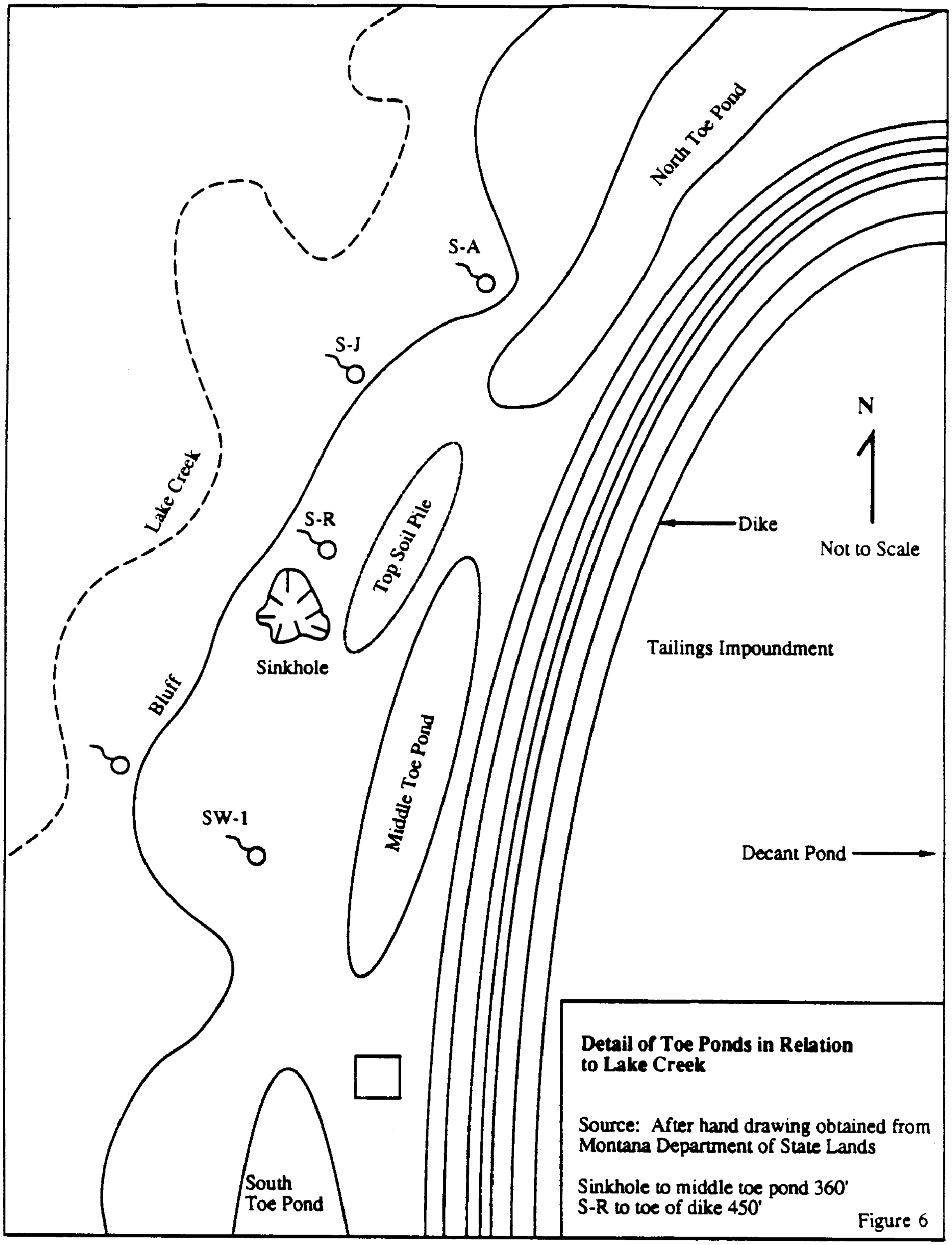


Figure 2-5. Detail of Toe Ponds in Relation to Lake Creek

According to the mine manager, tailings impoundment construction included compacting native materials on the bottom of the tailings impoundment and the placement of slimes along the perimeter dike before tailings were deposited in the pond. The tailings impoundment is composed of three quadrants (term used by mine manager) separated by east-west dividing dikes. Quadrant 1 is the south section of the pond, quadrant 2 is the middle section, and quadrant three is the northern most section. The mine alternates tailings disposal among the three quadrants. At the time of the site visit, the mine was spigoting tailings along the perimeter of quadrant 3. Perimeter dams and interior quadrant dividing dikes are heightened during a quadrant's period of inactivity when tailings are deposited in a different quadrant. Materials for these dam heightening operations are obtained from the borrow pits dug east of the pond's current eastern perimeter. The maximum current height of the tailings dam is 70 feet.

Tailings impoundment design allows for the addition of approximately 40 feet in height before design capacity is reached. ASARCO estimates that the remaining reserves will require 20 feet in additional height over the next 7 to 10 years.

Quadrants 1 and 2 were seeded with barley in the spring to prevent wind erosion of the tailings. A sprinkler system uses tailings water to irrigate the barley and prevent wind erosion. Large areas of quadrants 1 and 2 were barren. The mine manager stated that several of the large barren areas were not seeded due to saturated conditions at the time of planting. The southeastern section of the pond was developed over forested land, evidenced by the tops of trees extending through the tailings.

A tailings water decant pond is located along the eastern pond boundary, adjacent to quadrant 2. The decant pond was slimed on the bottom and sides, but the bottom was not compacted. It stores water reclaimed from the tailings impoundment for reuse in the mill process or for irrigation and dust control on the tailings impoundment. A pump barge is used to pump the reclaimed tailings water to the lined reclaim pond or to a sprinkler system. EPA observed the decant pond barge pumping tailings water to the sprinkler system. The reclaim pond is located approximately 100 yards southwest of the tailings impoundment. At the time of EPA's site visit, the lined reclaim pond contained aquatic plants and tadpoles.

Two borrow pits, excavated in what appeared to be medium to coarse grained material, lie between the bedrock hill and the current eastern boundary of the tailings impoundment. EPA observed one of the borrow pits filled with water to approximately the same level as the adjacent decant pond.

According to the mine manager, the water is collected surface-water drainage and, due to the placement of slimes along the pond perimeter during the construction of the decant pond, is not presumed by the company to be the result of tailings water movement through subsurface materials.

In 1982, not long after the tailings impoundment began to receive tailings, water emerged west of the tailings dam, apparently as a result of seepage through the west tailings impoundment dam. An earthen dike was constructed in 1983 to contain the seepage in three toe ponds. According to a February 10, 1983, letter from Dames and Moore, ASARCO's contractor, the dike would allow for the collection and pumpback of the water to the tailings impoundment. The contractor cautioned that although the ponded water would not significantly affect the overall stability of the tailings impoundment starter dam, it could result in localized sloughing of the terrace slopes adjacent to Lake Creek downgradient of the starter dam. According to the mine manager, the water in the toe ponds is tested quarterly and the water is pumped back to the tailings impoundment for use in the sprinkler system. (MDSL, 1983)

In January 1991, MDSL conducted an inspection at the ASARCO tailings facility to investigate a report that a sinkhole had formed downgradient of the west tailings dam and toe ponds, near the edge of the terrace adjacent to Lake Creek. At the time of the State inspection, the hole was estimated to be 15 feet by 12 feet and 10 feet deep and water was heard draining through the hole. The origin of the sinkhole was attributed to water "piping" below ground surface. MDHES stated that water-quality data from a seep in the terrace slope, assumed to be directly associated with the sinkhole, indicates a component of tailings water. (ASARCO states that the original design engineer did not believe the sinkhole was the result of piping from the tailings impoundment and that springs and seeps existed in the area prior to construction.) MDHES suggested that the only immediate danger to the environment is from increased sedimentation to Lake Creek. Worst case scenarios would be mass failure of the terrace bank and head cutting into the sinkhole area. During EPA's visit to the sinkhole, the fine-grained materials along the walls of the hole appeared to be saturated, but no water flowed in the hole. (MDHES, 1991)

Downhill (downgradient) from the toe ponds is a topsoil stockpile. Below and both north and south of the topsoil pile are several springs located at approximately the slope break. Approximately six feet downhill from one of the seeps is a sump which collects water to be pumped to the tailings impoundment. The pump was operating at the time of EPA's visit. The EPA team observed iron staining along the surface flow path of the seep water. According to the mine manager, the seeps have been sampled numerous times and a report on the water quality of the seep indicated that the seeps were not tailings water. The MDSL sampled the seeps on two separate occasions and concluded that the water was of good quality, but that there appeared to be a correlation between several of the seeps and the north toe pond water. According to the mine manager similar seeps are found on the other side of Lake Creek, both upstream and downstream of the tailings impoundment. (MDSL, 1991; and ASARCO, 1990)

2.3.2.4 Waste Rock

During adit excavation and mine development, development waste rock was used as fill for the construction of the patio for the mill buildings, maintenance buildings, and general mine office buildings. Although the patio area is not considered a waste unit by the facility, impacted surface water has been associated with the fill materials. Therefore, for the purposes of this report, it will be discussed in this section. Waste rock is currently excavated (exact amounts unknown) and is backfilled into the mine.

A Stanley Creek monitoring point is located at least two miles downstream of the patio area. From 1985 to 1987, extensive sampling was conducted of Stanley Creek surface water and sediment below the patio fill in response to elevated copper concentrations detected in Stanley Creek surface water. The cause of these elevated copper concentrations was determined to be due to erosion of the waste rock used as fill in the patio upon which the mill was constructed. The scope of the entire corrective action was not documented in the reviewed documents, but ASARCO apparently took some type of erosion control measures. (MDHES, 1990)

2.3.2.5 Waste Oil Storage Tanks

Waste oils are stored onsite in aboveground storage tanks. The volume of these tanks was not stated in the available references. Moore Oil in Libby, Montana picks up the oil approximately once a month. EPA viewed used oil shipping records during the visit to the mine. Used oil shipments are not manifested in Montana because State laws do not consider waste oil to be a hazardous waste.

2.3.2.6 Burn Pit

Mine personnel stated that plant domestic garbage is disposed of at a landfill in Libby. Reference was made to a burn pit used for general waste in a 1987 Montana Department of Health and Environmental Sciences - Air Quality Bureau inspection report. However, according to ASARCO, wood products are no longer burned. (MDHES, 1987)

2.3.3 Waste Water Management Units

2.3.3.1 Mine Water

According to facility representatives, the mine has no current discharges to surface water. Mine adit water is used in the milling operation. According to the mine manager, if mine water discharge volumes exceed the needs of the mill circuit, the water is sent directly to the tailings thickener. As noted previously, the mine manager stated that the tailings are discharged from the thickener to the pipeline at a specific density in order to maintain proper performance of the pipeline. Consequently, it is unclear how large volumes of excess mine water may affect the performance of the tailings thickener and the density of the discharged tailings slurry. The facility does not monitor the quality of the mine water.

2.3.3.2 Sanitary Sewage

The facility maintains a septic tank system to treat sewage and sanitary waste. Treated septic tank effluent flows to the tailings thickener for reuse in the mill circuit or disposal in the tailings impoundment. The septic tank sludge has been removed once by a contractor. There is no specific monitoring of the septic system.

2.3.3.3 Assay Laboratory Wastes

According to ASARCO, dilute acids and other wastes from the wet chemistry laboratory are disposed of in lab sinks that drain to the flotation cells.

2.3.4 Hazardous Waste Management Unit

2.3.4.1 Waste Solvent Tanks

Waste halogenated solvents are stored on site and picked up every two weeks by Safety Kleen. These tanks were not visited by the EPA team. The location of the waste solvent storage tank(s) is not available in the reviewed information.

2.4 MONITORING

During the period prior to mine development and up to the time of the site visit, ASARCO has sampled surface water, ground water and air in the areas likely to be impacted by the Troy mining activities. As presented in ASARCO/Troy's mining operating permit, MDSL requires the mine to sample and analyze surface-water-quality. Since the permit was issued, MDSL has modified the monitoring requirements as needed. Ground-water monitoring was required at one time and tailings water monitoring has become a requirement. In addition, monitoring requirements for surface water have changed as monitoring results over time indicated a need for new analytical procedures, new monitoring locations, biological monitoring, termination of monitoring due to inconclusive or spurious results, and other difficulties. ASARCO conducts air monitoring as required by the Air Quality Bureau - Montana Department of Health and Environmental Sciences. The MDSL and MDHES also sample and analyze environmental media to check analyses reported by ASARCO and in response to citizen complaints of alleged violations.

2.4.1 Surface Water

The earliest surface-water-quality monitoring results discussed in MDSL documents provided to EPA are from Lake Creek in 1974. This early baseline surface-water monitoring conducted by ASARCO was complemented with monitoring in the Stanley Creek drainage in 1977, both above and below the mill site. The United States Geological Survey (USGS) and the Montana Bureau of Mines and Geology (MBMG) have both sampled Lake Creek below the tailings dam; the USGS monitored from 1976 to 1978 and the MBMG monitored from 1979 to 1982. EPA did not have access to the raw data collected by ASARCO (prior to 1985), MBMG or the USGS.

Water-quality monitoring required by the MDSL since 1985 includes surface-water and aquatic benthic macroinvertebrate sampling of three streams that could potentially be impacted by releases from the mine: Stanley Creek, Fairway Creek and Lake Creek. Five locations are sampled during the spring, summer, and fall quarters each year (see Figure 2-6

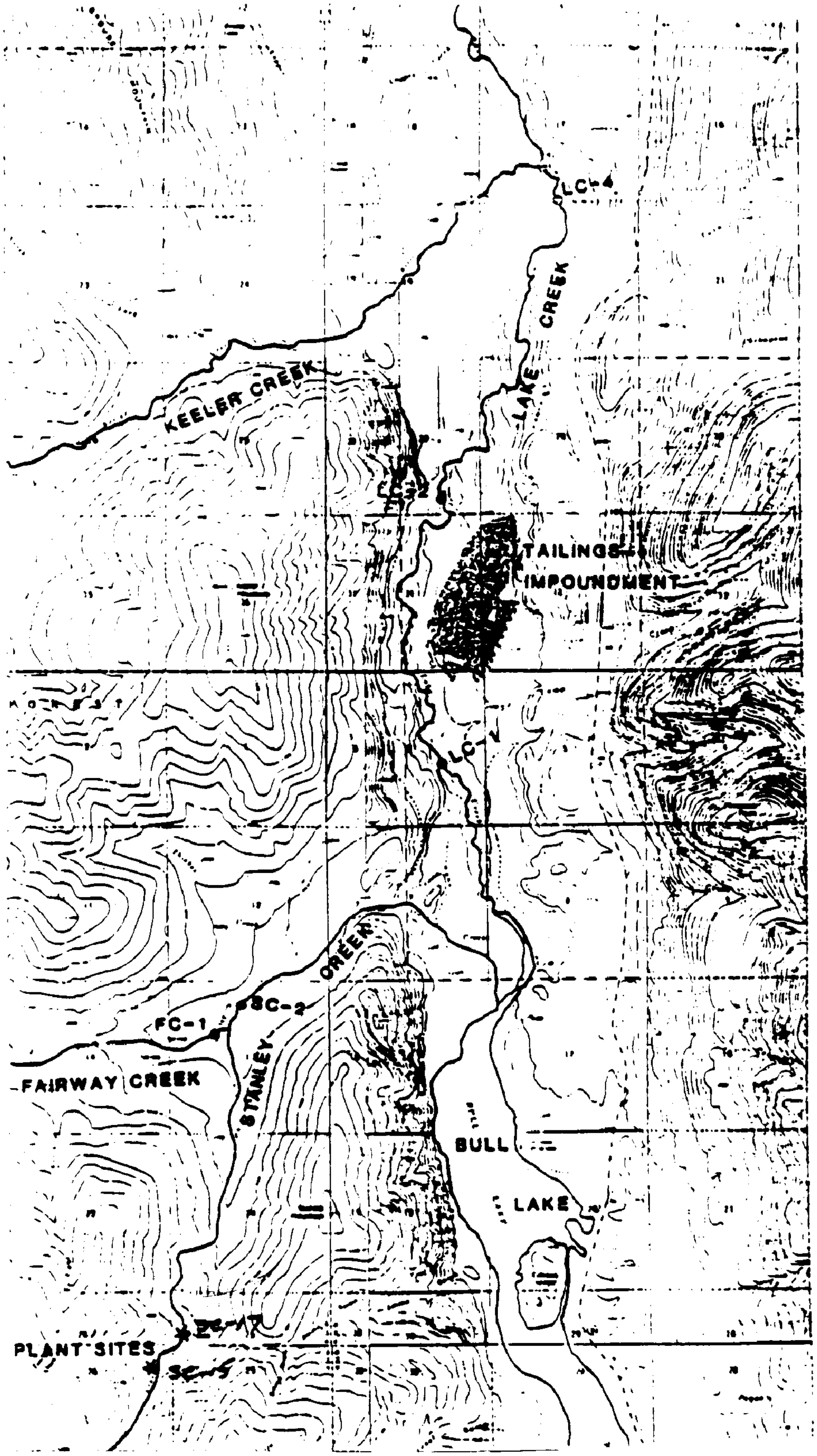


Figure 2-6. Sampling Locations (e.g., FC-1): ASARCO Troy Unit Macroinvertebrate Monitoring Program

). MDSL also requires the sampling and analysis of tailings water during each of the three quarters. Three sample locations are upstream of the tailings impoundment and two are downstream of the impoundment (see Figure 2-5). Two sample sites on Stanley Creek have been and continue to be sampled by the facility, the MDSL and the MDHES, but not as a monitoring requirement. (MDSL, MDHES, and ASARCO, 1987)

ASARCO is required to submit quarterly and yearly reports of the monitoring results. ASARCO also submits an interpretation of the results using statistical models, although this analysis is not required by the MDSL. The statistical models applied by ASARCO determine significant variations between study locations, seasons, and years. In addition, ASARCO determines statistical correlations between the macroinvertebrate and key water-quality data. (MDSL et al., 1987)

The most recent MDSL report (1990 draft) performed basic statistics on all the ASARCO/Troy water-quality data available in the files of the MDSL and MDHES through 1988. The results of these analyses indicate that there is no statistical difference in the quality of the water at LC-1, LC-2, and LC-4 due to the operation of the mine and tailings impoundment. (MDSL, 1990)

2.4.2 Ground Water

According to a MDSL review of ASARCO Troy monitoring data, five ground-water wells were monitored from September 1983 to December 1984. The program was discontinued in 1984 due to several reasons. For example, the background well consistently produced water that exhibited toxicity to aquatic life (i.e., Daphnia). In addition, there were unresolved questions about the well completions, particularly the materials used. Finally, the MDSL determined that the ground-water hydrology under the tailings impoundment was too complex and the four ground-water wells surrounding the impoundment were not adequate to monitor a 400-acre area. The ground-water monitoring data for the period of September 1983 to December 1984 were not provided to EPA, and thus was not included in this report.

Although the facility is not currently required to conduct formal ground-water monitoring at the tailings impoundment, EPA noted several piezometers drilled into the west dam. According to the facility manager, these piezometers were installed for yearly stability analysis of the tailings dam and have not been used to monitor ground-water quality.

2.4.3 Air

The Air Quality Bureau of the MDHES requires ASARCO/Troy to maintain one-high volume six-day air sampler to measure total suspended particulates. Quarterly sampling reports are submitted to the State. According to the facility, the most recent monitoring report submitted to the State (at the time of the site visit) was for the winter quarter of 1990.

2.5 REGULATORY REQUIREMENTS AND COMPLIANCE

2.5.1 Operating Permit

The MDSL issues Mining Operating permits as required by the Montana Metal Mine Reclamation Act, and under the regulations promulgated in the Montana Hard Rock Mining Requirements. The operating permit is not issued until the Montana Environmental Policy Act/National Environmental Policy Act (MEPA/NEPA) process is complete [i.e., an Environmental Assessment (EA) or Environmental Impact Statement (EIS) has been completed] and the State makes a decision. When Federal lands are involved, the applicant must also obtain approval from the appropriate Federal agency (MDSL, 1990).

On February 18, 1974, ASARCO submitted application number 00128 to the MDSL for an operating permit. Numerous additional reports and data were submitted prior to the permit approval. The Montana Board of Land Commissioners approved ASARCO's $2,752,000 Reclamation Bond on August 19, 1978 following MDSL's approval of ASARCO's operating plan. The MDSL issued Operating Permit Number 00093 on November 27, 1978. The permit required ASARCO to establish and/or continue a satisfactory water-quality monitoring program for Lake Creek (Montana Board of Land Commissioners, 1978; and MDSL, 1978).

The location of ASARCO/Troy on USFS lands necessitated the involvement of the Forest Service in the operating permit approval process. In 1978, The MDSL and the USFS jointly produced an EIS for the Troy mine. The USFS concurred with the operating plan (as included in the Draft EIS and slightly amended in the Final EIS) and issued a Forest Service Record of Decision (MDHES and USFS, undated).

The MDSL has approved three operating permit amendments for the mine and ASARCO/Troy has applied for a fourth amendment to date. USFS involvement in amendment approval was not found in the available file documents. Amendment No. 1, approved on July 19, 1979, incorporated the construction of the Stanley Creek tailings impoundment, the staging area, and the aggregate storage area. According to the mine manager, Amendment No. 2 was for the construction of the emergency pond below the tailings thickener. Amendment No. 2 was not available for review. Amendment No. 3, approved on July 14, 1983, incorporated the construction of a secondary dike below the tailings dam toe and installation of a permanent pumping station to maintain the water level of water ponding behind the secondary dike to at least 10 feet below the dike crest. According to the Mine Manager, the pending fourth amendment application requests approval for the excavation of an 18 x 18 foot ventilation adit to be driven from within the ore body to the southwest side of the mountain (MDSL, 1979, 1983).

2.5.2 Solid Waste

Mine waste is not regulated as a solid waste in Montana. The only solid waste managed on site is domestic garbage, used oil, and sewage sludge. The facility ships this waste offsite and is not subject to the State solid waste permitting requirements.

2.5.3 Surface Water

A Montana Pollutant Discharge Elimination System (MPDES) permit is required for all point source discharges to State surface waters, under authority of the Federal Clean Water Act. Montana operates a Federally delegated program. The MPDES permit is required regardless of whether an operating permit was issued by MSDL. The administrating office is the Water Quality Bureau (WQB) of the MDHES, which issues these permits and enforces the State surface-water discharge regulations. In addition, in accordance with the issuance of a full-scale mining operation permit, MDSL requires monitoring of surface-water and ground-water quality (MDSL, 1990).

The 1978 DEIS stated that discharges to surface water would be required to meet the State's water-quality nondegradation standard contained in the Montana Laws Regarding Water Pollution [Title 69. Chapter 48, 69-4801.2(1)(c)(iii)]. The nondegradation standard requires waters of higher quality than the State's water-quality standards to be maintained at the stream's higher quality. However, according to the MDSL, the Troy mine is exempt from this standard because it was permitted prior to 1984, and therefore, must only meet the State's water-quality standards (MDSL and USFS, 1978). In 1978, ASARCO/Troy Mine applied for a MPDES permit to discharge mine adit water from a settling sump.

2.5.4 Ground Water

Facilities which discharge directly to ground water, such as through the operation of a percolation pond, must obtain a Montana Groundwater Pollution Control System (MGWPCS) permit issued by the WQB of the MDHES. This permit is not required if MDSL issues a full-scale mining operating permit. This exemption exists because MDSL reviews and monitors the proposed operation at the same level as the WQB would review a MGWPCS permit application (MDSL, 1990). ASARCO is exempted from the ground-water permit requirement and the MDSL does not require ASARCO/Troy to conduct ground-water monitoring at any of the facility areas under the facility's operating permit.

2.5.5 Air

The MDHES, Air Quality Bureau (AQB) issues permits and regulates emissions from projects expected to exceed certain threshold values for various parameters. Generally an air permit is required if emissions of any pollutant, including fugitive dust, exceed 25 tons/year.

ASARCO/Troy applied for an Air Quality permit in December 1978 for the construction and operation of the secondary crushing plant. The application proposed the use of 5 baghouses to control particulate emissions from the secondary crushing plant, specifically, to address the ore transfer points in the plant (MDHES, 1978). The AQB issued permit No. 1316 to ASARCO/Troy on May 16, 1979 with a conditional approval (MDHES, 1979). One condition to be met by ASARCO was the installation and operation of at least one high-volume sampler in the vicinity of the mill site. The permit was modified in 1982 when the baghouse at the secondary crusher building was replaced with a high efficiency wet scrubber (ASARCO, 1978; and MDHES, 1979).

2.5.6 Hazardous Waste

The Montana Department of Health and Environmental Sciences issues identification numbers to generators of hazardous wastes. The ASARCO/Troy Mine is assigned the hazardous waste generator identification number MT13096199989. The facility transports its hazardous wastes (e.g., safety-kleen solvents) offsite. There was no information in the files reviewed concerning releases of hazardous wastes. According to the mine manager there have been no releases of any waste type (small spills of reagents, fuels, solvents, etc. excepted).

2.6 REFERENCES

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APPENDIX 2-A

ASARCO COMMENTS ON DRAFT SITE VISIT REPORT

[ASARCO comments on the draft site visit report were submitted in a letter dated January 14, 1992. This letter is not reproduced in this electronic version of this document. Copies of the comment document may be received from U.S. EPA, Office of Solid Waste, Special Waste Branch.]

APPENDIX 2-B

EPA RESPONSES TO ASARCO COMMENTS ON DRAFT SITE VISIT REPORT

A copy of the draft *Mine Site Visit: ASARCO/Troy Mine* was provided to ASARCO for their review. EPA addressed the comments submitted by ASARCO on January 14, 1992 (see Appendix 2-A) in the revised report as described below.

Comment 1. The text on page 2 was changed to read: "...and concentration of the silver-bearing and non-silver bearing copper minerals." The text on page 6 now reads: "...the copper minerals chalcocite, bornite, and chalcopyrite and small amounts of copper oxides and native silver."

Comment 2. The location of the fuel storage tanks on the sketch map (Figure 2-2 in the report) was not changed since the map was taken from the 1979 MDHES Air Quality Bureau Construction Permit Application. However, the text now includes a parenthetical note: "According to ASARCO, the fuel storage tank is actually located south of the shop rather than as shown in Figure 2-2."

Comment 3. The sentence has been changed to read: "According to the mine manger, the mine is located on patented claims, the mill is located on unpatented claims, and the tailings impoundment is located on land owned by Kennecott Minerals and leased to ASARCO."

Comment 4. EPA found that using the term "pond" for the tailings disposal area (rather than "impoundment" could lead to confusion with the various toe ponds on the site. Accordingly, the term "impoundment" was used when referring to the tailings disposal area and "pond" for the toe ponds. (Note that ASARCO used the term impoundment throughout its comments when referring to the tailings disposal area.)

Comment 5. The text was changed to read: "The facility maintains two mill supply water wells in lower Stanley Creek between the mine administration building and the tailings impoundment. Each well is approximately 300 to 400 feet deep. The facility representatives were unable to provide information about the yields of the wells or the volume of water provided to the mill by these wells. Three domestic water wells or the volume of water provided to the mill by these wells. Three domestic water wells are also present in lower Stanley Creek. Another well, located 400 yards from the site, supplies potable water to the facility. Accordingly to ASARCO, the domestic wells are completed in bedrock, but the lithology within which the other wells are completed remains unidentified."

Comment 6. The text was changed to read: "According to ASARCO, the domestic wells are competed in bedrock, but the lithology within which the other wells are completed remains unidentified."

Comment 7. The text now reads: "...according to ASARCO, four ground water test wells are located on the west side of the impoundment. No information was available concerning the completion, date of installation, or monitoring of these wells."

Comment 8. The reference to chemical separation has been deleted.

Comment 9. The text now reads: "The underground mining operations are accessed by vehicles through two main adits, the service adit and the main haulage adit. These two adits and the south and conveyor adits function as air intake adits. The EPA team entered and exited the mine via the service adit. Two additional aids (the east and west adits) function as air exhaust adits. Rock removed during adit excavation was used as fill to construct patios for the surface buildings."

Comment 10. ASARCO was subsequently offered the opportunity to review a more legible copy of this figure. Unfortunately, they were not able to provide comments prior to the time the report had to be revised.

Comment 11. The text has been changed to read: "Fines falling through the double deck screen (minus 1/2 inch) are transported by belt conveyor to the fine ore bin. Oversized material is fed to the cone crusher for size reduction. The product of the cone crusher is transferred to single deck screens (with slots minus 1/2 or 5/8 inch by 3 inches) for sizing. Material falling through the single deck screen is conveyed to a fine ore surge bin. Coarse materials remaining on the screen are fed into the short head cone crushers."

Comment 12. The reference to Wagner ST8 machines has been removed and the use of Caterpillar 980 loaders is noted.

Comment 13. The reference to a grizzley has been deleted.

Comment 14. The text has been changed to read: "A 10,000 gallon diesel fuel tank is stored in a topslice heading down grade from the underground maintenance facility."

Comment 15. ASARCO commented that the drum storage rooms noted by EPA as having been observed during the mine tour are unknown to them. EPA did not tour the fuel storage areas or the room seen during the mine tour, so it is not known if the drums were of "antifreeze or grease," as ASARCO suggested in their comment, or whether they were stored in the fuel storage area, as also was suggested. The revised report continues to note that "drums were observed during the mine tour."

Comment 16. The text now reads: "Material falling through the single deck screen is conveyed to a fine ore surge bin."

Comment 17. The text has been changed to read: "The collected dust is returned to the belt and transported to the fine ore bin. The mill superintendent stated that the bags are disposed of "at the end of their life."

Comment 18. The text in these paragraphs now reads:

"Potassium amyl xanthate, a collector, and an 80/20 mixture of pine oil and Dowfroth 250 (a frothing agent) are added to the cyclone overflow before it enters the flotation circuit. Collectors coat the target minerals and make them hydrophobic. Frothers stabilize air bubbles in the flotation unit, allowing the target minerals to cling to the bubbles and rise to the top of the flotation unit. The 80/20 frother mixture is added to the slurry at a rate of 0.02 pound per ton of feed and the xanthate at 0.03 pound per ton of feed.

"The chemically fortified ore slurry is pumped to one of three banks of seven 500 Wemco 144 flotation cells. Air is injected from the bottom of the cell into the slurry. The ore mineral particles attach to the air bubbles and rise to the surface. A bubbling froth is created at the surface of the cell as the minerals float to the top of the cell. This surface froth of air bubbles, water and ore minerals overflows the cell into a collection launder. The froth from the first three cells of each bank is collected and pumped to the first column cell. The froth from the remaining four cells of each bank is collected and pumped to cyclones for size classification.

"Ore froth overflow (fine material) from the cyclone flows to a middlings thickener. The cyclone underflow (coarse material) flows to the regrind ball mill. The valueless gangue minerals (e.g., quartz) settle and flow out the bottom of the flotation cells to the tailings thickener.

"The first column flotation cell is used to "clean" (remove unvaluable minerals) the froth from the first three Wemco cells. The froth from the first column cell reports to the final concentrate thickener. The tailing or underflow of the first column cell is pumped to a cyclone for size classification.

"The thickened middlings concentrate is sent to two banks of first cleaner flotation cells. Concentrate overflow from the first cleaner cells is sent directly to the second column cell. The tailings are sent to the cleaner scavenger cells. Second column cell concentrate overflow goes directly to the concentrate thickener and the tailings report back to the first cleaner flotation.

"The cleaner scavenger froth concentrate is pumped to a cyclone for size classification. The cleaner scavenger tailing flows to the tailings thickener."

Comment 19. The sentence has been changed to refer only to <u>water</u> removal from the emergency overflow basin.

Comment 20. The text has been changed to read: "Four toe ponds (two north toe ponds, toe pond number two, and south toe pond) are located along the toe of the west dam,..."

Comment 21. The purpose of the map is to show the relationship between various elements in the tailings disposal area. The legend of the map notes that it is "Not to Scale." In addition, the text now includes a note that emphasizes that the map is not drawn to scale.

Comment 22. The text now reads: "Tailings impoundment design allows for the addition of approximately 40 feet in height before design capacity is reached. ASARCO estimates that the remaining reserves will require 20 feet in additional height over the next 7 to 10 years."

Comment 23. The text now notes that "...according to ASARCO, wood products are no longer burned."

Comment 24. The text has been changed to read: "According to ASARCO, dilute acids and other wastes from the wet chemistry laboratory are disposed of in lab sinks that drain to the flotation cells."

Comment 25. The report referred to by ASARCO was compiled by the Montana Department of State Lands and was furnished to EPA by the Department.

Comment 26. The text now reads: "The well water is currently sampled monthly for coliform and a chlorinator is available for use if needed."

Comment 27. ASARCO's position has been noted in the report.

Comment 28. ASARCO's position has been noted in the "Tailing Impoundment" section (the "Conclusions" section, which contained the wording on which ASARCO commented, has been deleted).

MINE SITE VISIT:

SAN MANUEL FACILITY
MAGMA COPPER COMPANY

U.S. Environmental Protection Agency
Office of Solid Waste
401 M Street SW
Washington, DC 20460

3.0 MINE SITE VISIT: SAN MANUEL FACILITY
MAGMA COPPER COMPANY

3.1 INTRODUCTION

3.1.1 Background

The U.S. Environmental Protection Agency (EPA) has initiated several information gathering activities to characterize mining wastes and mining waste management practices. As part of these ongoing efforts, EPA is gathering data related to waste generation and management practices by conducting visits to mine sites. As one of several site visits, EPA visited Magma Copper Company's San Manuel Facility in San Manuel, Arizona on May 5 and 6, 1992.

Sites to be visited were selected by EPA to represent both an array of mining industry sectors and different regional geographies. All site visits have been conducted pursuant to RCRA Sections 3001 and 3007 information collection authorities. When sites have been on Federal land, EPA has invited representatives of the land management agencies (Forest Service and Bureau of Land Management). State agency representatives and EPA Regional personnel have also been invited to participate in each site visit.

For each site, EPA has collected information using a three-step approach: (1) contacting the facility by telephone to obtain initial information, (2) contacting state regulatory agencies by telephone to get further information, and (3) conducting the actual site visit. Information collected prior to the visit is then reviewed during the site visit.

The site visit reports describe mine activity, mine waste generation and management practices, and regulatory status on a site-specific basis. These reports principally discuss extraction and beneficiation operations, although a brief discussion of processing operations is also included. In preparing this report, EPA collected information from a variety of sources including Magma Copper Company and the Arizona Department of Environmental Quality. The following individuals participated in the San Manuel Facility site visit on May 5 and 6, 1992:

<u>Magma Copper Company</u>

Eldon D. Helmer, Director of Environmental Affairs	(602) 575-5600
Dale Deming, Manager of Environmental Services	(602) 385-3540
Norm Greenwald, Consultant, Norm Greenwald Associates	(602) 795-0471

EPA

Van Housman, Chemical Engineer, Office of Solid Waste (OSW)	(703) 308-8419
Patty Whiting, Environmental Protection Specialist, Office of Solid Waste (OSW)	(703) 308-8421
Haile Marion, Chemical Engineer, Office of Solid Waste (OSW)	(703) 308-8439
Lisa Jones, Chemical Engineer, Office of Solid Waste (OSW)	(703) 308-8451

Science Applications International Corporation

Ingrid Rosencrantz, Environmental Scientist	(703) 734-2508
Susan McCarter, Environmental Analyst	(703) 734-3187

Participants in the site visit were provided opportunity to comment on a draft of this report. Comments were submitted by the Arizona Department of Environmental Quality. Comments and EPA's responses are presented in Appendix 3-A.

3.1.2 General Description

Magma Copper Company operates the San Manuel mine, mill, and smelter facilities located north of Tucson, Arizona, in Pinal County. The facility encompasses approximately 12,000 acres of patented land with operations that extract, beneficiate, and process both sulfide and oxide ore to recover copper and molybdenum (See Figure

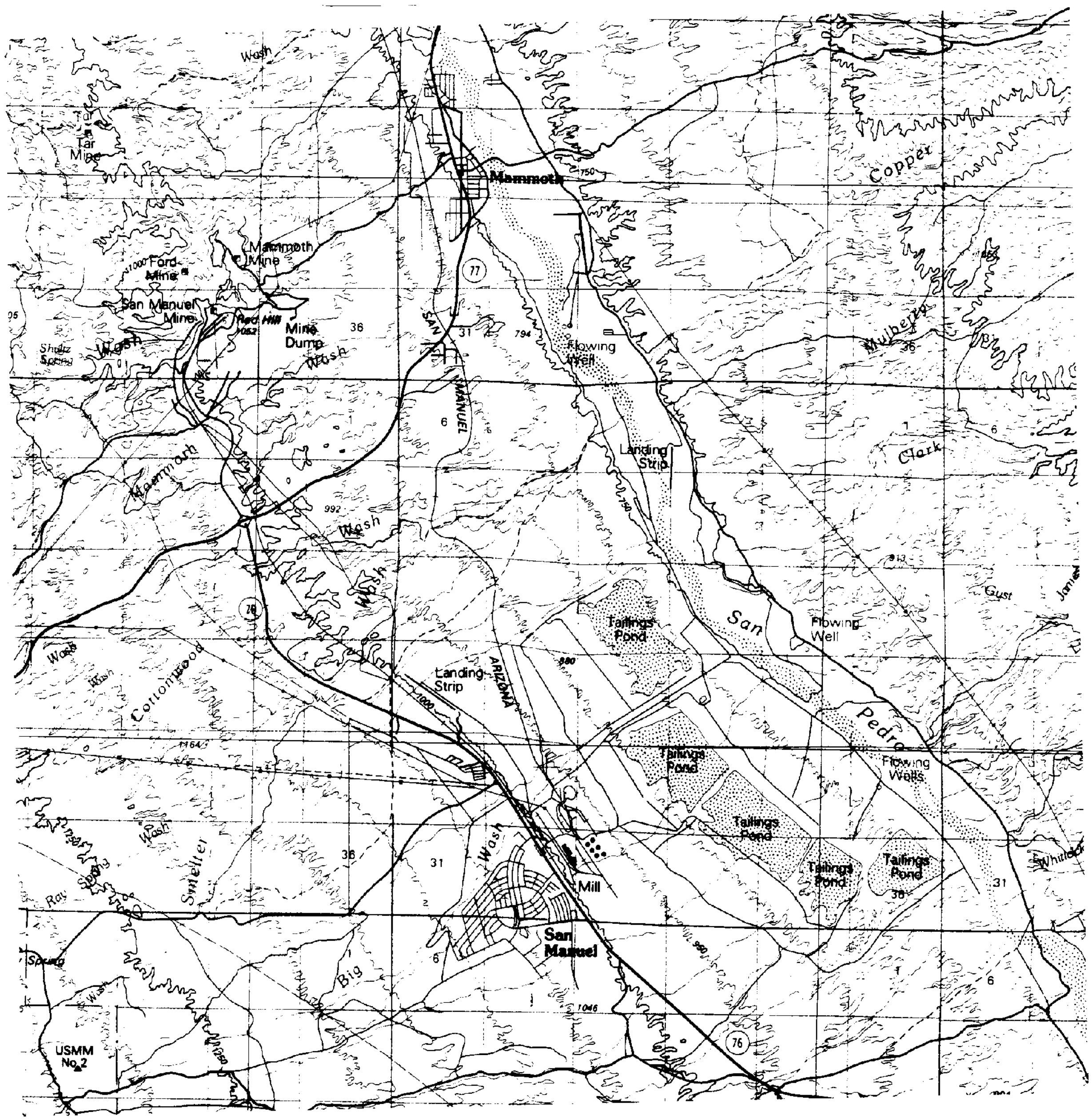


Figure 3-1. San Manuel Mine and Mill Facilities

(Source: USGS, 1986)

3-1). The sulfide ore operations consist of an underground mine, a mill facility for copper and molybdenum flotation, and a smelter and electrolytic refinery. Oxide ore operations consist of an open pit mine and an *in situ* leach operation, a lined leach pile containing ore mined from the open pit, and a solvent extraction/electrowinning facility. Both the electrowinning operation and the electrolytic refinery produce a copper cathode that is 99.99 percent copper. San Manuel has 3,600 employees, 1,500 of whom work in the underground mine. All product is shipped via railroad to Hayden for distribution. A warehouse is being constructed approximately one half mile from the facility to allow for local product storage and distribution and thereby decrease transportation costs associated with distributing the copper products from Hayden, Arizona.

In addition to San Manuel, Magma Copper Company operates an underground copper mine and a smelter in Superior, Arizona, and an open pit copper oxide mine in Pinto Valley, Arizona. The San Manuel mine is currently the largest underground copper mine in the world in terms of production capacity, size of the ore body, and "installed facilities" (Magma, 1988, Weiss, 1985). Mining of the one billion ton San Manuel ore body will continue until 1995 when the ore is expected to be depleted. Magma also owns the Kalamazoo ore body, a faulted segment offset from the San Manuel ore body. The Kalamazoo ore body is located approximately one mile to the west of the San Manuel ore body, between 2,500 and 4,000 feet below the surface. Magma Copper Company began to develop the Kalamazoo in 1990 and is currently mining at a rate of 6,000 tons per day (tpd). A long-term (15 year) labor contract was agreed to in late October 1991 in anticipation of fully developing the Kalamazoo ore body (*Minerals Today*, February, 1992).

In addition to producing copper from sulfide ore from the underground mine, San Manuel operates a flotation circuit for "re-concentration" and "re-processing" of smelter slag. Molten slag from the furnace contains 1.8 percent copper while slag from the converters contains 7 percent copper, concentrations that far exceed concentrations found in the ore (0.7 percent found in sulfide ore) (Weiss, 1985; Magma, 1992c). Slag concentrate from flotation is combined with sulfide ore concentrate and is fed to the smelter.

Investigations of the San Manuel area began in the early 1940s during an increase in demand for copper and other metals. Exploration drilling began in 1943 in the Red Hill area of San Manuel, and claims were purchased by Magma Copper Company in 1944. In 1952, the Reconstruction and Finance Corporation authorized a $94 million loan to Magma for construction of the mine, the plant, railroads, and the company town. (Weiss, 1985) Production level mining began in 1954, following six years of underground exploration and development work. In 1969, Magma was sold to Newmont Mining Corporation. In 1987, Magma was reorganized and spun off to Newmont stockholders. Within two years, Newmont interests in San Manuel were recapitalized and repurchased by Magma. (Magma, 1988)

3.1.3 Environmental Setting

Magma Copper Company's San Manuel facilities are located in the semi-arid southwest desert, an area characterized by a dry climate with warm summers and moderate winters. Average temperatures are approximately 45 degrees Fahrenheit in January and approximately 80 degrees Fahrenheit in July. Based on records dating back to 1954, precipitation in the area averages 13.4 inches/year, with most precipitation occurring during the months of February and August. Although evaporation rates were not measured at the site, evaporation rates in the area are high, with annual evaporation rates of 116 to 118 inches per year measured near Tucson, Arizona.

The town of San Manuel was built in 1950 and is located near the mill. The community consists of approximately 1,200 homes with a population of 5,000. Drinking water is pumped from deep wells in the San Pedro regional aquifer located near the tailings impoundments. As shown in Figure 3-1, the town of Mammoth is the closest town to the mine site, located 1.6 miles northwest with a population of 2,000. The site is bordered on the east by the Galiuro mountains and on the west by the Santa Catalina mountains. The site is west of the Coronado National Forest. The area's primary economic activity is mining, though the San Pedro valley is also used for ranching. According to Magma representatives, cattle graze in areas near the tailings impoundments but not in the area near the mine. No endangered, threatened, or State-protected species are known to be present on facility property or located within one mile of any facility boundary.

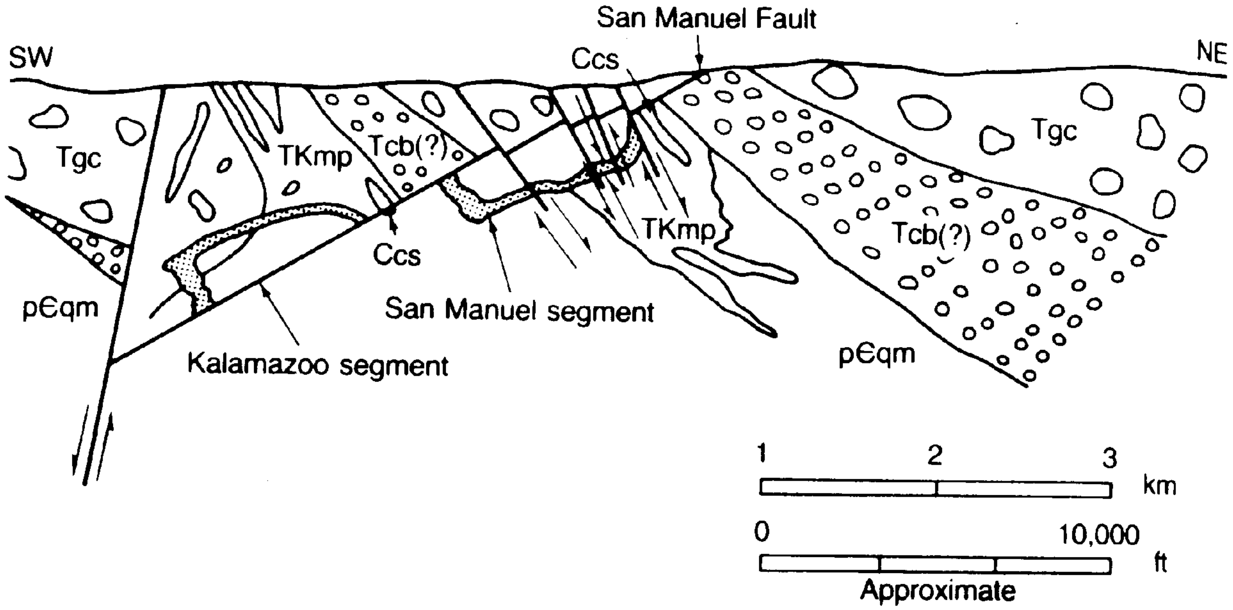
3.1.3.1 Geology

The San Manuel ore body is located in the Lower San Pedro River Basin, in an area of bedrock characterized by intrusive, extrusive, and sedimentary rocks from Precambrian to Pleistocene age, with the oldest rocks found being the quartz monzonite, granodiorite, and minor diabase and aplite of Precambrian age. (Magma, 1992a) The ore body is an elliptical-shaped granodiorite porphyry cylinder that measures 8,000 feet in length, 2,500 feet across, and is situated from 700 to 3,000 feet below the surface. A faulted segment of this ore body lies one mile to the west and is similar in size to the San Manuel ore body. This ore body, called the "Kalamazoo," lies approximately 2,500 to 4,000 feet below the surface and is currently being developed by Magma Copper Company (Magma, 1988).

The San Manuel ore body contains "zones of disseminated copper mineralization at an average grade of 0.65 percent copper or approximately 13 pounds of copper per ton of ore" (Magma Copper, 1988). The Kalamazoo porphyry contains a copper grade of 0.75 percent copper and 0.015 percent molybdenite. Copper porphyry is characterized by small percentages of copper in a large deposit of country rock containing other minerals, such as molybdenum, lead, zinc, manganese, gold, and silver. (Guilbert, 1986) The oxide ore mined at San Manuel is chrysocolla ($Cu_2H_2(Si_2O_5)(OH)_4$). The sulfide ore is chalcopyrite ($CuFeS_2$).

The San Manuel mine and mill facilities are located in the Lower San Pedro River Basin. (Magma, 1992a) The Lower San Pedro River Basin extends from a bedrock constriction dividing the Upper and Lower San Pedro River basins near Benson, Arizona, to the confluence of the San Pedro and Gila Rivers near Winkelman, Arizona, a distance of approximately 65 miles, ranging from 15 to 30 miles wide. Rock types found in the area include Precambrian quartz monzonite found in the Santa Catalina Mountain Block; Cretaceous and Tertiary intrusive, extrusive, and sedimentary rocks found in the Galiuro Mountain Block; the Pliocene-Pleistocene Gila Conglomerate; and Quaternary and Recent gravels and alluvium found in the pediment layers and San Pedro River channel.

The most extensive surficial rock unit in the area is the Gila Conglomerate, which lies above the ore body (See Figure 3-2)



pЄqm = Oracle Granite, actually quartz monzonite
TKmp = monzonite porphyry
Tcb = Cloudburst Formation
Tgc = Gila Conglomerate

(From Lowell and Guilbert, 1970).

Figure 3-2. Structure of San Manuel-Kalamazoo Deposit

(Source: Guilbert, 1986)

. The Gila Conglomerate is characterized by indurated boulder and cobble conglomerate, unconsolidated marls and limy silts and clays, and an unconsolidated sand and conglomerate. These units of the Gila Conglomerate are identifiable on a topographic map based on the degree of dissection of each unit (See Figure 3-1). The marl and limy silt unit is the most heavily dissected showing steep-sided, narrow gullies, while the unconsolidated sand and conglomerate unit exhibits a smooth land surface. An unconsolidated cap of gravel overlies the Gila Conglomerate. Adjacent to the San Pedro River channel, Recent alluvium consisting of unconsolidated gravels, sands and silts, ranging in thickness from 40 to 90 feet, covers the gravel and Gila Conglomerate layers (Magma, 1992a).

The landscape is characteristic of the Basin and Range physiographic province. The area is defined by four distinct physiographic features: the modern San Pedro River channel and floodplain, the Whetstone pediment, the Tombstone pediment and the Santa Catalina Mountain Block. The San Pedro River floodplain extends 5 to 75 feet above the current channel level and roughly 1/2 mile on either side of the channel. The alluvium in the floodplain is comprised of sedimentary and volcanic rocks common to the region. Alluvial deposits extending beyond the floodplain are locally referred to as the Whetstone pediment, characterized by unconsolidated fine-grained silts and clays that extend outward 3 miles (slopes on the Whetstone pediment range from 150 to 180 feet per mile extending from gradational contact with the floodplain to an elevation of approximately 3,400 feet). (Magma, 1992a) The Tombstone pediment is characterized by semi-consolidated to consolidated coarse-grained gravels and cobbles and begins at the gradational contact with the Whetstone pediment at the 3,400 foot elevation. This pediment extends to the Santa Catalina Mountain Block which begins at the 4,000 foot elevation with slopes ranging from 200 to 250 feet per mile. The Santa Catalina Mountain Block comprised of quartz monzonite reaches an elevation of 4,200 feet in the area. Weathering of this mountain block has contributed to the thin layer of unconsolidated quartz and granitic gravel on the Gila Conglomerate in the area of the San Manuel mill site. (Magma, 1992a)

Based on the above description of terrace development in the Lower San Pedro River Basin, San Manuel's tailings impoundments located 1/2 mile west of the San Pedro River are situated on the Whetstone pediment, while the mill facility, smelter, and town of San Manuel extend beyond the Whetstone pediment to the Tombstone pediment. According to Magma representatives, the tailings impoundments were constructed on Gila Conglomerate, an alkaline rock type by nature, that is capable of neutralizing the more acidic nature of the tailings. No data discussing the alkalinity of the Gila Conglomerate or its neutralization potential was available. (According to a 1989 study by PEI Associates, Gila Conglomerate in the San Manuel area is capable of neutralizing 200 pounds of sulfuric acid per ton of Gila Conglomerate.) (U.S. EPA, 1989)

3.1.3.2 Hydrology

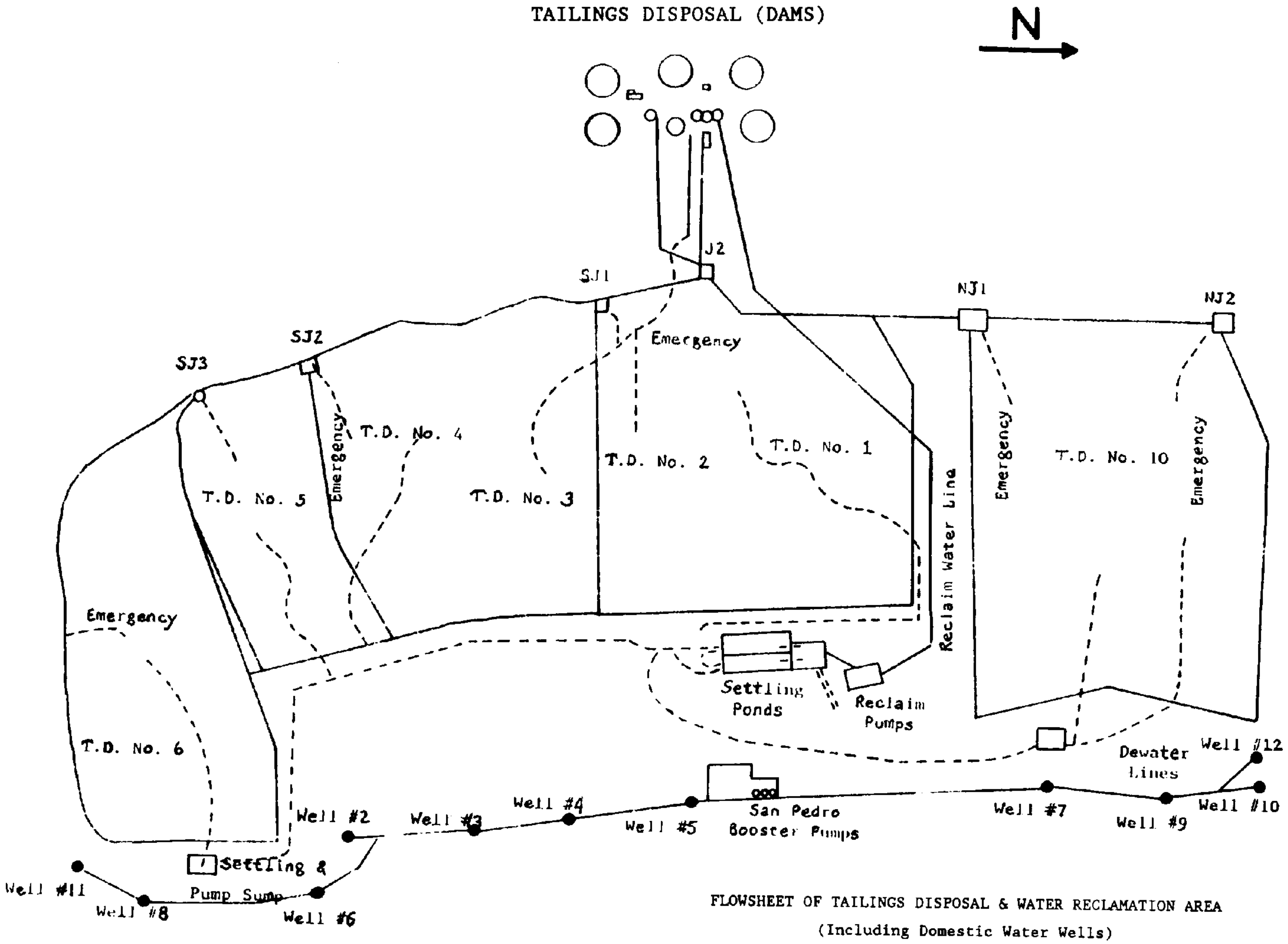
Surface Water: Surface water in the vicinity of the site is limited to the San Pedro River and its tributaries. These are all intermittent streams which maintain highest flows after late summer and winter rainstorms. The San Pedro River is approximately 1/2 mile from the tailings ponds near the mill facility and was not visibly flowing at the time of the site visit. Numerous washes (tributaries to the San Pedro River) also drain naturally through the mill facility and tailings impoundments. In order to prevent storm water runon from reducing tailings impoundment capacities, Magma diverted Courthouse Wash away from the tailings impoundments. References do not indicate when the diversion was implemented (Courthouse Wash is not shown on Figure 3-1). The mine site is 3.5 to 4 miles from the river and is situated between the Tucson Wash to the north and the Mammoth Wash to the south. (Magma, 1992a)

Groundwater: According to Magma, two aquifers underlie the San Manuel area, the floodplain aquifer and the regional San Pedro aquifer (Magma 1992a). The floodplain aquifer consists of groundwater in the floodplain alluvium, which is recharged by precipitation, San Pedro River flow, and seepage from irrigation systems. The regional aquifer is located within the Gila Conglomerate and is recharged by precipitation along the mountain fronts and by stream channel infiltration. According to Magma personnel, dewatering operations at the mine have created an 8-mile cone of depression in the surrounding water table. Impacts on the local groundwater regime caused by the tailings impoundments and pumping from production wells near the tailings impoundments are unclear from the available references. According to an undated study provided by Magma, groundwater in the floodplain is found at depths ranging from 17.4 to 19.3 feet. However, production well information also provided by Magma indicates that two of 11 production wells located below the tailings ponds are artesian.

As described earlier, dewatering in the vicinity of the mine site has created a cone of depression in the area's water table. According to Magma representatives during the site visit in May 1992, the cone of depression encompassed an area eight miles in diameter from the underground mine. However, the 1991 Aquifer Protection Permit (APP) for the heap leach facility describes the cone of depression impacting an 11 square mile area surrounding the mine, substantially less area than the eight mile cone of depression statistic presented during the site visit. In addition, the APP also describes the cone of depression extending from one half to two miles around the mine. Thus, the extent of the cone of depression is not clear.

Published material on the cone of depression is described below. Since the current depth of underground operations is at the 4,080 foot level, the lowest point in the cone of depression may be this same level. According to Magma, the cone of depression has impacted water levels in all directions except to the south and southwest where recharge occurs from the Santa Catalina Mountain Block. A borehole monitored between 1967 and 1982 in the recharge area near the minesite found very little change in ground water levels that measured 2,952 feet above mean sea level (amsl) (Magma, 1991-APP, undated). The deepest area of the cone of depression may be directly under the underground mine. Water level measurements in boreholes in this area have decreased up to 1850 feet due to the dewatering. In addition, as of 1982, the elevation of the water table 1/2 mile to the southeast of the mine had dropped 800 feet. Between 1935 and 1959, ground water levels in the Mohawk mine shaft northwest of the San Manuel mine had decreased almost 400 feet, from 2,504 to 2,106 amsl. (Magma, 1991-APP, undated) The Mohawk mine shaft is currently the most northwestward monitoring point for water levels.

Groundwater from the San Pedro regional aquifer is used as drinking water by the town of San Manuel and also supplies make-up water to the facility. Eleven water supply wells were drilled between 1947 and 1974 that pump approximately 14 million gallons per day from the regional aquifer for use by the town and the mill facility. All eleven wells are situated below the tailings impoundments and just west of the San Pedro River, as depicted in Figure 3-3. Well depths range from 967 feet at Well #2, completed in 1947, to 1,520 feet at Well #9, completed in 1964. According to 1988 information, Well #'s 8 and 11 are artesian. These wells are located immediately below tailings impoundment No. 6. (See Figure 3-3 and Table 3-1) (Magma, 1992d) According to facility personnel, water from these wells has been shown to contain elevated fluoride levels. However, no information was obtained explaining the source of the high fluoride levels.



FLOWSHEET OF TAILINGS DISPOSAL & WATER RECLAMATION AREA

(Including Domestic Water Wells)

	Well #2	Well #3	Well #4	Well #5	Well #6	Well #7	Well #8	Well #9	Well #10	Well #11	Well #12
Depth to GW/ Date Measured	36 ft 5/88	28 ft 5/88	186 ft 5/88	286 ft 5/88	32 ft 5/88	289 ft 4/88	Artesian	180 ft 5/88	153 ft 4/88	Artesian	227 ft 4/88
Land Elevation (above msl)	2539	2510	2550	2525	2539	2520	2560	2470	2444	2564	2507.4
Well Depth (feet?)	967	1305	1411	1007	1006	1300	1290	1520	1435	1238	1380
Date Well Completed	1947	1961	12/61	7/54	7/54	8/58	5/64	10/64	8/69	1/70	2/74
Well Capacity (gpm)	1000 Active	1600 Active	730 Active	1240 Active	1200 Active	1360 Active	1180 Active	1370 Active	1680 Active	1260 Active	1790 Active
Casing Diameter	24-20	24-20-12	20-16-10-8	20-16	24-20-16	22-22-18-14	24-20-18	24-20-18-14	24-20-18	24-20	24-20
Perforated Interval	200-577, 628-900	130-995, 999-1300	440-595, 610-990,540 -1406	130 - 1440, 450-715	200-400 450-725, 770-990	240-480, 470-1106, 1054-1264	390-987, 989-1290	200 - 1100, 1122-1425, 1410 - 1520	210 - 1265, 1265 - 1415	200-1235	200 - 1380

Figure 3-3. San Manuel Tailings Impoundments and Water Supply Wells

(Source: Magma, 1990)

Table 3-1. San Manual Production Wells

(Source: ADEQ Public Water Supply System and Inventory)

3.2 FACILITY OPERATIONS

The San Manuel facility includes two separate areas of operation: the mine complex, which includes the underground and open pit mines, the heap leach, and SX/EW facilities; and the mill and smelting complex, which includes the concentrator, tailings ponds, and smelter facilities. The mine site area is located approximately six miles north of the mill facility and tailings area on Route 76. The main structures at the mine complex are the open pit; seven mine shafts and headframes; the heap leach pad that covers approximately 230 acres; the pregnant, plant feed, and raffinate ponds; the SX/EW plant; a train line for hauling sulfide ore to the mill; a mine water pond; and many support facilities (e.g., shops, offices, etc.). The mine site also includes old Tiger Mine workings and waste rock piles generated largely from open pit operations. (See Figure 3-4)

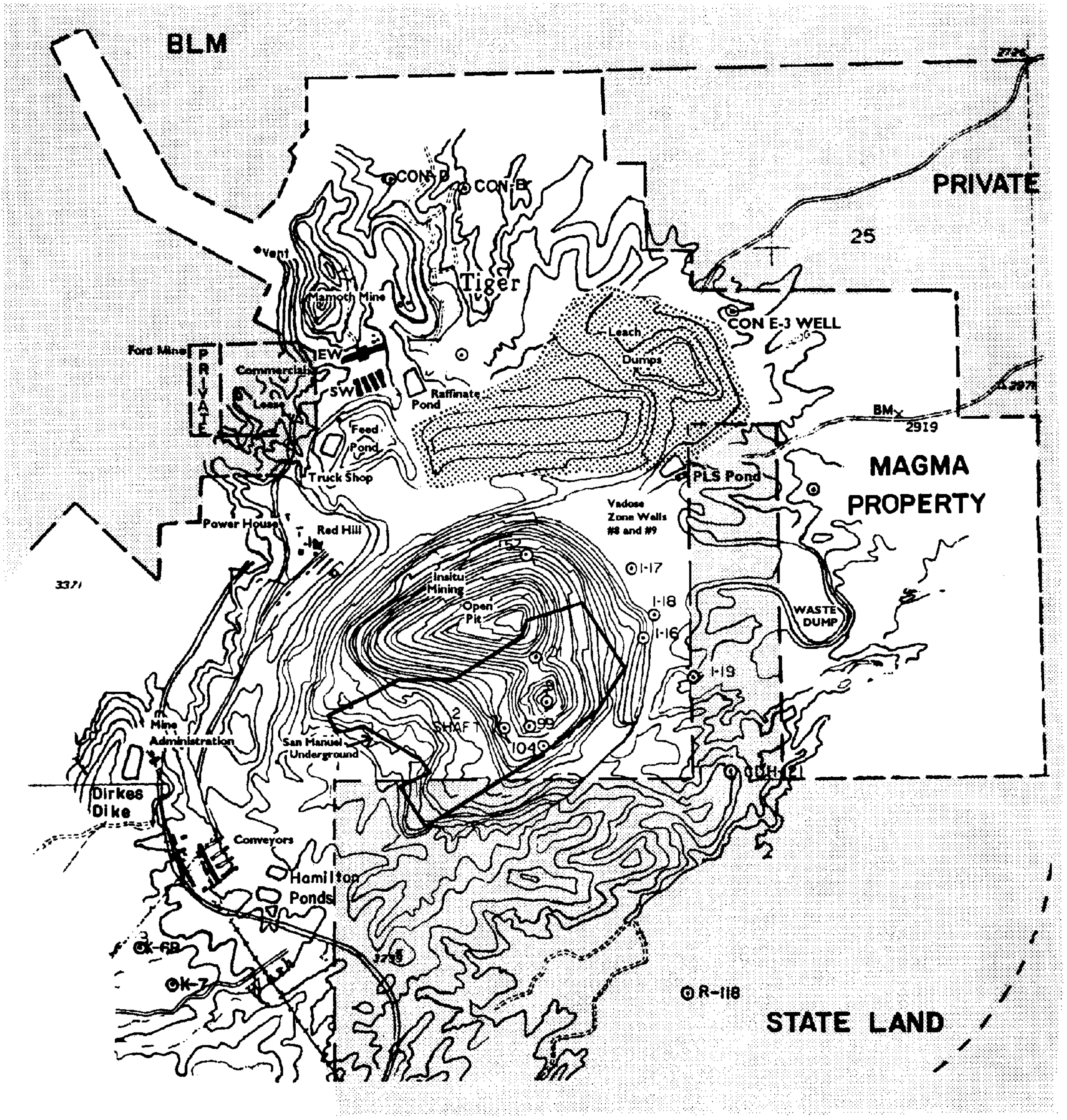


Figure 3-4. San Manuel Open Pit and Mine Site

(Source: Magma, 1992k)

The mill and smelter complex includes the crushing and concentrator buildings, the molybdenum plant, the smelter, the acid plant, the electrolytic refinery, tailings thickeners, seven tailings ponds covering approximately 3,600 acres, and associated support facilities. (See Figure 3-5)

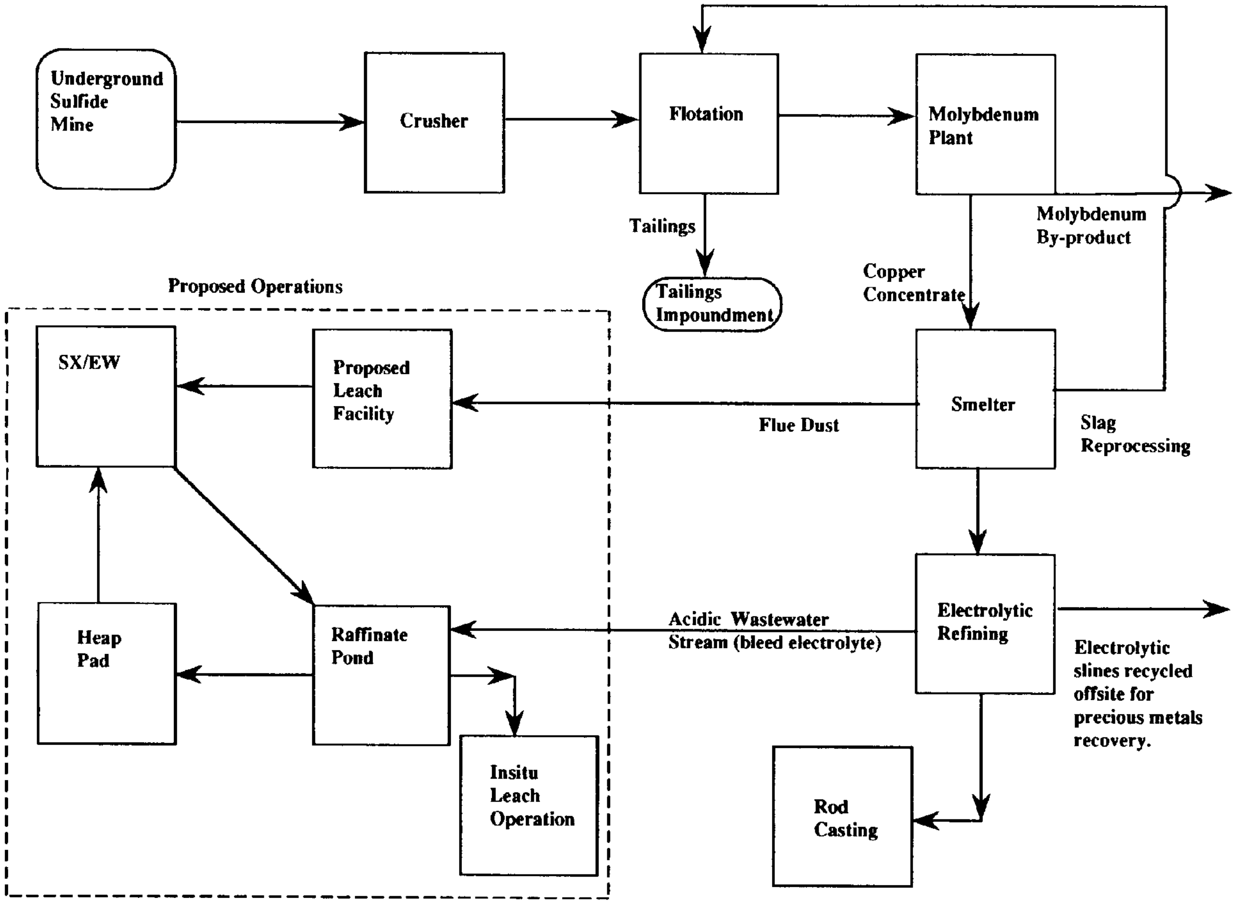


Figure 3-5. Mill Facility

(Source: Magma, 1990)

Both sulfide and oxide copper ores are mined at San Manuel. Magma uses underground block-caving mining methods to extract sulfide ore, and open pit mining to extract oxide ore. The sulfide ore is sent to the mill, smelter, and refinery, while oxide ore goes to the leach pad and the SX/EW plant. The average grade of copper ore removed from the San Manuel ore body is approximately 0.65 percent copper, or 13 pounds of copper in each ton of ore (Magma, 1988). Because Magma recovers two types of ore by distinctly separate methods, the following description of facility operations is divided into two sections by type of ore.

3.2.1 Oxide Ore

The oxide ore body, containing the oxide mineral chrysocolla, was identified in the Northeast portion of the subsidence zone of the San Manuel underground mine (Magma, 1988). Mining of this ore body began in 1986 with the availability of new technologies that made recovery of copper from these lower copper oxide ores economically feasible. The subsidence area created by underground mining currently extends to a depth of 1,300 feet. Magma's open pit mine currently extends to a depth of 800 feet into the subsidence area. Open pit mining generates approximately 30,000 tons of ore and up to 68,000 tons of waste rock per day and is expected to continue until 1995, when the oxide ore body will be exhausted (Magma, 1988).

3.2.1.1 Open Pit Extract

Ore is removed from the pit by drilling blastholes that are 37 feet deep and 9 and 7/8 inch in diameter. Blast patterns consist of between 30 and 120 blastholes. Blasting locations are generally spaced 24 feet apart, and the holes may be pump-drilled prior to blasting.

Blasting is accomplished with ANFO, a mixture of ammonium nitrate and diesel fuel (Magma, 1992b). Periodically, mining hits small perched aquifers and drill holes are pumped prior to blasting. Front end loaders are used to convey the ore to 100 or 120 ton electric diesel haul trucks. Two Caterpillar graders, as well as several track dozers, are also used in the mine (Magma, 1992b). Roads in the open pit are 80 feet wide, graded to a maximum slope of 10 percent to accommodate the haul trucks (Magma, 1988). Pit benches are built with a cut/strip ratio of 1:1, to ensure the stability of underground mine workings. Benches on the north side of the pit accommodate *in situ* leaching wells; these are described below (Magma, 1988). Ore from the pit is transported to the 230-acre heap leach pad. Waste rock is currently being disposed of in an angle of repose waste rock pile located inside the pit, on the southern edge.

According to a 1989 topographic map (Figure 3-4), the pit extends approximately 4,200 feet east to west, and 5,200 feet north to south. The current dimensions of the open pit were not determined. At the time of the site visit, mining had reached a depth of 800 feet and is anticipated to reach a final depth of 1,200 feet.

Between January 1, 1986 and March 31, 1991, almost 46 million tons of ore and over 95 million tons of waste rock were removed from the open pit mine, with approximately 50,000 tons of waste rock moved per day (Magma, 1992b). The stripping ratio for this period is 2.08:1 (waste to ore). After March 1991, an estimated 73 million tons of material remain to be removed from the pit, which will yield an additional 46 million tons of oxide ore and over 27 million tons of waste rock, a rate of 40,000 tpd (Magma, 1992). For this period the mine has a stripping ratio of 1:1.7 (waste to ore).
References do not indicate the reason for the decrease in the waste rock mined.

3.2.1.2 Beneficiation of Oxide Ores

Beneficiation of oxide ore at San Manuel includes the *in situ* leach operation and the heap leach facility, where oxide ore is leached with acid, as well as the associated solvent extraction and electrowinning plants. Pregnant leach solution from the *in situ* leaching operation and the heap leach is sent through a series of extraction and stripping stages in the solvent extraction plant. Cathode copper (99.999 percent pure) is recovered in the electrowinning plant.

In situ Leaching

In situ leaching is a solution mining method designed to leach out valuable minerals without removing the ore. *In situ* leaching production began in 1987 at San Manuel and is currently located on the north side of the open pit. *In situ* wells drilled into mine benches access additional reserves of acid-soluble oxide ore located beneath the open pit and above the depleted portion of the underground sulfide ore mine (Magma, 1988). The wells are made of PVC piping and are drilled to a depth of 1,000 feet into an area of the ore reserve that has been rubbleized by underground mining. Approximately 272 million tons of ore in the underground mine workings are currently subjected to the *in situ* leaching process with an estimated two billion gallons of solution currently in inventory within the *in situ* ore body (Magma, 1988). Because open pit operations may extend into *in situ* leach areas, *in situ* wells at San Manuel are typically temporary wells that fail when open pit mining extends into those areas. Magma drills new wells to accommodate additional leaching operations as necessary. The number of *in situ* wells in operation at any one time was not determined.

Raffinate solution, a weak sulfuric acid, from the solution extraction (SX) plant is pumped from the raffinate pond to the *in situ* wells and injected into the ore body at a rate of 3,000 gpm, or approximately 21,000 tpd (Magma, 1992b). Solutions percolate through the oxide ore body, and copper-bearing solution is collected in one of two ways: via recovery wells equipped with submersible pumps; or in the old portion of the underground sulfide mine at the 1,400 and 1,800 levels (Magma, 1988, 1992). In these old underground workings, panels have been dammed to create collection areas. Total pregnant leach solution recovered by these collection methods is 3,100 gpm or 22,320 tpd (Magma, 1992). The recovered solution has a high solids content (approximately 600 ppm), and is pumped to the surface from the underground sulfide mine to a sedimentation pond, where the solution is clarified. Flocculants are added to encourage precipitation of the solids in this pond and pregnant solution is fed to the Plant Feed pond where it is combined with pregnant leach solution from the heap leach and sent to the SX/EW plant. Solids that accumulate in the sedimentation pond are dredged and disposed of in the subsidence area beside the open pit. (Magma, 1992b) Information on the constituent analysis of the sediments, and the location and construction of the sedimentation pond, was not obtained.

Copper production from the *in situ* operation was not fully operational until the ore body was completely saturated and the volume of raffinate solution injected equaled the amount of pregnant solution recovered. Initially, raffinate from the SX process was sent to the *in situ* wells at a flow rate of 8,000 gallons per minute (gpm). The amount of pregnant leach solution recovered in the panels measured a flow rate of only 7,200 gpm, a solution loss of 800 gpm to surrounding ore and country rock. It was not determined when the *in situ* operation achieved one hundred percent solution balance.

Heap Leach

Run-of-mine oxide ore from the open pit is hauled to the heap leach at a rate of approximately 30,000 tpd. The heap leach pad is approximately 3,000 feet wide by 6,000 feet long, covering 230 acres (See Figure 3-6)

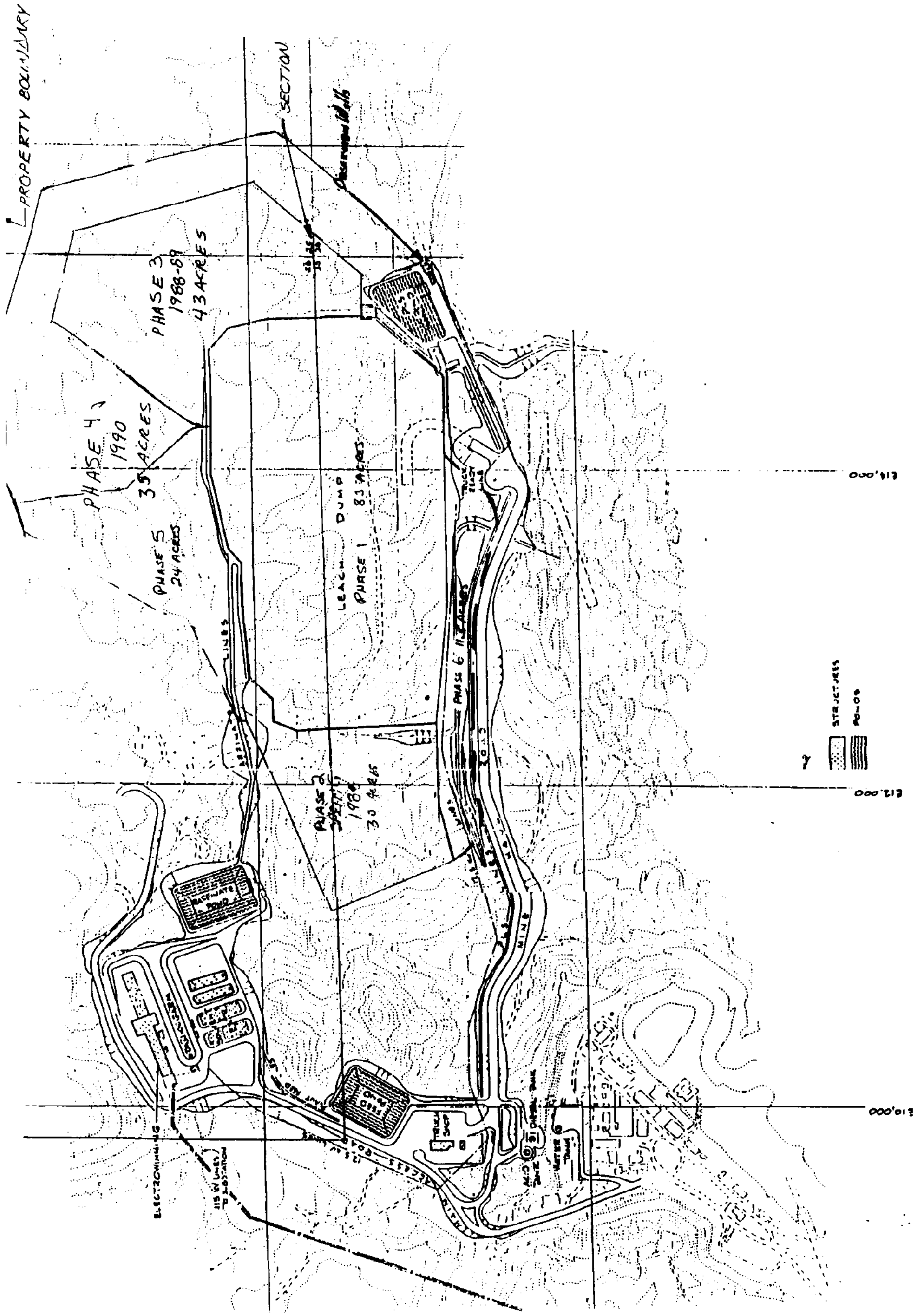


Figure 3-6. Heap Leach at San Manuel

(Source: Magma, 1992l)

(Magma, 1989). According to San Manuel's 1989 Groundwater Quality Protection Permit (GWQPP), the heap was originally designed to cover 113 acres with a planned 82-acre expansion, for a total of 195 acres. The latest additions to the heap leach were completed in 1991 and 1992, Phases 5 and 6, respectively. These additions added the final 35 acres to the heap leach pad.

The GWQPP specified a maximum height of 310 feet for the heap leach, while the Aquifer Protection Permit (APP) listed a height limit of 330 feet. However, according to Magma representatives and the Arizona DEQ, the height of the leach pile would reach approximately 400 feet in 1992. (Helmer, 1992) The height of the heap leach during the EPA site visit was not determined.

In construction of the initial leach pad, soils beneath the leach pad were compacted and a 7.5 ounces per square yard layer of geotextile material liner was installed prior to laying the leach pad. According to Magma representatives, the leach pad is underlain by Gila Conglomerate with a permeability of 3.3 X 10^{-7} cm/sec. Initially, three thicknesses of HDPE liners were used to make up the original 113-acre leach pad: a 60 mil high density polyethylene (HDPE) lined 71 acres; a 100 mil HDPE lined 17 acres; and a 40 mil HDPE lined the other 25 acres. Which areas of the leach pad were covered with which liner type was not described. Apparently, 60 mil liners were used over ridges of the heap, while 100 mil liners were used to line collection ditches. Eighteen inches of gravel were placed on top of the liner to prevent tearing and a collection system of perforated pipes was installed to prevent pooling. (State of Arizona, 1989) According to San Manuel's 1989 Aquifer Protection Permit, construction and expansion of the heap leach was organized in six phases. Phase I of the heap leach, 83 acres, was completed in 1985; Phase II, 30 acres, in 1988; Phase III, 43 acres, in 1989; Phase IV, 35 acres, in 1990; Phase V, 24 acres, in 1991; and Phase VI, 11.5 acres in 1992. (Magma, 1991-APP)

Leach pad extensions were prepared in a similar manner to the Phase I pad, however only 6 ounces per square yard of geotextile material was used to overlay the prepared sub-grade, and 60 mil and 80 mil HDPE liners were installed. The 80 mil liners were used to line solution collection channels. (State of Arizona, 1989) Collection channels along the southern edge of the heap are 16 feet wide by 2.7 feet deep, designed to handle in excess of 80,000 gpm, in addition to a 100-year/six-hour storm event (3.4 inches). Solution collected in the channels empties pregnant leach solution into two 30-inch diameter pipes that divert contents to the PLS pond. (Magma, 1991-APP)

Ore is dumped on the heap in 15 foot lifts, with each lift containing approximately 110,000 tons of ore, covering 125,000 square feet. Wobbler sprinklers and pipes are laid on top and initially pretreat the lift by spraying a weak sulfuric acid (250 grams per liter, (gpl)) solution from the raffinate pond for an unspecified duration, followed by the application of 10 gpl solution until the heap is saturated. Pretreatment typically requires 20 pounds of acid per ton of ore to ensure 85 percent recovery of acid soluble copper in 60 days or less. The length of time required for pretreatment was not determined except that the heap reaches saturation point prior to beginning the treatment operation.

After pretreatment, solution of 250 grams per liter (gpl) sulfuric acid is applied to the lift at an application rate of 0.80 gpm/100 square feet (14,500 gpm total) (Magma, 1991-APP) . (According to the 1989 GWQPP, 11,000 gpm were applied at the same application rate for a total of 25 pounds of acid per ton of ore.)

Once the heap is saturated, it is left to rest for three days and is saturated again with 250 gpl sulfuric acid. The heap is then left to percolate for three more days, after which barren solution from the raffinate pond, averaging 10 gpl acid, is applied (at an unspecified rate) to the heaped ore for the remaining life of the heap. (Magma, 1991-APP) Each lift of 110,000 tons is leached for approximately 60 days. Drainage pipes were placed in the sides of the heap to prevent heap erosion. (Magma, 1991-APP) The total volume of sulfuric acid used per year to operate the heap leach and *in situ* operations was not determined. However, in 1987, a total of approximately 16,136 gpm of raffinate were circulated to the leach operations. Acid content of this raffinate was recorded as 10.6 gpl, or 85,877 pounds per hour. (See Table 3-2) Combined solution loss from the heap leach and *in situ* leach operations exceeded 1,000 gpm in 1987. The destination of lost solution was not determined. According to the Arizona DEQ, the loss of raffinate (1000 gpm) may have been due to *in situ* leaching not stabilizing, with solution diverting into cracks in the rock where flow is not recoverable.

Pregnant leach solution forms as the acid percolates through the pile and dissolves copper minerals; it flows into drains and collection pipes beneath the pile. The collection pipes flow into lined ditches ranging from 16 to 22 feet wide and a minimum of 32 inches deep. The collection ditches extend alongside the heap, directing pregnant solution through sediment traps to the Pregnant Leach Solution (PLS) Pond (Magma, 1988, 1992). Sediment traps prevent 90 percent of material greater than No. 40 sieve from entering the PLS pond and must be cleaned periodically to ensure continued effectiveness. Sediment removed from sediment traps is deposited on the heap; quantities were not determined.

The heap leach facility was designed as a no-discharge operation, with the heap, its collection ditches, the pregnant solution pond and plant feed ponds all lined to prevent solution loss. Perforated 3-inch diameter drainage piping was installed at 20-foot intervals at the toe of the heap leach to facilitate PLS drainage to the solution drainage ditches (Magma, 1991-APP). The expected life of the leach pad is approximately fifteen years, while the life of the pond liners is not specified. Heap leach operations are expected to be discontinued in 1996.

Solvent Extraction/Electrowinning Operation

The Solvent Extraction/Electrowinning plant (SX/EW) consists of the PLS pond, the plant feed pond, the raffinate pond, and the SX/EW plant. The solvent extraction plant operates at an 18,000 gpm throughput that yields up to 50,000 short tons of copper per year from the electrowinning plant. (Magma, 1991-APP) Operations at the SX/EW began in 1986 with the beginning of oxide ore open pit mining. The PLS pond is lined with an 80-mil HDPE liner overlying compacted subgrade material and has a 5-million gallon capacity.

Table 3-2. Leach Solution Flow at San Manuel

Leach Flow/ Description	Flow Rate (gpm)	Copper Content (gpl)	Copper Content (lb/yr)	Acid Content (gpl)	Acid Content (lb/yr)	Stream Medium	Specific Gravity
Raffinate to Dumps	9200	0.15	680	10.0	46061	AQ	1.04
PLS from Dumps	8924	1.57	7020	3.00	13404	AQ	1.03
Dump Leach Solution Losses	276	-	-	-	-	AQ	1.00
Raffinate to Acid Mix Tank	1287	0.15	95	10	6443	AQ	1.04
Pretreatment Acid	213	-	-	1700	181305	LIQ	1.83
Pretreatment Solution	1500	0.13	95	250	187748	AQ	1.18
Raffinate to *In situ*	7953	0.15	588	10.0	39816	AQ	1.04
Acid Addition to *In situ*	47.3	-	-	1700	40290	LIQ	1.83
Barren Leach Solution to *In situ*	8000	0.15	588	20.0	80106	AQ	1.04
PLS from *In situ*	7200	1.57	5664	3.00	10814	AQ	1.03
***In situ* Leach Solution Losses and Retention**	800	-	-	-	-	AQ	1.00
Raffinate Make-Up Water	1017	-	-	-	-	AQ	1.00
Raffinate Total	16136	0.16	1268	10.6	85877	AQ	1.04
Total Acid to SX	3.28	-	-	1700	2790	LIQ	1.83

(Source: Bechtel, Inc. Leach Process Flow Diagram, February 10, 1987)

Berms surround the outside of the pond to prevent run-on of precipitation, and overflow wier ditches were installed on the southeast corner of the existing leach dump that are designed to receive excess PLS or PLS rainwater, and direct their flow to the subsidence area in the open pit. (Magma, 1991-APP)

Pregnant leach solution from the heap leach is directed to the PLS pond. References do not indicate the rate at which pregnant leach solution enters the pond. From the PLS pond, PLS is directed to the plant feed pond, where it is mixed with pregnant leach solution from the *in situ* operation. The plant feed pond has a ten million gallon capacity and is lined with a 60 mil XR5 reinforced fabric liner placed on top of compacted subgrade material. Berming around this pond is designed to reduce storm water runon and to contain the maximum process volume in addition to precipitation from a 100-year/6-hour storm event. (Magma, 1991-APP)

Two of three 4,800 gpm capacity pumps in the PLS pond are used to pump effluent from the PLS pond to the plant feed pond via a 14-inch diameter stainless steel pipe. The 14-inch diameter pipes feed a 24-inch diameter stainless steel manifold that discharges the PLS to two 18-inch diameter polybutylene pipelines. The pipes flow into the SX/EW plant feed pond. The average concentration of sulfuric acid in solution entering the pregnant leach solution pond and the plant feed pond is 19.12 gpl, with approximately 1.25 gpl copper. (Magma, 1992b)

The solution extraction/electrowinning (SX/EW) operation recovers copper from the pregnant leach solution (PLS). The PLS is subjected to a series of extraction and stripping stages that isolate the copper mineral to produce an electrolyte for electrowinning. Electrowinning uses an electrical charge to plate metals to the cathode. Concurrent with deposition of copper on the cathode, acid and oxygen are generated at the anode, creating spent electrolyte solution. (Weiss, 1985)

Pregnant leach solution from the plant feed pond flows into the solvent extraction plant (SX) at a rate of 18,000 gpm. The SX plant has two parallel trains. Each parallel train has two extraction tanks and one stripping tank. In the extraction tank, the pregnant solution is mixed with organic solution containing 95 percent kerosene (organic) and 5 percent of an unspecified chelating agent. The chelating agent in the organic complexes with the metal ions and a density separation is allowed, where copper preferentially transfers from solution in the aqueous phase to solution in the organic phase. After mixing, the solvent and aqueous solutions separate, since they are immiscible. Copper-loaded organic then flows over wiers into a collection system. Not all copper has been removed from the partially stripped leach solution, so it is routed to a second extraction tank to be mixed again with barren organic. The loaded organic from the second extraction tank is sent back to the first extraction tank as make-up organic to collect copper from the richer PLS. (The amounts of reagents used in this operation were not determined.) In each train, loaded organic from the first extraction tank flows from the collection system to the stripper tank. In the stripper cell, an electrolyte (spent electrolyte from the electrowinning plant, as well as additional fresh electrolyte) is added in the mixer portion of the tank, stripping the organic of the copper. Copper binds to the electrolyte, and is sent to the electrowinning plant for final recovery of copper onto lead anodes. (See Figure 3-7)

Figure 3-7. Solvent Extraction/Electrowinning

(Weiss, 1985, Magma, 1992b)

Solvent extraction leads to periodic buildup of impurities in the organic tanks, including silt that may collect on the bottom of the tanks. Magma "cleans" (or recycles) the organic by flooding off the organic and piping it to a centrifuge. The centrifuge removes the "gunk" from the organic, then the organic is sent through a clay filter to remove any remaining impurities. "Recycled" organic is sent to the second tank of solvent extraction for mixing with partially loaded PLS. According to Magma representatives, gunk from the centrifuge as well as spent clay from the clay filter, is collected and transported to the heap for leaching. It was not determined how often this procedure is performed, nor the volumes of gunk or clay material generated. In addition, SX tanks are periodically drained and accumulated silt is vacuumed off. It was not determined how the silt is managed.

Once stripped of copper in the extraction tanks, the barren solution (raffinate) from the solvent extraction tanks is routed to the 10 million gallon capacity raffinate pond where solution is recycled to the heap leach and *in situ* leach operations by three 5300 gpm capacity pumps. Unlike the PLS and plant feed ponds, the raffinate pond is not equipped with an overflow ditch. (Deming, April 1991).

The electrowinning operation at San Manuel has a total production capacity of 50,000 short tons per year, twice what was produced when operations began in 1986. The electrowinning operation consists of rectifiers that provide power to the electrowinning cells, and the cathode stripping machine (Magma, 1992b). The rectifiers provide an output capacity of 1,000 to 18,100 amperes and 55 to 217 volts that are distributed to the lead anodes. The EW facility contains four rows with 47 cells per row, with each cell constructed of concrete and lined with flexible PVC, and rigid PVC buffer sheets. Sixty-one lead anodes and sixty stainless steel mother blank cathodes are placed in each cell (Magma, 1992b). Small amounts of cobalt (from 0.5 to 1.0 pounds per ton of copper) are added to the electrolyte bath to maintain the integrity of the lead anodes. The current is distributed to the lead anode and flows through the electrolyte to the cathode. Copper is plated onto stainless steel plates to obtain concentrate that is 99.999% copper. The operation takes place over a 7-day period, yielding 100 pound plates of copper. The loaded plates are removed from the bath and washed at the cathode stripping machine to remove any residual electrolyte. Information on the amount of wash solution waste generated during washing of the plates was not determined, nor was it determined how the wash solution is managed. The cathode stripping machine also loosens the sheets from the stainless steel blank, which is re-cycled for re-use in the electrowinning baths. Cathode copper sheets are transported to San Manuel's Rod Plant or sold as sheets to customers (Magma, 1992b).

Reagent storage tanks are located in between the SX and EW operations, on a gunite surface sloped to prevent runoff (Magma, 1992b). Magma lists monthly reagent usage of diluent (kerosene) at approximately 32,000 gallons, used at a rate of 6 to 10 gallons per ton of copper produced. Magma also lists a usage rate of 2500 gallons per month of an unspecified lixiviant ("Lix 984"). Approximately 2,800 pounds of cobalt sulfate are used per month at a rate of 0.5 to 1.0 pounds per ton of copper produced.

3.2.2 Sulfide Ore

Sulfide ore mined at San Manuel is chalcopyrite and is found in the one billion ton ore body. Sulfide ore is extracted by underground mining, which had reached a depth of over 4,000 feet below the surface at the time of the site visit. Beneficiation of chalcopyrite at San Manuel takes place at the mill facility and includes crushing, grinding and flotation operations. Mining of the San Manuel ore body will continue until 1995 when it is expected to be depleted. Mining of the Kalamazoo ore body will continue for at least an additional 15 years. Development work at the Kalamazoo underground workings currently hauls ore at a rate of 10,000 tpd. A 15-year labor agreement was a prerequisite for developing the Kalamazoo ore body and that has recently been concluded (Greeley, 1992).

3.2.2.1 Extraction

Since 1948, when underground mining began at San Manuel, over half of the one-billion ton ore body has been mined using underground block-caving methods. Currently, 56,000 tpd is hauled to the surface and transported to the mill. Magma is currently attempting to keep costs at San Manuel to less than $4/ton of ore hauled. No additional information was obtained regarding costs of current operations or projected costs of mining at San Manuel or Kalamazoo. Relatively little waste rock is generated from underground mining compared to waste rock generated in the open pit. It was not determined where waste rock is placed in the underground mine. As of 1987, the areal extent of the underground mine appeared to be approximately 0.4 by 0.7 miles, oriented below the southern half of the oxide open pit mine. (Magma, Undated Map, "Magma Copper Minesite") (See Figure 3-3).

Underground mine workings are located between 700 and approximately 4,000 feet below the surface and are accessed by four "production" shafts and three "service" shafts. The service shafts (Shafts 1, 4, and 5) provide intake ventilation and supplies, while the production shafts (3A, B, C and D) are used to haul ore to the surface as well as serving as exhaust shafts. The ventilation system circulates up to 1 million cubic feet per minute of forced ventilation air into the underground mine workings. Production and service shafts range in depth from 2,729 feet (shaft 4) to 4,123 feet (shaft 5). Shaft 4 is the main service shaft transporting employees and supplies, as well as containing the primary compressed air lines and the *in situ* leaching pump lines that return collected PLS to the surface for transport to the plant feed pond. Shaft 5 is a multipurpose shaft that also provides access to the Kalamazoo ore body (Magma, 1988). Production and service shaft activities at the mine site are monitored by computers in the mine surface control room.

The underground sulfide ore body is mined using the block-caving method, which entails blasting sections of the ore body above the grizzly level and allowing gravity to collapse horizontal slices of ore (Magma, 1988). Ore falls through the grizzly level and goes through a series of vertical or inclined shafts that transfer ore to the haulage level into ore cars. At the grizzly level, very large pieces are reduced in size manually with a sledge hammer. The train of ore cars transfers the ore to dump pockets, where it is drawn up to the surface with five ton skips to the top of the production shafts, and dropped into coarse ore storage bins (Magma, 1988).

Prior to transport to the mill, ore is subjected to primary crushing in one of four gyratory crushers located in the underground workings. Iron detectors are installed on conveyors for the removal of tramp iron. The ore is sent to receiving bins for transport to the mill in 100 ton capacity rail cars. Ore from the underground mine is generally sized to less than six inches. In some cases, the coarse ore may be shipped directly to the mill without primary crushing. A 20,000 ton coarse ore storage bin at the mill is used to store ore for holding prior to secondary and tertiary crushing.

Ground water that infiltrates the mine workings must be pumped to the surface in order for underground operations to proceed. Sumps are used to dewater the underground workings, pumping water through two lines to the surface through shaft 1 (approximately 1,800 gpm) and shaft 2 (approximately 3,000 gpm). Mine water is combined in a 24-inch steel pipe and stored in a one acre settling pond prior to storage in two 100,000 gallon holding tanks for subsequent use at the mill. (It was not determined if the settling pond is lined.) Five 2,000-2,500 gpm pumps located adjacent to the holding tanks in the No. 3 yard area send water to the mill. Each pump's operating status is monitored from the mine surface control center, and they are inspected daily. (Magma, 1992) Dewatering in the underground mine workings has created a large cone of depression in the area of the mine.

3.2.2.2 Beneficiation

Sulfide ore beneficiation at San Manuel takes place at the mill facility where ore is initially sized by crushing and grinding. Both ore and slag are subjected to rougher and cleaner stages of flotation. According to Magma representatives, all slag (including older slag piles), has been milled and concentrated in the flotation circuit since 1974. The flotation concentrate is then sent to the molybdenite plant where by-product molybdenite is recovered from the copper concentrate. The mill has the capacity to beneficiate 60,000 tpd of ore producing 30 percent copper concentrate. (Magma, 1992c) The number of tons of molybdenite by-product produced per day was not determined.

Crushing and Grinding

The first step in ore beneficiation is grinding. Ore from the coarse ore storage bins at the mill site is conveyed to a double deck screen, where undersized material (< 1 inch) is conveyed to fine ore bins, while oversized material (>1 inch) is sent to one of the cone crushers. The amount of ore sent to each grinding circuit is monitored by a weightometer. (Magma, 1988) The cone crushers discharge to the surge bins. Ore from the surge bins is conveyed to the single deck vibrating screen for further sizing.

Fine material (<1 inch) from the single deck screen goes to the fine ore bins, and remaining oversized material goes to another standard cone crusher for additional crushing. After final crushing, all material meets the grinding circuit size requirements of minus one inch. (Magma, 1990)
The crushed ore is then conveyed at a rate of 3,500 tons per hour into one of 13 automated wet grinding circuits, each with one rod mill and two ball mills. Water used in the grinding operations is pumped from mine water surge tanks. The rate at which water enters the mill circuit from the mine water surge tanks was not determined. Ten of the circuits consist of a 10 foot x 13 foot rod mill and two 10 foot x 10 foot ball mills. The remaining three circuits include one 12 1/2 foot x 16 foot rod mill and two 12 1/2 foot x 14 foot ball mills. The rod and ball mills are cylindrical vessels filled with the ore and steel rods or balls that rotate on a horizontal axis grinding the ore. After initial grinding in the rod mills, ore is sent to a cyclone for sizing. Greater than 3mm material (the cyclone underflow) is transported to the ball mills for additional grinding. The hydrocyclone overflow, less than 3mm material, is transferred to the pulp distributor flotation feed. Overall, the grinding circuits reduce the ore size to 80 percent passing 200 mesh. (Magma, 1990)

Copper Flotation

There are two separate, two-stage froth flotation systems, one for ore and another for slag and other materials (e.g., refractory bricks). The incoming ore feed is 0.63 percent copper; the incoming slag feed is 1.8 percent copper. There are eight flotation circuits in the mill at San Manuel, an unspecified number of which are allocated to slag flotation. Approximately 56,000 tons of ore feed and 2,300 tons of slag are beneficiated at the mill each day**.** Methylisobutyl carbonol (MIBC) is used as a frother in the flotation circuit. Collectors include sodium xanthate, fuel oil (jet fuel A, which is used as a molybdenum collector), and VS M8, a proprietary flotation agent containing carbon disulfide. The underflow is sent to the tailings thickener. The two types of flotation circuits, one for ore and one for slag, are exactly the same except that a different primary collector, Dithiophosphate 55741 (an American Cyanamid product), is used in slag flotation. (See Figure 3-8)

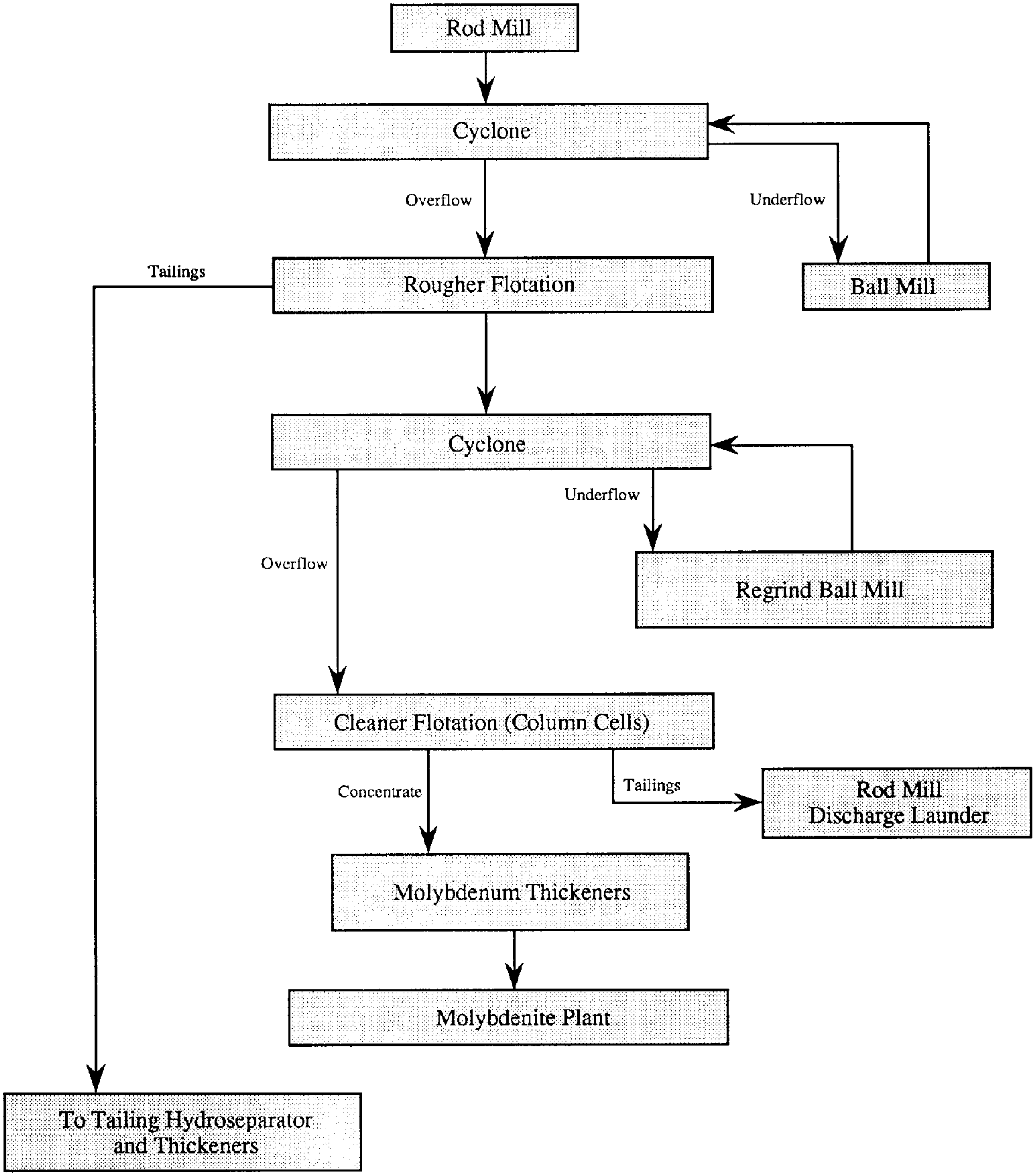


Figure 3-8. Copper Flotation Circuit

(Source: Magma, 1990)

The eight flotation circuits include a total of ten 2,000 cubic foot rougher cells, and 143 rougher cells of smaller dimensions (300 cubic feet). Cleaner flotation takes place in sixteen 39-foot high column flotation cells that concentrate the ore to 30 percent copper. The overflow from rougher flotation (containing an unspecified copper concentration) is transferred to a cleaner/column flotation stage in each circuit, while underflow goes to Magma's tailings thickeners. In each circuit, the second stage of flotation occurs in the 39 foot high, 40 cubic feet cleaner/column cells. The overflow from cleaner cells is 30 percent copper and goes to separate collection launders for transport to the molybdenite plant. The underflow goes to the ball mills and is returned to the rougher for additional flotation.

Tails from the rougher circuit are sent to the tailings hydroseparators. Tailings from the mill are typically 30 percent solids and after the thickeners, tailings are 53 percent solids. The quantity of tailings generated per day was not determined. Approximately 400 million tons of tailings have accumulated in San Manuel's tailings impoundments. (Helmer, 1992)

Molybdenum Plant

Copper concentrate from flotation contains approximately 1.0 percent molybdenum disulfide (molybdenite), a concentration high enough to generate a saleable by-product. The molybdenite plant consists of additional stages of flotation: one rougher stage, three cleaner stages, and five recleaner stages that separate the molybdenum from the copper concentrate. (See Figure 3-9)

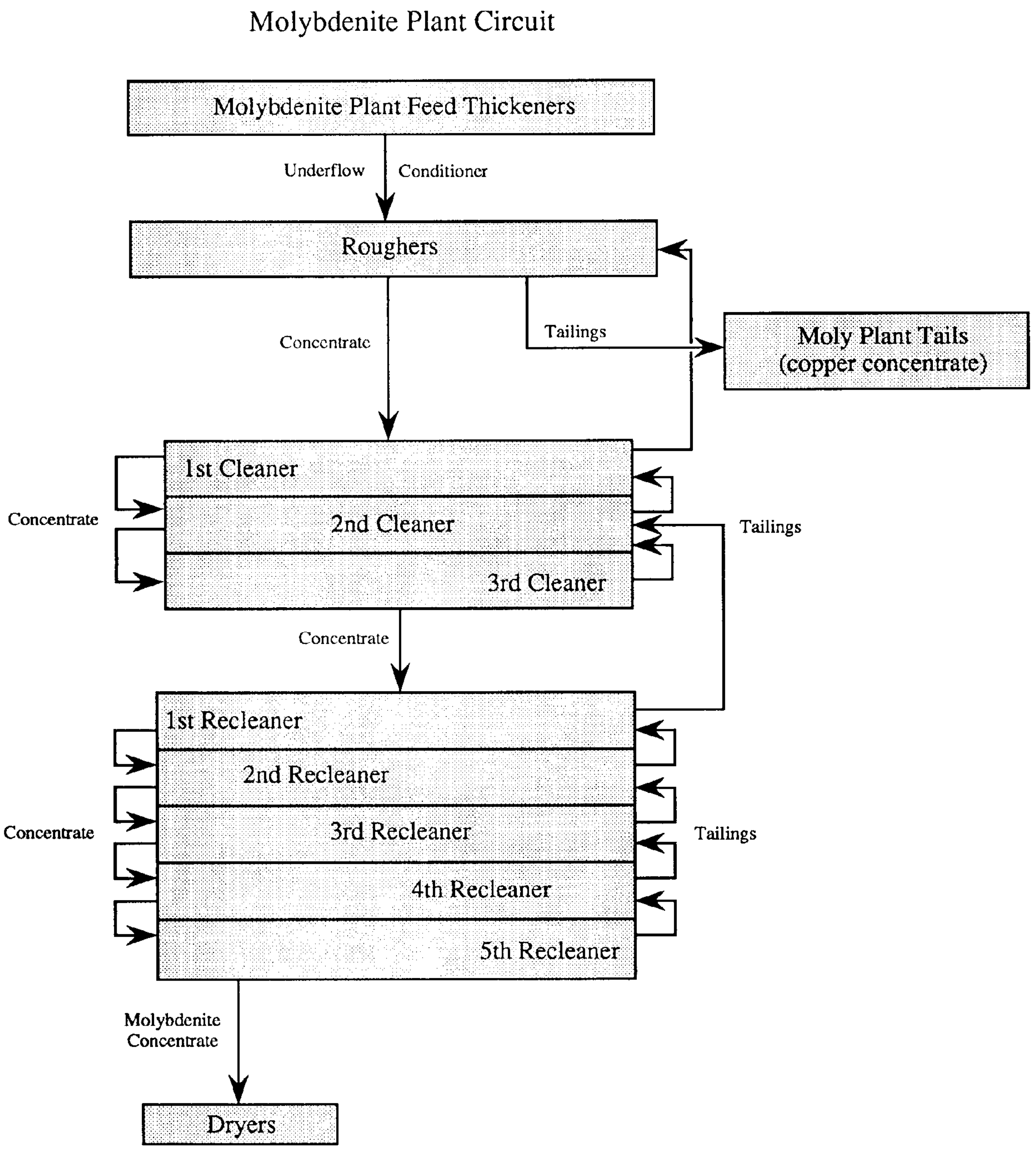


Figure 3-9. Molybdenite Plant Circuit

(Source: Magma, 1990)

The copper concentrate is first added to rougher flotation cells, where sodium cyanide is used to suppress the copper (the quantity of sodium cyanide used per year was not determined). The molybdenum floats while the copper concentrate becomes underflow or "tailings" and is sent to drying and thickening prior to smelting. Overflow (the floated molybdenum concentrate) is sent on to three cleaner stages. Overflow from the last cleaner stage is filtered, then "agitated," before being subjected to five more "special" flotation or recleaner circuits. The percent of water filtered out during the recleaner stages was not determined. Overflow from each recleaner stage passes on to the next recleaner flotation step, while underflow is returned to the previous flotation cell for additional flotation. Overflow from the fifth recleaner circuit goes to the "splitter box" (not further described), where about 70 percent is sent to filtering and drying. The remaining 30 percent of the overflow from the last recleaner is returned to a filter at the beginning of the recleaner circuit. Filtering and drying produces a 95 percent molybdenum disulfide product, which is shipped offsite in 55-gallon drums and sold as molybdenite (Magma, 1990).

Copper Concentrate

Copper concentrate is dried in the "hydroseparator" (description not obtained), and dewatered to 10 percent water. The water removed in the filtering/thickening operation is returned to the mill, and the dried copper concentrate is placed on conveyor belts and transported to the flash furnace for smelting. The copper concentrate consists of 30 percent copper, 30 percent iron, and 30 percent sulfur and oxidizes easily. In both the dryer and on the conveyor, silica is added as flux for smelting. A total of 113 tpd of silica is used. The amount of copper concentrate produced and the complete water balance for the mill were not determined.

Slag Reprocessing

Magma is one of the few copper facilities that reprocesses copper slag for copper recovery. San Manuel's Outokumpu flash furnace generates a copper slag that contains high concentrations of copper.

As a result, slag from the flash furnace (copper concentrate 1.8 percent) and from the converters (copper concentrate 5 percent) is reconcentrated at the mill along with ore at a rate of 2,300 tons of slag per day. Beneficiation of slag recovers almost 90 percent of the copper values, and produces approximately 55.2 tpd of copper anode. (Magma, 1992c) Copper recovery from slag is dependent on the treatment of the material prior to beneficiation at the mill. Slag from both the flash furnace and the converters is transported to the slag cooling area for very gradual cooling. Slow cooling in the initial stages is imperative to allow pure particles to coalesce and crystallize. The initial cooling takes place in shallow unlined pits for 24 hours of air cooling, after which the slag is cooled with water for an additional 8 hours. The slag is then broken, crushed and transported to the mill for reflotation and recovery of copper. (See Figure 3-10)

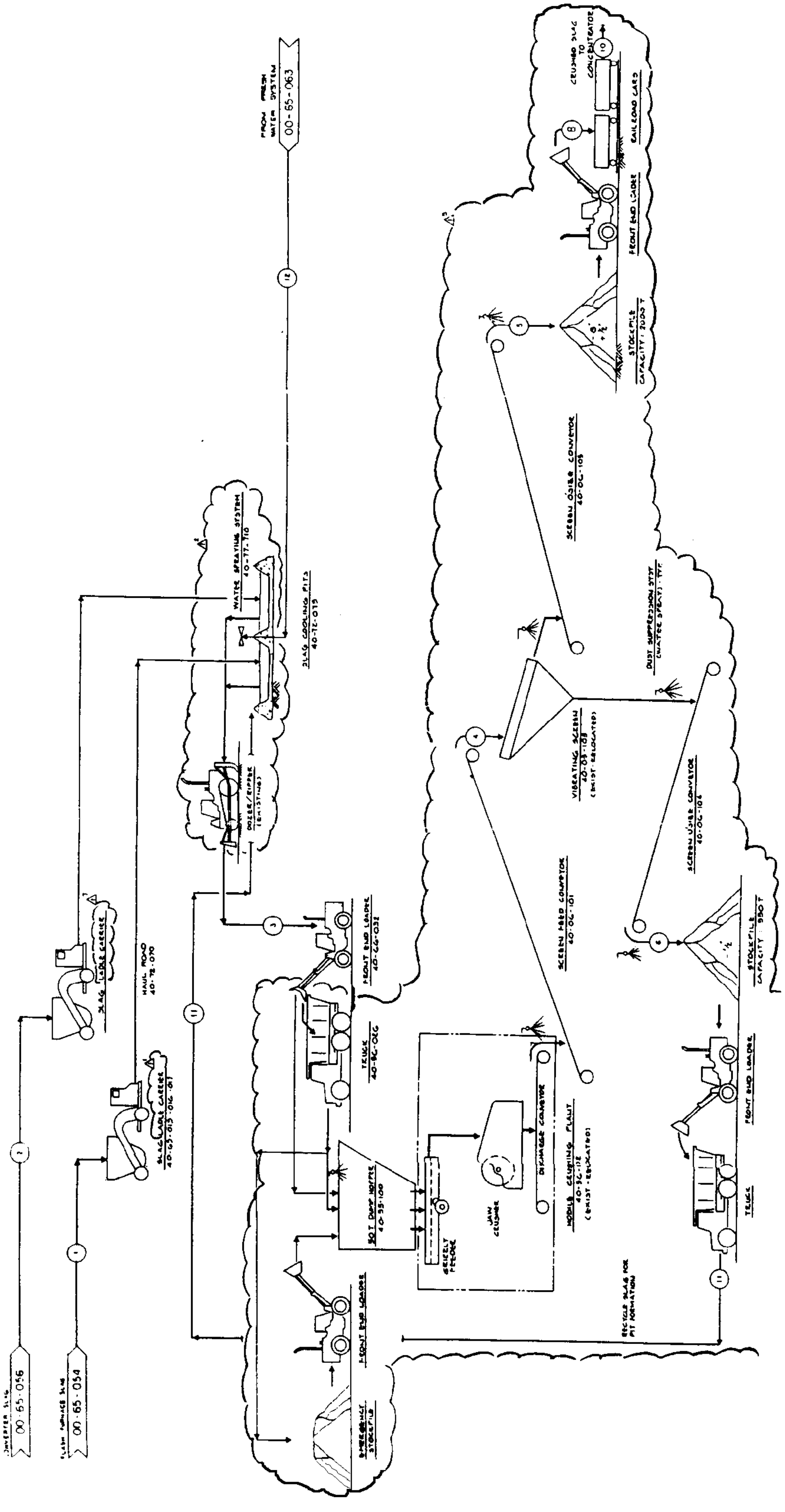


Figure 3-10. Slag Reprocessing

(Source: Magma, 1992c)

(Magma, 1992c)

Slag is transported to the cooling pits by Kress slag ladle carriers, rubber-tired, diesel-powered truck-like vehicles that deposit one ladle at a time into each pit to form a layer 20 to 24 inches thick. Once the slag is dry, bulldozers are used to break up the hardened material, and front end loaders place the ripped material into dump trucks for transport to a portable crushing plant with a 24-inch grizzly and then on to the mill. The slag is subjected to several stages of crushing and screening and sizing prior to being transported to the mill. In the concentrator, flotation yields copper concentrate which is returned to the flash furnace feed. (Magma, 1992c)

Because all slag is reprocessed, there is not permanent disposal of slag at the Magma site. Slag pits are used to allow cooling of the slag prior to crushing, flotation, and re-smelting. The slag pits are managed to avoid storm water runon or runoff. Magma does not foresee a problem with potential release or transport of slag constituents to the environment as the slag pits are located in an area 140 meters above the uppermost useable aquifer, and the San Pedro River is located over 790 meters away. (U.S. EPA, July 1990)

3.2.2.3 Processing Operations

Processing operations at San Manuel include the smelter, the associated acid plant, the electrolytic refinery, and the rod casting plant (See Figure 3-11)

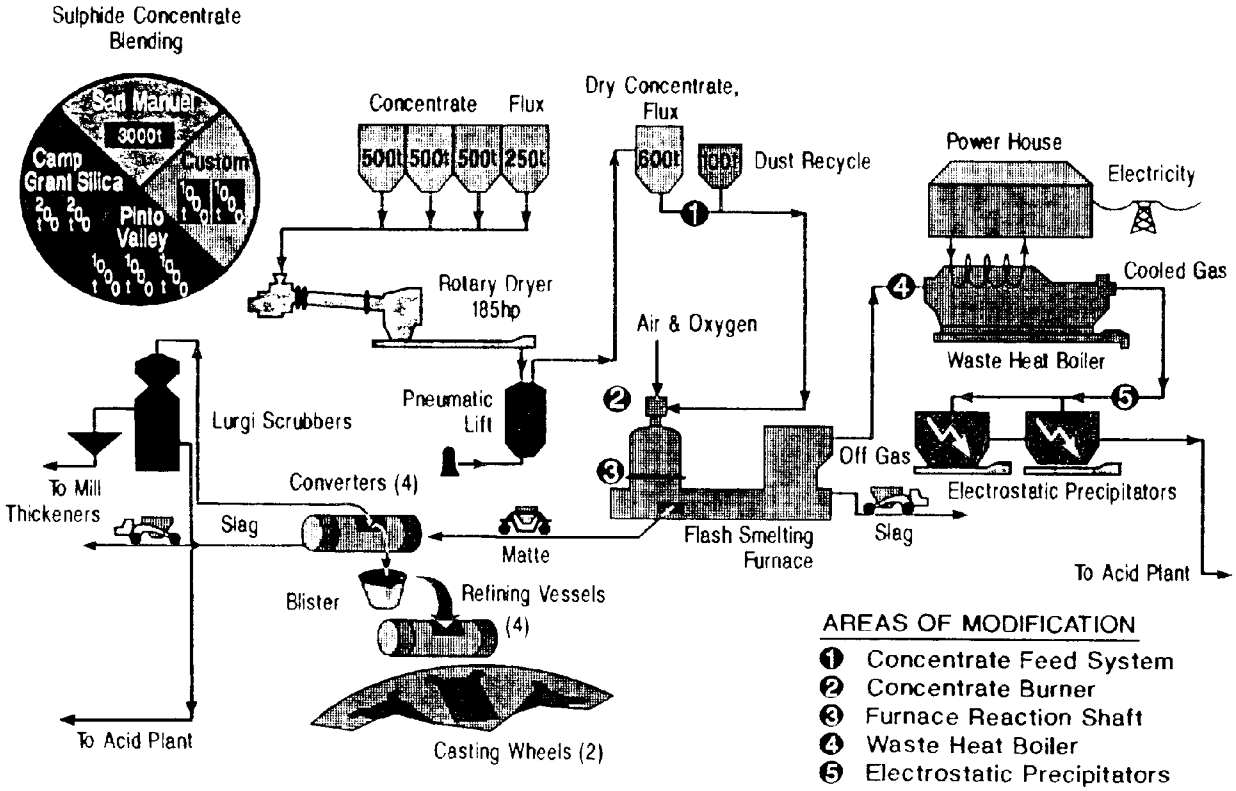


Figure 3-11. Smelter Circuit at San Manuel

(Source: Magma, 1992m)

. The San Manuel flash furnace is modeled after the Outokumpu design and has a processing capacity of 1 million tons of copper concentrate per year. During 1991, 703 million pounds of copper were produced by the smelter, using concentrate from San Manuel as well as concentrate from Superior, Pinto Valley, McCabe, and other mines. Approximately 520 million pounds of copper were produced from San Manuel ore.

Flash Furnace and Converters

The copper concentrate from the mill consists of 30 percent copper, 30 percent iron, and 30 percent sulfur. The reaction shaft of the smelter maintains a fire in the center of the stack initiated by natural gas, and concentrate is sprinkled down around the sides. The sulfides react with oxygen to create a flash that melts all the ingredients of the charge on their way down the 22-foot shaft into the settler. Gas containing dust and nearly 30 percent sulfur dioxide is transported through the uptake shaft into a waste heat boiler for cooling. The waste heat boilers remove heat from the gases for use in producing steam (Magma, 1988). The gases continue through the electrostatic precipitators and on to the acid plant, which converts the sulfur dioxide-rich gases to sulfuric acid (a usable and/or saleable product.)

Electrostatic precipitators recover molten dust particles contained in the flue gases. Flue dust from the flash smelter furnace contains up to 25 percent copper. Fugitive dust emissions are also collected from the converter. Fugitive dusts from the converter contain up to 80 percent copper. (Greenwald, 1992) The dust from all three sources is then combined with fluxed, dried concentrate and sent back to the smelter. (Weiss, 1985)

Copper matte from the smelter contains approximately 63 percent copper and is placed in ladles and transported to the "converter isle." The molten matte is poured into one of three converters, where further oxidation of sulfur and slagging of waste metals takes place over a period of seven to eight hours, until the matte reaches a purity of 99 percent copper. An estimated 1,390 tpd of matte is fed to the converters, including 113 tpd of flux (silica) and 135 tpd of scrap copper. Copper matte is poured into empty cells in the 12- to 15-foot diameter converter. From the converters, the molten copper, now called "blister," is transported to the casting department, where it is fire-refined for final removal of sulfur and oxygen before being poured into molds to produce 820 pound anodes for transport to the electrolytic refinery. (Magma, 1988)

Acid Plant

The double absorption sulfuric acid plant receives sulfur dioxide rich gases from the flash smelting furnace and converter furnaces (Magma, 1988). Approximately 45,600 standard cubic feet per minute (scfm), containing 26 percent sulfur dioxide (SO_2), is sent as off-gas from the flash furnace to the acid plant. The gas stream from the converter scrubber is also conveyed to the acid plant at a rate of approximately 70,000 scfm per converter. The acid plant cleans, dries, and converts SO_2 into saleable grade sulfuric acid by the addition of sulfur trioxide (SO_3) (Magma, 1988). Sulfuric acid is produced at 93 percent and 98 percent purity. Acid plant blowdown is generated and is transferred via a pipeline to a mixer tank, where the blowdown is combined with the tailings prior to being deposited in the tailings ponds. The amount of acid plant blowdown generated from the acid plant, and concentrations of its constituents prior to mixing with the tailings, was not determined.

Electrolytic Refinery

At the electrolytic refinery, the anodes from the smelter, along with 13-pound copper starter sheets, are placed in baths of an electrolyte made up of sulfuric acid and copper sulfate. There are 28 refining sections at San Manuel's refinery. Each refining section includes 42 lined concrete cells that hold 46 anodes and 45 copper starter sheets. An electrical current flows through the anodes and electrolyte to plate the copper from the anodes onto the starter sheets (while hindering co-deposition of impurities) over a 12-day period. (Magma, 1988) After 12 days, the cathode produced weighs approximately 365 pounds.

Several residues are generated in the electrolytic refining operation. Residue falls out of solution, settling to the bottom of the tanks. According to Magma, these residues are collected and leached to obtain their copper content, while precious metals-containing slimes are filtered and dried for offsite precious metals recovery. (Magma, 1988) Thirty-five thousand ounces of gold and 171,000 ounces of silver are produced per year in this process. The electrolyte is generally recycled, however, an acidic wastestream is produced during purification of the electrolyte that is mixed with tailings and discharged to the tailings impoundments.

Rod Casting Plant

From the electrolytic refinery, the cathode sheets are placed into a melting shaft at Magma's rod plant. The molten copper is drawn on a wheel around the shaft and fed into finishing roles with cutting solution (95 percent water, 4 1/2 percent soluble oil, and isobutyl alcohol). The rod is reduced down to 5/16-inch in diameter. The drawing process is also a continuous cooling process. The 5/16 diameter rods are sprayed with sulfuric acid from the acid plant to remove oxide copper and are covered with a fine wax coating before being sent to a continuous coiling machine and coiled at 6 miles per hour, or 36 to 40 tons per hour. Cooling waters and rinse solutions used at this plant are recycled to the refinery, the mill, or within the plant.

3.3 WASTE AND MATERIALS MANAGEMENT

3.3.1 Types of Wastes and Materials

Wastes managed onsite at the San Manuel facility include large volumes of waste rock, tailings, and other wastes from extraction and beneficiation operations, as well as wastes generated from processing operations. In addition, other materials, such as mine water, ore on leach pads, and leaching solutions, are managed onsite. Because these materials ultimately become wastes when intended for disposal (when operations cease), they are also addressed in this section.

This section emphasizes management of extraction and beneficiation wastes and materials, and the units in which they are managed, as well as areas where processing wastes and materials are commingled with those from extraction and beneficiation. Although processing is generally beyond the scope of this report, limited information on these wastes and materials will be discussed below in order to characterize the material balance throughout the facility.

3.3.2 Underground Workings, Open Pit, and *In situ* Leach Area

As described above, three separate operations take place at the mine site to extract and beneficiate copper minerals from sulfide and oxide ores at San Manuel. Because underground mining, open pit surface mining, and *in situ* leaching all occur in the same general vicinity, these units (pit, underground workings, etc.) and the wastes and materials they generate are discussed together in this section. Generally, underground methods mine the deeper area of the ore body (sulfide ore), while open pit methods are used to mine oxide ore at the surface with *in situ* operations recovering copper from areas in-between. As a result of underground mining, subsidence has intersected the open pit and *in situ* areas.

As described above, underground workings extend from 1,400 feet below the surface to the 4,080- foot level where dewatering operations occur. The workings cover an approximate areal extent of 2,000 by 3,700 feet (Magma, 1992h). (See Figure 3-4.) The main haulage level is currently at a depth of 2,875 feet.

Dewatering is of major importance to underground operations. Magma pumps approximately 3,440 gpm of water to the surface to keep the workings free of infiltration. Water generated from the mine is stored in a settling pond prior to being transported to the mill circuit (It was not determined whether this pond was lined). Though the original level of the water table in the area of the mine was not provided, dewatering operations at the time of the site visit were on-going at a depth of 4,080 feet. As of 1982, the water table elevation one-half mile to the south east had dropped 800 feet due to mine pumping (Magma, 1991). According to Magma representatives, dewatering operations have created an eight mile cone of depression in the water table. According to the Aquifer Protection Permit for the leach pad, the cone of depression covers approximately 11 square miles, surrounding the mine in all directions except to the south and southwest where recharge occurs. EPA did not obtain information regarding original groundwater levels, water elevation levels, or any changes that have occurred to these levels over the years.

Underground mining generates relatively little waste rock at San Manuel. According to Magma representatives, no waste rock from the underground is hauled to the surface. Additional details on the quantity and management of waste rock underground was not obtained.

Located in the subsidence area above the underground mine, the open pit has currently reached a depth of 800 feet, with an additional 400 feet remaining before reaching a final depth of 1,200 feet. The approximate areal extent of the open pit is 5,000 feet by 4,400 feet (Magma 1992i). During the site visit, EPA observed underground mining subsidence features at the edge of the pit and in the waste rock piled at an angle of repose into the pit. In addition, EPA observed *in situ* mining within the pit, showing the close proximity of activities at the mine site. During EPA's site visit, the pit appeared relatively dry. Open pit mining at San Manuel generates from 60,000 to 70,000 tons of waste rock per day with a stripping ratio (ore:waste rock) that ranges from 1:1.3 to 1:2. Waste rock is disposed of in one of several waste rock dump piles onsite. These are discussed in the Waste Rock section below.

Magma extracts copper from the ore in the areas between the open pit and the depleted portion of the underground mine using *in situ* leaching techniques. Approximately 272 million tons of ore rubbleized by underground block caving methods is subjected to *in situ* leaching (Magma, 1988). According to Magma, approximately 2 billion gallons of raffinate solution was held within the ore at the time of the site visit. The pregnant solution is collected in specific panels of the underground workings, or captured by submersible sumps, and pumped to the surface to the plant feed pond. Copper production from the *in situ* operation was not fully operational until the ore body was completely saturated (when this occurred was not determined). Initially, raffinate from the SX process was sent to the *in situ* wells at a flow rate of 8,000 gpm (gpm). The rate of PLS recovery in the underground panels measured 7,200 gpm, a solution loss of 800 gpm to surrounding ore and country rock (See Table 3-2). Current injection rates are 3,000 gpm, or 21,600 tpd. The number of injection wells used and the extent of the area subject to the *in situ* leaching operation were not determined.

According to Magma's APP for the Leach Pad, "because of the cone of depression, mine pumping removes any precipitation infiltrating the mine area, the recharge of groundwater passing through the area, all spillage, leakage or injection above the area, and ambient groundwater present within the mine. As long as Magma maintains this cone of depression, it is hydrologically impossible for ambient groundwater outside the mine to become impacted by mine operations." No information was obtained on actions that may be taken when the underground mine operations cease and dewatering stops.

3.3.3 Waste Rock

The open pit mine is the source of most of the waste rock generated at San Manuel (60,000 to 70,000 tons of waste rock per day). Approximately 20 million tons of waste rock have been generated each year. Based on that estimate, roughly 131 million tons of waste rock may be contained in the three waste rock piles on-site (based on 60,000 tpd, 365 days per year for 6 years) (See Figure 3-12)

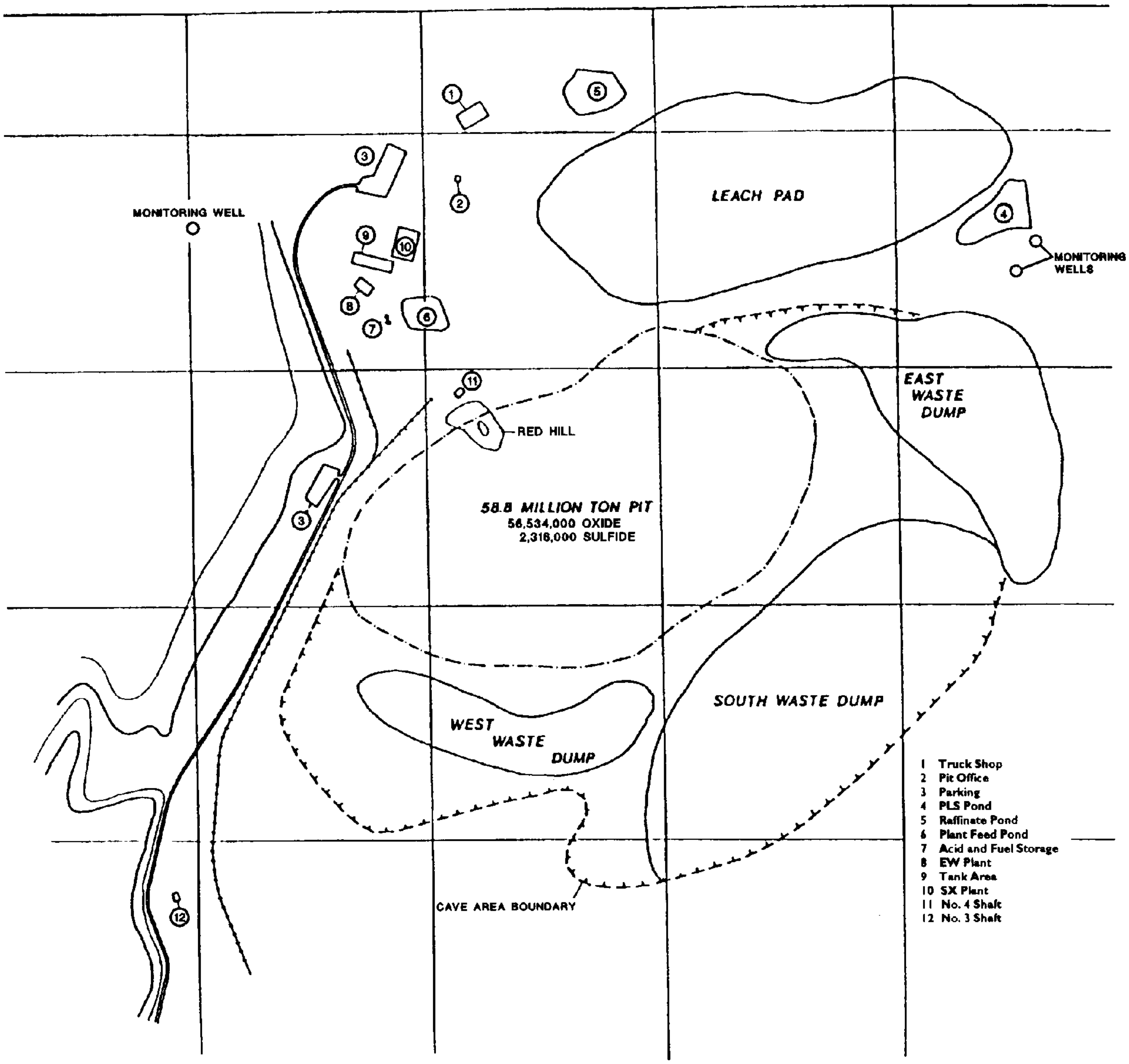


Figure 3-12. Waste Rock Piles

(Source: Helmer, 1988)

. Limited information on waste rock piles 1,3, and 4 was obtained. The Agency does not have data on waste rock pile number 2.

Waste dump no. 1 is located on the northeast edge of the pit. The pile is approximately 3,200 feet in length, with two lobes extending as far east as 1,900 feet from the pit (Magma, 1992i). According to Magma representatives, this was the original waste rock pile formed during the excavation of the pit. Magma reported that no waste rock had been deposited in this waste rock pile in over two years, as all waste rock generated in this time period had been used to backfill the subsidence area in the open pit. Waste dump no. 3, as labeled on Figure 3-12, measures approximately 1,200 by 500 feet and is also located southwest of the pit, opposite waste rock dump no. 4. (Magma, 1992i)

Waste dump no. 4 is located on the southwest edge of the pit and is currently active. Here, the waste rock is dumped back into the pit, and left to fall at an angle of repose inside the pit. Based on Figure 3-12, dimensions of this waste rock pile are approximately 300 feet by 2400 feet. No information was obtained on the acid generating potential, if any, of these waste rock piles, nor were details on specific runon and runoff controls or monitoring of the waste rock piles.

3.3.4 Dump/Heap Leach and SX/EW Plant

As discussed previously, the heap leach operation entails the beneficiation of oxide ore from the open pit by leaching copper with sulfuric acid to produce a pregnant leach solution, which is then sent to the SX/EW. This operation involves recirculation of a sulfuric acid solution through a series of tanks, ponds, and ore (both *in situ* and on the heap) to extract copper. Units associated with this operation include the lined heap and collection ditches, the lined pregnant leach solution and plant feed ponds, the raffinate pond, and the SX/EW plant. The raffinate (barren leach solution) is also used in the *in situ* mining operation discussed above. Although the materials generated are not wastes during the time leaching continues, they are discussed here because the toxic constituents that are handled in the operations and/or that occur in the materials present some potential for environmental contamination.

3.3.4.1 Heap Leach

The heap leach operation was designed as a no-discharge operation, with the heap, its collection ditches, the pregnant solution pond, and plant feed ponds all lined to prevent solution loss.

The heap has been constructed on a pad near the open pit mine. The heap leach pad is approximately 3,000 feet wide by 6,000 feet long, covering 230 acres (Magma, 1989). According to Magma representatives, the leach pad is underlain by Gila Conglomerate with a permeability of 3.3×10^{-7}.

Soils beneath the leach pad were compacted and a 7-1/2 ounce per square yard layer of geotextile material liner was installed prior to laying the leach pad. Initially, three thicknesses of HDPE liners were used to make up the original 113-acre leach pad: a 60 mil HDPE lined 71 acres; a 100 mil HDPE lined 17 acres; and a 40 mil HDPE lined the other 25 acres. Apparently, 60 mil liners were used over ridges of the heap, while 100 mil liners were used to line collection ditches. Eighteen inches of gravel were placed on top of the liner to prevent tearing and a collection system of perforated pipes was installed to prevent pooling. (State of Arizona, 1989) Run-of-mine oxide ore from the open pit is hauled to the heap leach at a rate of approximately 30,000 tpd. The heap is rising in 15-foot lifts. Current estimates by Magma state that the heap leach will reach a maximum height of 400 feet, (almost 100 feet higher than described in the 1989 Groundwater Quality Protection Permit).

After being collected in ditches along the side of the heap, the solution is sent to the PLS pond (5- million gallon capacity) and on to the plant feed pond (10-million gallon capacity). Both ponds are lined: the PLS with 80 mil HDPE overlying compacted subgrade material, and the plant feed with 60 mil XR5 reinforced fabric liner overlying compacted subgrade material. After being stripped of copper in the SX/EW plant, barren solution (raffinate) is sent to the raffinate pond (capacity 10 million gallons). The raffinate pond is lined with a 60 mil XR5 reinforced fabric liner overlying compacted subgrade material. Each of the three ponds is designed to contain a 100-year/six-hour storm event (3.4 inches of precipitation), in addition to working solution.

At heap closure (expected in 1996), the leached ore will remain on the pad (the amount was not determined). According to the APP, closure will entail rinsing of the heap by "natural meteoric events," with no cover or reclamation planned. Magma plans to collect the solution generated by natural meteoric events and send it to the SX/EW plant. The ponds and the SX/EW are scheduled to continue operating until closure of the entire facility, possibly for an additional 35 to 40 years, supplied in part with pregnant solution from the *in situ* mining operation. Specific details on life of pond liners or plans for replacing the liners were not obtained, nor were anticipated flow rates from the heap after application of lixiviant ceases. There are apparently no "trigger levels" established for determining the point at which leachate collection will no longer be required.

Although the heap leach operation was designed as a no-discharge operation (with the heap, its collection ditches, the pregnant solution pond, and plant feed ponds all lined to prevent solution loss), solution losses from the heap to undetermined sources (e.g. evaporation) during 1987 were estimated to be 276 gpm. The application rate of barren solution to the heap at that time was 9200 gpm. The 1989 application rate was 14,500 gpm. According to Magma, any solution released will not impact ambient groundwater during operation of the mine, due to mine pumping and the associated cone of depression (Magma, 1991-APP).

According to Magma representatives, the heap may periodically be used as a disposal area since most spills that occur at the mine site may be dug up and transported to the heap for leaching. Also, "gunk" generated at the solvent extraction facility is centrifuged and deposited on the heap for leaching of residual copper.

Magma Copper has received an Aquifer Protection Permit from the State (Arizona Department of Environmental Quality, ADEQ). Magma has accordingly monitored groundwater with one well, and the vadose zone with 2 wells. According to Magma personnel, the well is currently upgradient rather than downgradient of the heap due to the cone of depression caused by dewatering the mine. The rationale for this approach is not clear. Monitoring results and other details of this program are presented in Section 4 below.

3.3.4.2 Solvent Extraction/Electrowinning Wastes

As described above, solvent extraction is a two-stage operation that selectively removes copper from leach solution using a concentrated organic (kersosene), which is then stripped of copper by a strong electrolyte to prepare a higher grade copper solution for electrowinning. Electrowinning then recovers copper from the electrolyte by plating the copper onto lead anodes. The solvent extraction tanks are located outside of the electrowinning building and are covered with roofs to diminish evaporation of solutions in the open tanks.

All solutions (leaching, organic, and electrolyte) are reused in the operation. Electrolytic solutions decrease in effectiveness over time due to an accumulation of impurities. The method used to remove impurities from spent electrolyte and any wastes generated by this operation, were not determined. A monthly reagent use information sheet provided by Magma indicates that 32,000 gallons of organic are used per month. It is unclear whether this number refers to consumption or to the total amount used and recycled. A certain volume of organic is assumed to be lost to evaporation. The organic is directed through a centrifuge and clay filter to remove impurities prior to reuse. "Gunk" (the material removed by the centrifuge) and spent clay from the filter is placed on the heap for leaching of residual copper values. The quantity of gunk generated was not determined.

Silt may accumulate on the bottom of the tanks over time. According to Magma representatives, the silt is removed by vacuum. The volume of silt generated, and its method for disposal, were not determined; it may be deposited on the heap for additional leaching.

At the electrowinning facility, completed copper cathodes are washed with water to remove residual electrolyte from the sheets. The amount of wash water generated in this operation and its disposition were not determined. Waste lead anodes (quantities unknown) are sent to a Doe Run lead smelter. Information on the generation of sludge or other residues from the electrowinning facility was not obtained.

3.3.5 Tailings Impoundments

3.3.5.1 Tailings

At San Manuel, tailings are generated during the initial copper-molybdenum flotation circuits, prior to the separation of the molybdenum from copper in the moly-plant. Tailings are comprised of gangue (nonvaluable materials) from the ore, water, and remnants of reagents used during the flotation operation. Slag tailings are comprised primarily of silicon, iron, magnesium, sodium, smaller amounts of copper, lead and zinc, as well as other trace elements (U.S. EPA, 1990). According to Magma personnel, typical copper concentrates in the rougher tailings are .07 to .08 percent. An analysis of copper mill tailings prior to disposal into Impoundment no. 10 found the following concentrations of constituents (ppm): arsenic, 6; cadmium, 1.1; chromium, 6; lead, 11; copper, 298; cyanide, <5; mercury, <0.005; nitrogen, 10; sodium, 1,400; selenium, <0.05; and zinc, 1,000. (Magma, November 21, 1990) It is not clear whether the sample was obtained prior to or after the tailings reached the mixing area where Magma combines acid plant blowdown with the tailings. According to the Arizona DEQ, Magma's tailings flow through the mixing area at a rate of approximately 1600 gpm with approximately 50 percent solids; acid plant blowdown is combined with the tailings at a rate of approximately 400 gpm.

Tailings are generated at each of the eight flotation circuits in the concentrator and are fed to a hydroseparator. Underflow from the hydroseparator is sent to a repulper and on to the tailing distributor to various tailings dams. Overflow of the hydroseparator is directed to the tailings thickeners, where water is removed for reuse during flotation, and underflow is sent to the repulper and on to tailings distribution (Magma, 1990). Slag tailings are co-generated with ore tailings, making up approximately 0.3 to 2.6 percent of the total tailings managed in the ponds (U.S. EPA, 1990).

3.3.5.2 Tailings Impoundment Construction and Reclamation

Thickened tailings (50 percent solids) are transported via a pipeline, flowing at an unspecified rate, from tailings thickeners to one of seven tailings impoundments which cover a total of 3,600 acres. Five of the seven tailings ponds are currently active. The two inactive ponds have a 50-million ton capacity still available. Magma reclaims water from the tailings impoundments, although the volume (and proportion) of water that is reclaimed to the mill was not determined. The tailings pond area is located west of the San Pedro River, with Tailings Impoundment 10 reaching to within one-half mile of the River. There have been at least three spills of tailings from the impoundments. These are described in Section 4.

At the unlined tailings impoundments, several hundred hydrocyclones are used to separate the coarse from fine tailings during tailings deposition. These devices are spaced at 52-foot intervals along the berms of the tailings impoundments. The coarse fraction is used to construct the berms and the finer material is pumped further into the impoundments. Due to the natural slope of terrain upon which the tailings impoundments were built, from the mill facility down to the River, the impoundments vary in height at their upper ends from 0 to 150 feet at the dams (US EPA, 1986, Weiss, 1985). (See Figure 3-3) Their ultimate height was not determined.

In constructing the tailings impoundments, starter dams were keyed into the bedrock for stability. The key-ways were built at right angles to the berm and cut to an unspecified depth, until they reached solid ground.

The key-ways were filled with sand and clays from the surrounding areas, with the remainder of the starter dams built using alluvial materials. In order to prevent storm water runoff from impacting the capacities of the impoundments, Magma diverted major washes located upslope in the area around the impoundments. For instance, Courthouse Wash was diverted around tailings impoundment no. 10. The mill division inspects the tailings systems regularly to identify and correct any problems associated with its operations (Magma, 1992).

There is no formal groundwater monitoring program currently in place to monitor groundwater in the area of the tailings impoundments. As discussed previously, there are 11 water supply wells installed at the foot of the tailings impoundments (See Figure 3-3). Water from these wells has been shown to contain elevated fluoride levels. However, no information on the source of the high fluoride levels was obtained. According to Magma representatives, vegetation density has increased in the area between the tailings impoundments and the river since the 1950's.

3.3.5.3 Tailings Reclamation

Although not required by any State or Federal law, Magma has initiated a tailings reclamation program in response to public complaints regarding windblown tailings in the area. Magma is trying different reclamation techniques on portions of impoundments 1 and 2. These impoundments cover 660 acres and have been partially covered with native soils (Gila conglomerate) and vegetation. Approximately 420 acres have been reclaimed using several reclamation techniques, with soil capping proving to be the most successful. A total of 230 acres of flat tailings and 115 acres of sloped tailings have been capped with borrow material. Approximately 70 acres of raw tailings are being used as a test plot. [For soil capping and hydroseeding, the cost per acre was $2,420. The revegetation of raw tailings cost $1,217 per acre. The total cost to reclaim 260 acres was $412,000. To stabilize an additional 160 acres at tailings impoundments 1 and 2, the cost is estimated to be $463,598 (Magma, 1992j). Once seeded, the soil cap and growth of vegetation on the tailings has minimized erosion. Sewage effluent and sludge are also applied to the tailings near the reclamation areas and appear to have had a positive effect on plant growth.

3.3.5.4 Co-mingling of Wastes in the Tailings Impoundments

Acid Plant Blowdown

In addition to tailings from the mill, Magma disposes of other wastes in the impoundments. At San Manuel, acid plant blowdown is mixed with tailings and discharged to the tailings impoundment. Magma indicated that mixing with the tailings serves to dilute or neutralize the acid plant blowdown before it enters the tailings impoundments. Acid plant blowdown is discharged from a pipe to a cement tank-like structure and allowed to mix with the tailings as the mixture flows through pipes to the tailings impoundments. According to the Arizona DEQ, Magma's tailings flow through the mixing area at a rate of approximately 1600 gpm with approximately 50 percent solids; acid plant blowdown is combined with the tailings at a rate of approximately 400 gpm.

Refinery Wastewater

According to Magma representatives, tailings are also used to neutralize acidic wastewater generated at the electrolytic refinery. At the electrolytic refinery, anodes produced from blister copper are placed in baths of acidic copper sulfate, and a current is applied that deposits copper from the anode onto the cathodes. Impurities from this process either dissolve in the electrolyte or fall to the bottom of the tank to form a slime. These slimes consist of suspended materials that may include precious metals. San Manuel manages this residue by sending the slimes to precious metals recyclers for recovery of the values. An acidic waste stream of process wastewater is also generated at the electrolytic refinery. This waste stream is added to the tailings slurry and recycled in the mill water reclaim system. The amount of acidic process wastewater generated from the electrolytic refinery was not determined and no information was obtained describing its characteristics.

Other Wastes

Magma has set aside an area on the northwestern edge of tailings ponds 3 and 4 for disposal of acid or fuel oil-contaminated soils from on-site spills. Under the Clean Water Act, San Manuel developed a Spill Prevention Control and Countermeasure (SPCC) Plan to reduce the potential for discharge of oil at San Manuel (Magma, 1992g). A 1990 SPCC Plan provided by Magma describes 11 oil or fuel spills that occurred between October 1989 and October 1991. Wastes generated during clean up of these spills were disposed of in the tailings impoundment or sent to asphalt contractors for use as input into an asphalt plant. (Magma, 1992g; Deming, 1989)

3.3.6 Other Waste

3.3.6.1 Landfill

The San Manuel mine site includes a landfill. No information was obtained on the dimensions of the landfill or the volume or types of materials disposed of there. According to Magma representatives, labeled and bagged asbestos waste was disposed at the landfill. The asbestos was buried and the landfill was expanded to cover the area of burial.

3.3.6.2 Sewage

Magma operates a sewage treatment plant for the plant site and for the town of San Manuel. Wastewater from this system is discharged to the tailings ponds for evaporation. According to the 1989 Groundwater Quality Protection Permit, discussed in more detail below, domestic sewage has to be disposed of in existing septic tanks and leach trenches. Sewage wastewater from the town of San Manuel is discharged to the tailings impoundments; the amount was not determined.

3.3.7 Waste Minimization

Because San Manuel is an integrated facility, Magma recycles and reuses many materials onsite or sends them offsite to recover material values. Table 3-3 provides a list of the materials at San Manuel that are managed to minimize disposal.

Table 3-3. Waste Minimization at San Manuel

MATERIAL GENERATED	MANAGEMENT TECHNIQUE
Chromic Acid	Use as a water treatment in cooling towers was discontinued.
Empty Drums	Drums that contained hazardous waste are recycled through an approved facility.
Lead Sulfate Sludges/Batteries	Sent offsite to be recycled.
Mercury Bearing Equipment	Is in the process of being replaced.
Paper	Recycled.
PCB Transformers	Retrofitted and replaced at the mine and plantsite.
Process Waters	Reused and recycled.
Rinsate from Oxide Truck Shop	Installed a new water/oil/particulate separation unit to process rinsate from shop floor to make useable products.
Solvents	Magma facilities use only recyclable solvents such as "Safety-Kleen" or non-chlorinated solvents.
TCE	Replaced by CS-187 as a parts cleaner in the locomotive shop. (CS-187 is a non-regulated, non-toxic cleaning solvent.)
Tires	Magma is investigating the possibility of recycling old tires for energy recovery.
Unused Chemical Reagents	Re-sold.
Vanadium Pentoxide (from acid plant)	Recycled.
Waste Oil and Greases	Collected and recycled or blended for use as fuel to produce steam and electric power, or used as fuel in the Plant Powerhouse.
Waste Paints	Transported to cement kilns for use as secondary fuels.

(Source: Magma, 1992e)

3.4 REGULATORY REQUIREMENTS AND COMPLIANCE

The San Manuel facility is subject to both Federal and State regulatory requirements and their attendant permits. A list of Magma's permits for the San Manuel facility is presented in Table 3-4. Only the Groundwater Quality Protection Permit and the Aquifer Protection Permit for the Heap Leaching Facility, both issued by the State of Arizona, were examined during preparation of this report. San Manuel is located on private and State land; as no Federal land is involved, Federal land-managing agencies (e.g., Forest Service, Bureau of Land Management) do not have jurisdiction over the facility. The majority of requirements and permits are implemented by the State of Arizona and are described below.

3.4.1 Federal Permits

3.4.1.1 Clean Water Act

NPDES Permit

EPA has not delegated Federal authority to the State of Arizona to implement the requirements of the NPDES program under the Clean Water Act. Although there have been NPDES issues at the site, EPA Region IX has not issued a NPDES permit for the San Manuel facility. In December 1987, EPA conducted an inspection of the site to determine if the facility required an NPDES permit to control surface mining and process water discharges to area dry washes and to the San Pedro River. No discharges from the ponds at the heap leach operation were found. However, evidence of a discharge of solution from the heap was noted. The discharge area was discolored and eroded and extended to the Little Mammoth Wash. EPA Region IX recommended that Magma apply for an NPDES permit to cover the above described discharge as well as discharge of process water to Tucson Wash from a reservoir under construction at the time of inspection. Magma applied for two discharge permits under NPDES on February 16, 1988. According to Magma representatives, the area impacted was covered with an additional phase to the heap leach. According to Region IX, San Manuel's NPDES permit is still in draft form.

Storm Water Group Application

Under the NPDES Storm Water Final Rule (55 FR 47990), active and inactive mine facilities (SIC codes 10-14) discharge storm water associated with industrial activity. As such, mines are required to obtain an NPDES permit that covers contaminated storm water discharges.

Table 3-4. Environmental Permits for the San Manuel Facility

TYPE OF PERMIT	AREA AFFECTED	PERMIT NUMBER	ISSUING AGENCY
Air	Mine/mill	0350-87	AZ OAQ
Air	Smelter	0355-88	ADEQ
Surface Water (NPDES)	Mine	AZ0023191	ADEQ/EPA
Groundwater (NOD)	Mine	Magma file #SMW-002	ADEQ
Groundwater (GPP)	Mine	G-0058-11	ADEQ
Groundwater (NOD)	Plant	Magma file #SMW-004	ADEQ
Groundwater (NOD)	Townsite	Magma file #SMW-005	ADEQ
Groundwater (NOD)	Old Reliable Mine	Magma file #SMW-006	ADEQ
Potable Groundwater	Mine	11-347	ADWR
Radioactive Material	Entire property	Magma file #SMR-001	AZ Radiation Regulatory Agency
Radiation	Mill	Reg No. 11-I-3539	AZ Radiation Regulatory Agency
Radioactive Material	Entire Property	License 11-2	AZ Radiation Regulatory Agency
Ionizing Radiation Machines	Entire property	11-I-3539	AZ Radiation Regulatory Agency
Hazardous Waste	Entire property	AZD001886597	EPA
Air	Hospital	6909	Pinal Co. AQCD
Air	Hospital	6910	Pinal Co. AQCD
Air	Hospital	6911	Pinal Co. AQCD
UST	Mine	0-003023 1	ADEQ
UST	Mine	0-003023 10	ADEQ
UST	Mine	0-003023 11	ADEQ
UST	Mine	0-003023 2	ADEQ
UST	Mine	0-003023 3	ADEQ
UST	Mine	0-003023 4	ADEQ
UST	Mine	0-003023 5	ADEQ
UST	Mine	0-003023 6	ADEQ
UST	Mine	0-003023 7	ADEQ
UST	Mine	0-003023 8	ADEQ
UST	Mine	0-003023 9	ADEQ
Wastewater Treatment Operator License	Mill	WW03857	ADEQ
Water Operator License	Mill	WW03633	ADEQ
Distribution System Operator License	Mill	WW03633	ADEQ

(Source: 1991 APP)

According to Magma representatives, San Manuel was included on the Part I Storm Water Group Application submitted to EPA by the American Mining Congress (AMC). However, AMC's group application, identification code number 570, did not include the San Manuel site. No further information was obtained.

3.4.1.2 Resource Conservation and Recovery Act

EPA has issued to Magma a hazardous waste identification number, Number AZD001886597, which addresses the entire property. According to the Arizona DEQ, Magma is a large quantity generator. The facility does not have a RCRA treatment, storage and disposal permit.

3.4.2 State of Arizona Permits

The Arizona Department of Environmental Quality's Office of Water Quality and Office of Air Quality are responsible for issuing various permits applicable to mining operations. The Office of Water Quality is responsible for administering Arizona's Water Quality Control Law, a part of Arizona's Environmental Quality Act (AEQA, 1986), and its applicable permitting programs. The aquifer protection permit program, administered by the Office of Water, was established to regulate point source discharges to groundwater. This office previously issued Groundwater Quality Protection Permits (GWQPP), the predecessor of the APP. The Office of Water Quality also conducts inspections for compliance with the GWQPP and the APP, as well as handling violations of surface and aquifer water quality standards and other water quality rules.

3.4.2.1 Groundwater Quality Protection Permit (GWQPP)

In 1989, Magma was issued a Groundwater Quality Protection Permit (Number G 0058-11) for the operation of its San Manuel Mine by ADEQ. The permit states that it is "valid for the operational life" of the facility, provided that no conditions of the permit or of the applicable groundwater quality standards and aquifer water quality standards are violated. (The GWQPP was the predecessor to the APP. Magma applied for the APP in 1991, as described below).

The permit requires the facility and later expansions of the facility to be constructed and maintained to prevent discharge of pollutants to the land surface or sub-surface which may adversely impact groundwater quality. The facility and later expansions must include the following: the use of the heap leach process, components of which include an impervious lined leach pad; impervious lined solution collection channels; three (3) impervious lined solution ponds; leach solution transmitting lines; solvent extraction plant; tank farm; and electrowinning plant.

The GWQPP also contains monitoring requirements and record-keeping requirements, including the monitoring of leach solution used in the hydrometallurgical heap leach process, sampling, and analysis on an annual basis of the pregnant leach solution (PLS) drainage into the PLS for major cations, anions, total dissolved solids (TDS), and trace elements. Raffinate and PLS were to be sampled and analyzed for specified substances daily. Records pertaining to these monitoring activities and results and the daily levels of solution in the PLS Pond, the plant feed pond, and the raffinate pond were not obtained.

3.4.2.2 Aquifer Protection Permit (APP)

On February 20, 1991, Magma applied for an Aquifer Protection Permit for the mine site. The permit was issued on June 3, 1991 and applies to the existing heap leach and SX/EW facilities, to a 35.5-acre expansion of the heap leach and to realignment of raffinate and preconditioning solution transmitting pipelines. A modification to this APP was requested, and granted, in 1992, to increase the heap leach expansion an additional 10.5 acres. At the time of the site visit, Magma was in the process of completing their APP application for the remaining units at the site that were not included in the APP. Operations to be included in the upcoming application include the tailings impoundments, the open pit mine, the *in situ* injection holes, the underground sulfide mine, and surface facilities (such as ore crushing operations and assorted shops). According to the Arizona DEQ, this information was due to the State on December 1, 1992.

Under the Aquifer Protection Permit Rules, promulgated under the AEQA, any person who owns or operates a facility that discharges must obtain an individual APP. Mining facilities are considered discharging facilities and are subject to permitting if the sites operate with any surface impoundments, solid waste disposal, injection wells, tailings piles or ponds, leaching operations, wastewater treatment facilities, addition of pollutants to underground caves or mines, or any point source discharges to navigable waters. Mining-related activities exempt from being subject to this permit include mining overburden returned to the excavation site, which has not been treated with any chemicals or leachate agent, and mines that closed prior to January 1, 1986 (State of Arizona, APP Rules, 1989).

Design and Performance Standards

Arizona's APP sets requirements for Best Available Demonstrated Control Technology (BADCT) at permitted facilities. At San Manuel, the heap leach pads, ponds and solution ditches, storage tanks, and electrowinning tanks all meet BADCT requirements under the permit. For the heap leach pad, 80 mil HDPE liners must be covered with 18 inches of protective gravel. Geotextile liners must also be installed in areas used for internal drainage, and underneath the toe berms. Expansions of the heap pad must include preparation of the subgrade to a range of maximum dry density from 90 percent in areas that receive less leach dump overburden, to 95 percent maximum dry density in areas where the heap height will be more than 100 feet. All leach pad seams must be oriented to minimize stress in the joint areas, and must be tested to demonstrate impermeability. Outer collection ditches are constructed of 80 mil HDPE liner over compacted subgrade as well.

The PLS, plant feed, and raffinate ponds have all been lined using standards of preparation for their liners similar to those used to prepare the heap leach. In addition, berms were constructed around these ponds to prevent precipitation runon into the ponds. The electrowinning cells are constructed of concrete with a flexible PolyVinyl Chloride (PVC) liner protected by rigid PVC buffer sheets. All process solution storage tanks must be made of compatible materials. Proposed BADCT technology for the rest of the facility had not been submitted at the time of the site visit.

Monitoring

The Aquifer Protection Permit requires monthly monitoring reports. The reports cover the following topics: daily process solution monitoring at the PLS Pond, Plant Feed Pond, and Raffinate Pond; monthly vadose zone monitoring at certain wells; and facility inspection of operating components integrity and pond solution levels on a daily or weekly basis.

San Manuel's Aquifer Protection Permit (APP #100421) requires groundwater monitoring at the mine site. A monitoring well, Con E-3, scheduled for construction in February 1992, is located at the northeast side of the heap leach pads and is sampled once a month for primary and secondary drinking water standards. The well was drilled to a depth of 1,000 feet, with a design pump capacity of 10 gpm. During underground mining operations, dewatering activities leave the Con E-3 well hydrologically upgradient of the leach pad facility. According to Magma, at closure of the mine, and subsequent cessation of dewatering activities, changes in the direction of groundwater flow may occur so that Con E-3 will be downgradient of the heap leach facility. Magma does not feel the change in groundwater flow will impact water quality since the heap leach facility units are lined.

The Con-E well was the original monitoring well installed at the heap leach site that collapsed during bailing operations in May 1991 (Date of original installation was not obtained). Magma attempted to drill the Con-E2 well but had not reached sufficient depth to allow groundwater sampling as of January 1992. Magma notified the Arizona Department of Water Resources that a new Con-E3 well was scheduled for installation in the same area as the original Con-E well for the first quarter of 1992. According to Magma representatives, the monitoring well was installed at the time of EPA's site visit.

The Aquifer Protection Permit also requires monitoring of two vadose wells, numbers 8 and 9, located south/southeast of the pregnant leach solution pond to detect any seepage from the pond. The wells are 150 feet deep and are monitored on a monthly basis. It was not determined if these vadose wells have ever detected seepage from the PLS pond.

Closure

The APP requires notification of any intent to temporarily or permanently cease operations. For permanent closure, Magma must submit a closure plan to ADEQ for approval within 90 days of the date of notification. The plan must include, among other things, a description of methods used to treat remaining materials at the site as well as an explanation of methods used to control discharge of pollutants from the facility. It is unclear if the closure requirements would address such issues as the potential impacts of rising water levels in the underground mine, *in situ* leaching area, open pit, spent ore on the heap pad, and waste rock piles, after the sump pumps in the mine are turned off.

3.4.2.3 Air Permits

Magma has one air quality permit that was issued by the State for the entire facility. This permit was not examined by the site visit team. According to Magma, during copper blows at the smelter, there is continuous monitoring from the stacks of gases released to the air. There is a 650 ppm SO_2 limit from the stacks and an hourly limit rate of 18,275 pounds. Ninety-eight percent of SO_2 generated is captured by the hooded and wet lurgi scrubbers for transport to the acid plant. Additional details were not obtained.

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Woolfe, B. 1988. Correspondence from Brian R. Woolfe, Magma Copper Company to ADEQ. Re: W-4-1, IX-FY88-19 "Finding of Violation and Order for Compliance" Dated January 15, 1988 and Magma's Partial Response Dated February 11, 1988; February 16, 1988.

APPENDIX 3-A

COMMENTS SUBMITTED BY ARIZONA STATE REPRESENTATIVES (ADEQ) ON DRAFT SITE VISIT REPORT AND EPA RESPONSES

Arizona Department of Environmental Quality (ADEQ) representatives provided a copy of the draft site visit report on which they had made comments and corrections. A copy of the marked-up draft is not reproduced here for brevity's sake. In general, ADEQ's comments were clarifying in nature, providing information that the draft report indicated had not been obtained during the site visit or correcting minor factual errors in the draft. EPA has revised the report to incorporate all of the comments and suggestions made by the ADEQ. In some cases, EPA made minor changes to wording suggested by ADEQ in order to attribute the changes to ADEQ or to enhance clarity.

MINE SITE VISIT:

COPPER RANGE COMPANY
WHITE PINE MINE

U.S. Environmental Protection Agency
Office of Solid Waste
401 M Street SW
Washington, DC 20460

4.0 MINE SITE VISIT: COOPER RANGE COMPANY WHITE PINE MINE

4.1 INTRODUCTION

4.1.1 Background

The Environmental Protection Agency (EPA) has initiated several information gathering activities to characterize mining wastes and management practices. As part of these ongoing efforts, EPA is gathering data by conducting visits to mine sites to study waste generation and management practices. As one of several site visits, EPA visited Copper Range Company's White Pine mine in White Pine, Michigan, on May 5 and 6, 1992.

Sites to be visited were selected to represent both an array of mining industry sectors and different regional geographies. All site visits have been conducted pursuant to RCRA Sections 3001 and 3007 information collection authorities. When sites have been on Federal land, EPA has invited representatives of the land management agencies (Forest Service/Bureau of Land Management). State agency representatives and EPA regional personnel have also been invited to participate in each site visit.

For each site, EPA has collected information using a three-step approach: (1) contacting the facility by telephone to get initial information, (2) contacting state regulatory agencies by telephone to get further information, and (3) conducting the actual site visit. Information collected prior to the site visit is then reviewed and confirmed at the site.

The site visit reports describe mining operations, mine waste and material generation and management practices, and regulatory status on a site-specific basis and are based on information gathered from State and Federal agency files, as well as observations made during the site visit. This report focuses on extraction and beneficiation activities at the site and includes only a limited discussion of mineral processing at White Pine. In preparing this report, EPA collected information from a variety of sources, including the Copper Range Company, the Michigan Department of Natural Resources, and other published sources. The following individuals participated in the Copper Range Company site visit on May 5 and 6, 1992:

<u>Copper Range Company</u>

C. Craigie Balfour, Smelter, Superintendent	(906) 885-5111
Bill Maksym, Environmental Engineer	(906) 885-5111
George Miskovich, Surface, Safety Engineer	(906) 885-5111
Bob Mitten, Director, Environmental Affairs/Safety	(906) 885-5111
Curt Ahonen, Training Supervisor	(906) 932-6057
Mike Rusesky, Mill Metallurgist	(906) 885-5111
Ronald Woody, Manager, Process Operations	(906) 885-5111

Michigan Department of Natural Resources

C. Robert Reska, Jr., Geologist, Geological Survey Division	(517) 334-6976
Hank Switzer, Environmental Engineer, Waste Management Division	(906) 228-6561

U.S. Environmental Protection Agency

Matthew Straus, Director, Waste Management Division	(703) 308-8414
Stephen Hoffman, Chief, Mining Section	(703) 308-8413
Scott Ellinger, Geologist	(703) 308-8410

Science Applications International Corporation

Jonathan Passe, Regulatory Analyst	(703) 821-4831
Ron Rimelman, Chemical Engineer	(703) 821-4861

4.1.2 General Facility Description

Copper Range Company's White Pine mine is located near White Pine (approximate population 1,200) in Ontonagon County, Michigan, on the Upper Michigan Peninsula. The company's mill and smelter are six miles south of Lake Superior. The facility's tailings impoundments extend north from the mill toward Lake Superior, with the downstream end of one impoundment located 2.5 miles from the lake.

The surface area owned by the Copper Range Company covers approximately 64,000 acres; surface facilities cover approximately 360 acres. The underground mine covers an area of about 25 square miles and extends from the surface to a maximum depth of 2,700 feet. Basic facility operations include underground mining, underground primary crushing, above-ground secondary and tertiary crushing, grinding in rod and ball mills, flotation, filtering, drying, smelting, and electrolytic refining.

The Nonesuch formation, which contains the White Pine ore body, was discovered in 1865 and first mined from 1879 to 1881. More development by the owner, Calumet & Hecla, led to additional mining (operating from 1913 to 1920) and establishment of the White Pine townsite. A drop in copper prices closed the mine in 1920 and led to Copper Range Company's purchase of the property in 1929 at a sheriff's sale (Copper Range Co. Undated).

Several years of testing proved that the previously "worthless" chalcocite, by-passed earlier in favor of native copper, could be mined and milled at a profit. With a $63 million loan from the Reconstruction Finance Corporation, construction of the new "White Pine" project began in March 1952, with the first ore mined a year later. The first copper product was generated in January 1955. In 1977, Copper Range Company was purchased by the Louisiana Land and Exploration Company. The new facility owners built a state-of-the-art refinery, costing $78 million. Construction of the new refinery was completed in 1981 (Copper Range Co. Undated).

Adverse economic conditions in the copper industry led to the gradual shutdown of the White Pine operation; the mine, mill, and most of the smelter closed in 1982. However, the company continued smelter operations to produce copper from scrap material until 1984, when the facility completely shut down (Copper Range Co. Undated).

In 1983, three officers of Copper Range Company began negotiating with Louisiana Land on the possible purchase of the White Pine mine. Echo Bay Mines, Ltd., purchased the White Pine operation in January 1985 and negotiations continued with Echo Bay. The three officers were successful and went on to form the new "Copper Range Company." Operations at the new White Pine began in November 1985 (Copper Range Co. Undated).

In May 1989, Copper Range Company was purchased by Metall Mining Corporation, a Canadian-based international mining company with interests in copper, zinc, lead, gold, and silver (Copper Range Co. Undated). As of January 1, 1992, 88 percent of the mine was owned by Metall and 12 percent by White Pine employees through their Employee Stock Ownership Plan. According to the conditions of the Plan, White Pine employees will eventually hold a maximum of 20 percent ownership of the mine (Skillings' Mining Review 1992).

The high employment mark of 3,140 people at the mine was reached in 1974. Employment at the facility in 1991 was 1,110 and, at the time of the EPA site visit, 1,070 employees worked at the facility (600 of whom work underground). The facility operates 24 hours per day, 7 days per week. Primary mining activities are conducted on all shifts, blasting occurs between shifts, and most support activities are conducted during the day shift.

Between 1955 and 1988, a total of 3.3 billion pounds of copper was produced at the White Pine property. Approximately 46,000 tons of cathode copper and one million ounces of silver were produced in 1989. According to Copper Range Company personnel, a 1988 study found that the mine had approximately 40 years of ore reserves remaining. This estimate, however, is highly dependent on copper prices.

4.1.3 Environmental Setting

Figure 4-1. Location of Copper Range Company's White Pine Mine

(Source: Copper Range Co. Undated)

The Town of White Pine (approximate population 1,200) is located one mile from the mine site. White Pine is about 860 feet above sea level and 260 feet above Lake Superior. Land use in the immediate area is generally limited to mining and residential activity. The location of Copper Range Company's White Pine mine is shown in Figure 4-1 (Copper Range Co. Undated).

According to Copper Range Company personnel, no rare or endangered species are found at the site, although eagles have been spotted flying over the facility. Copper Range Company personnel noted that Peregrine Falcons were reintroduced to the area nine miles south of White Pine. Previously, the State successfully reintroduced Canada Geese on Copper Range Company land, with the involvement of Copper Range Company personnel, equipment, and funds.

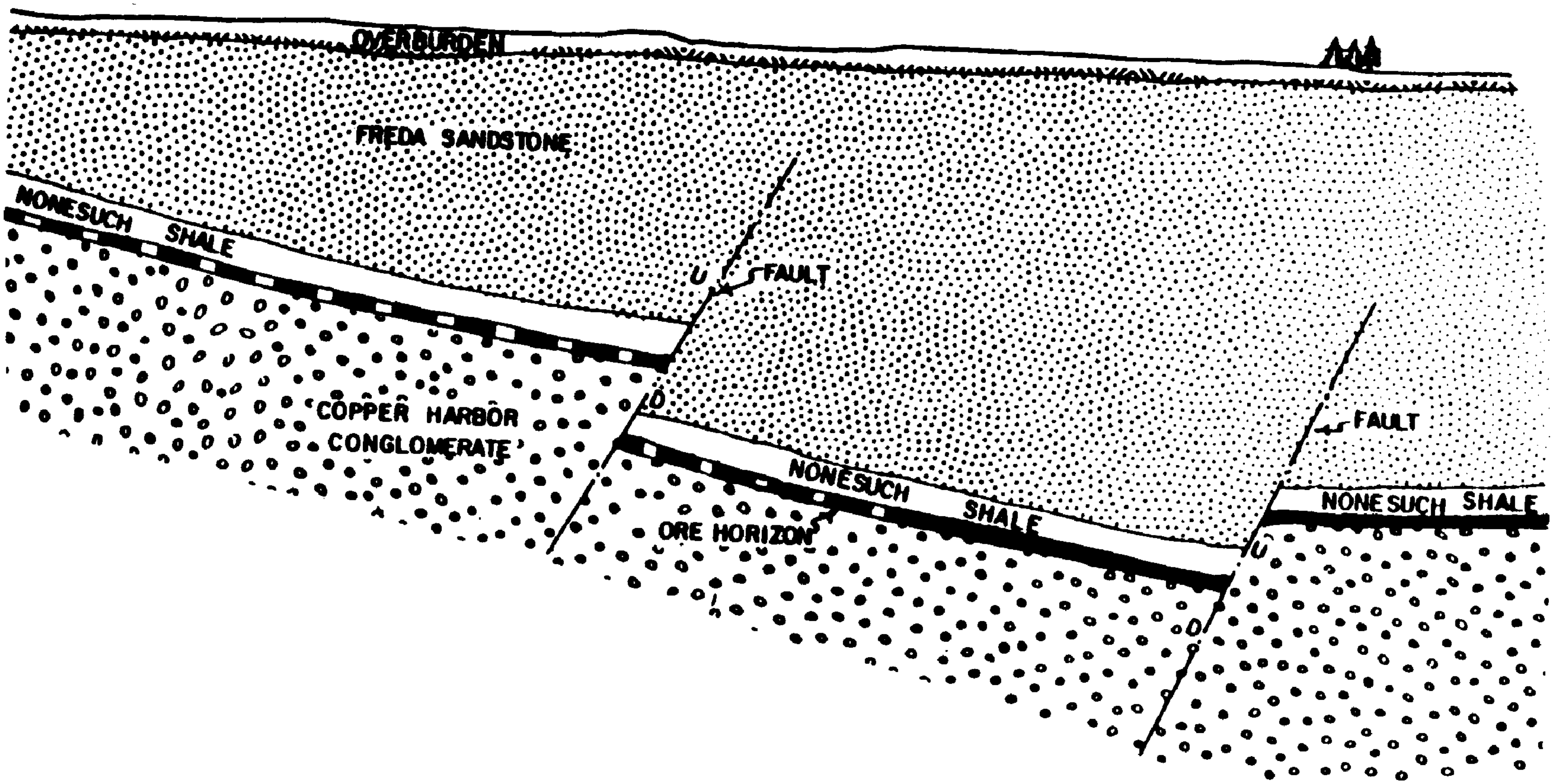
The White Pine area experiences a continental to semi-maritime climate, largely dominated by the passage of weather systems from west to east and the modifying influence of Lake Superior. The stabilizing effects of the Lake on air temperatures and prevailing winds provide for cold winters, cool summers, and consistently high humidity throughout the year. The mean annual temperature for White Pine, Michigan, is approximately 40
low 80's in the summer. Precipitation is well distributed throughout the year with the non-snow season (May-October) receiving an average of 18.43 inches (58 percent of the annual total rainfall of 32 inches). The annual snowfall at White Pine is 177 inches. The White Pine area averages 142 days with one inch or more of snow on the ground. Peak snow depth occurs in February and March (Copper Range Co. 1978).

4.1.3.1 Geology

Except as otherwise noted, the following geological information is excerpted from an *Environmental Assessment of the Operations of the White Pine Copper Division, Copper Range Company* (Copper Range Co. 1978). The copper deposit at the White Pine mine in Ontonagon County, Michigan, is the largest known copper-shale deposit in the United States. It lies within a belt of copper mineralization extending from Mellen, Wisconsin, to the eastern-most edge of the Keweenaw Peninsula, a distance of more than 100 miles. Ontonagon County is underlain by the Middle and Upper Keweenawan rock sequences. All the deposits in the Keweenaw sequences are of Late Precambrian time.

The Middle Keweenawan sequence is comprised of three parts: 1) the lower part, a thin series of interbedded sandstones, shales, and marls; 2) the middle part, composed of volcanic rocks with thin interbedded clastic sediments having a total thickness greater than 30,000 feet; and 3) the upper part (known as the Portage Lake Lava Series), composed of permeable lava flows and conglomerates with a thickness of approximately 20,000 feet.

The Upper Keweenawan sequence overlies the Portage Lake Lava Series. The Upper Keweenawan is divided into three major formations, the Freda Sandstone Formation, the Nonesuch Shale Formation, and the Copper Harbor Conglomerate (See Figure 4-2).


OVERBURDEN
FREDA SANDSTONE
NONESUCH SHALE
U
FAULT
D
COPPER HARBOR CONGLOMERATE
NONESUCH SHALE
ORE HORIZON
FAULT
U
D
NONESUCH SHALE

The Copper Harbor Conglomerate Formation overlies the Portage Lake Lava Series and ranges in thickness from approximately 350 feet in areas south of the White Pine mine to 6,000 feet near the shore of Lake Superior. It contains primarily reddish-brown to grayish-brown, fine- to coarse-grained conglomerate and reddish-brown siltstone, commonly cross-bedded with sandstone. Locally, flows of mafic lava up to several hundred feet thick are interbedded with the clastics.

The Nonesuch Shale and Freda Sandstone Formations overlie the Copper Harbor Conglomerate. In the vicinity of the White Pine mine, the Nonesuch Shale Formation is approximately 600 feet thick and composed of laminated gray to brownish-gray siltstone with minor shale and sandstone and reddish-brown partings. Gray to dark gray siltstone dominates the lower 100 feet, while massive light-gray siltstone to very fine-grained sandstone becomes abundant in the lower-middle and uppermost sections of the formation.

The Freda Sandstone Formation is the upper-most unit of the Keweenawan sequence. The formation is composed of fine arkosic sandstones alternating with red micaceous silty shale. Shale pebble conglomerate horizons occur frequently in the Freda. A conglomerate composed of Huronian iron formation, basalt, and quartzite pebbles, varying from 15 to 150 feet in thickness, occurs in many places as a distinctive horizon several hundred feet up from the base of the formation. Micro-cross-laminations of the fine-grained parts of the formation are a distinctive feature of the Freda. Ripple marks, mud cracks, rain imprints, rill marks, micro-cross-bedding, and angular shale pebble conglomerates attest to shallow water deposition and repeated interruption of sedimentation and exposure to subaerial erosion (Hamblin 1961).

Copper deposits mined at Copper Range Company's White Pine mine are located in the lowermost 18 to 22 feet of the Nonesuch Shale Formation and the uppermost 10 feet of the Copper Harbor Conglomerate Formation. These copper deposits have been locally subdivided into four sections: the Lower Sandstone, the Parting Shale, the Upper Sandstone, and the Upper Shale.

The primary minerals within the White Pine ore body are native copper (Cu), native silver (Ag), and chalcocite (Cu_2S), with minor amounts of bornite (Cu_5FeS_4), chalcopyrite ($CuFeS_2$), and pyrite (FeS_2) also occurring. Native Copper (Cu) is locally abundant primarily in the Lower Sandstone and Parting Shale units and constitutes between 7 and 9 percent of the copper processed at White Pine. It is generally associated with chalcocite as individual grains, but does occur separately in some laminae. It may be as thin as 0.2 millimeter (mm) in sheets 2 to 3 feet long. Native silver (Ag) occurs sporadically and is often associated with native copper. It is either in the form of individual grains, which may be associated with chalcocite only, or in the form of rims on sheets or grains of native copper. Other sulfide minerals including covellite (CuS), greenockite (CdS), sphalerite (ZnS), and galena (PbS) have also been identified at White Pine. A host of other trace elements are also present in the ore body.

Approximately 80 percent of the copper processed at White Pine is chalcocite. Chalcocite is often concentrated in dark-gray laminate shales, occurring most often as finely disseminated grains averaging 2 to 20 microns in diameter in siltstone and 1 mm in diameter shale in sandstone. It also occurs infrequently as 2 and 5 mm oblate ellipsoids in these areas.

4.1.3.2 Hydrology

The following discussion of hydrology in the area of Copper Range Company's White Pine mine is divided into separate Surface Water and Ground Water Sections.

Surface Water

Surface waterbodies associated with Copper Range Company's White Pine mine include Perch Creek, Mineral River, and Lake Superior. Perch Creek receives discharged effluent from the White Pine tailings impoundment system at the facility's NPDES outfall (see Figure 4-3)

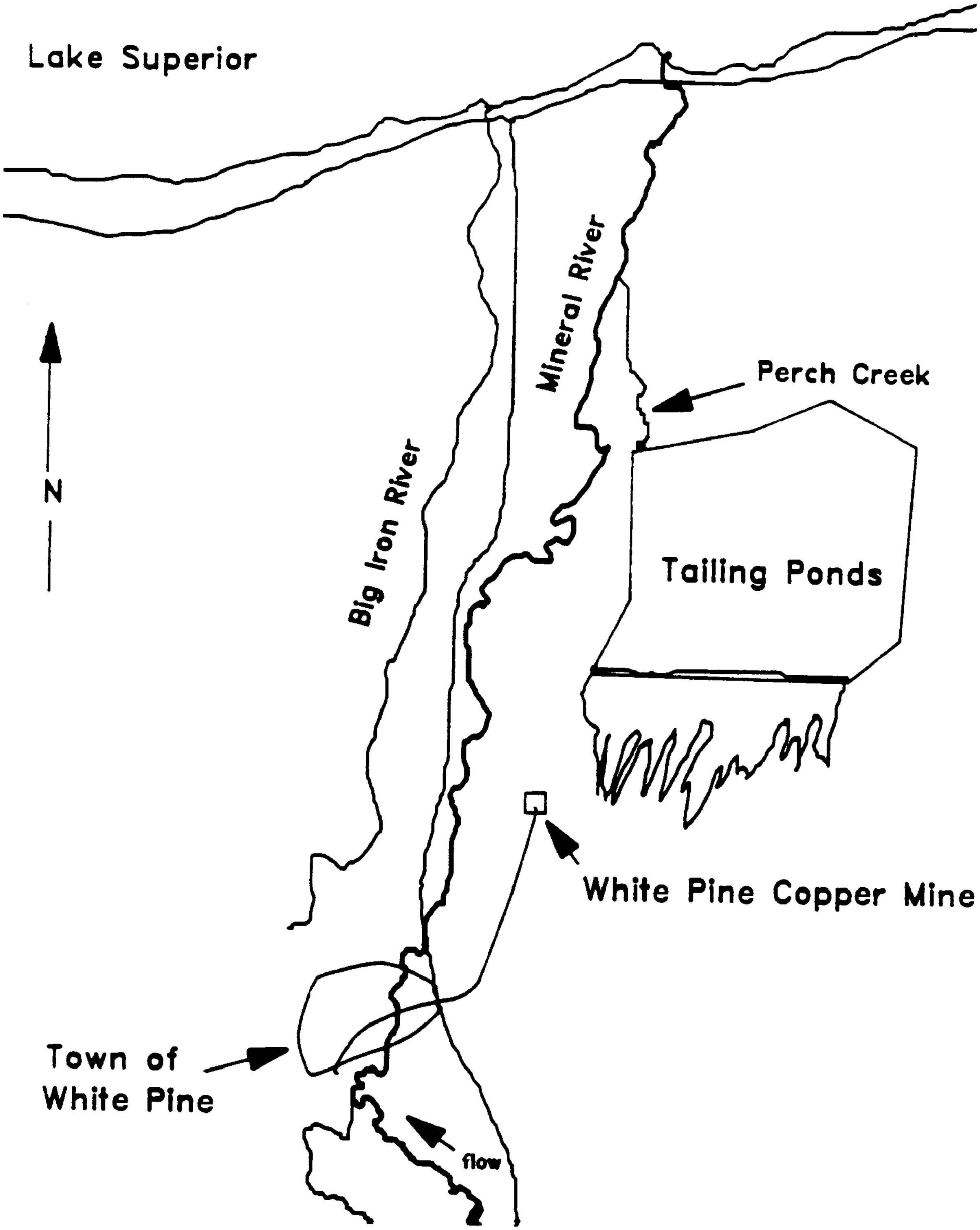


Figure 4-3. Surface Water in the White Pine Area

(Source: MDNR 1989)

. According to Copper Range Company personnel, Perch Creek flow is made up primarily of tailings impoundment effluent. The average flow in Perch Creek is approximately 25 million gallons per day (MGD).

Perch Creek flows north towards Lake Superior and discharges into the Mineral River at a point referred to as the Perch Creek confluence (see Figure 4-3). The Mineral River flows on bedrock and unconsolidated sediment in the vicinity of the White Pine mine, comprised mainly of shale slabs, boulder, and rubble, covered with red clay. The Mineral River flows north and discharges into Lake Superior (MDNR 1989).

The Mineral River experiences low-flow conditions in summer and winter upstream of the Perch Creek confluence. During these times, flow in the Mineral River downstream of the Perch Creek confluence is primarily tailings impoundment effluent from the White Pine mine. Elevated total dissolved solids (TDS) and chlorides are found in the Mineral River below the Perch Creek confluence due to calcium, magnesium, and sodium imparted by Perch Creek into the River (MDNR 1989).

Ground Water

According to the Michigan Department of Natural Resources (MDNR), ground water supplies are not abundant on the Upper Michigan Peninsula. The quantity and quality of water vary from one area to another. Wells yielding large supplies (several hundred gallons per minute or more) are rarely found on the peninsula. In some areas of the peninsula, it is almost impossible to obtain even the small supply needed for domestic use (MDNR 1969).

Glacial material is not considered a reliable source of water over much of Ontonagon County. Approximately 60 percent of the wells in the County are drilled into bedrock. Drilling into bedrock, however, does not ensure a satisfactory supply; according to MDNR, inadequate supplies or saline water are common problems. Ground water generally occurs in fractures with yield depending on the number and size of fractures penetrated. The first 50 feet of bedrock typically yield a moderate supply. At greater depths, open fractures are rare, and yields are smaller and have an increasing chance of contacting saline water (MDNR 1969).

Freda Sandstones and Nonesuch Shale underlie the glacial drift in most of the northern third of Ontonagon County. The sandstone beds yield fresh water to many shallow wells, but the deeper sandstone wells generally yield water too high in chlorides for domestic use. Near Lake Superior, most bedrock wells more than 75 feet deep yield saline water; farther south, most wells less than 150 feet deep yield fresh water. Many of the wells yield water containing iron. Most wells yield enough water for a domestic water supply, but in some instances drilling more than one well is necessary to obtain a satisfactory supply (MDNR 1969).

The Freda Sandstone formation is classified as a Class II aquifer by the U.S. EPA. This classification designates the aquifer as a current and potential source of drinking water and as having other beneficial uses. According to Copper Range Company personnel, the depth from the surface to the Freda Sandstone formation aquifer at the White Pine mine site ranges from less than 20 feet to 75 feet. The thickness of the aquifer ranges between 6 and 25 feet. The principle use of the aquifer is as a source of private drinking water. Although there are no drinking water wells on the Copper Range Company property, both drinking water and public water system wells are located within 1/4-mile of the site.

4.2 FACILITY OPERATIONS

4.2.1 Mining Operations

Mining at the Copper Range Company White Pine facility is conducted underground using standard room-and-pillar extraction techniques. The underground mine currently covers 25 square miles (5 miles by 5 miles). The mine extends down from the portal at the plant site to active mining at depths ranging from 1,500 - 2,700 feet. At the time of the site visit, mining occurred in two general areas of the mine (the Northeast and Southwest Areas). Within these areas, mining is performed in discrete sections; active sections in the Northeast Area include sections 21, 22, 23, 24, 25, 31, and 32, while active sections in the Southwest Area include 91, 93, 96, 97, and 99 (see Figure 4-4)

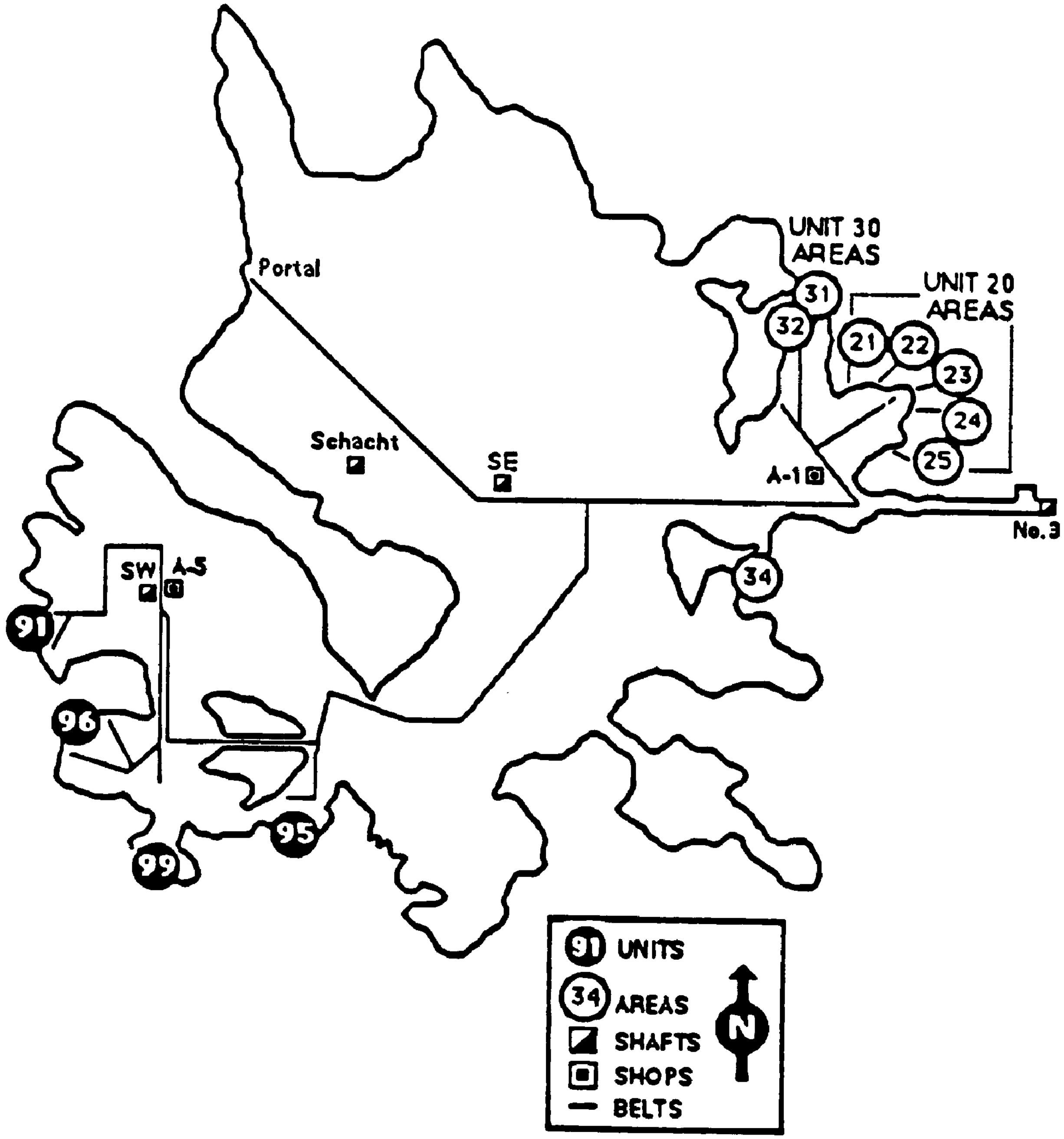


Figure 4-4. Map of White Pine Mine

(Source: Copper Range Co. Undated)

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The Southwest mine area has a 2,200-foot deep shaft used for personnel and materials transport, as well as ventilation. The Northeast mine area has a 440-foot deep shaft, located approximately 15,000 feet from active mining sections (Skillings' Mining Review 1992).

Mine ore is extracted from rooms about 20 to 28 feet wide and 8 to 17 feet high. In room-and-pillar mining, the rooms are mined on a regular pattern, separated from each other by pillars of un-mined ore (the width of pillars at the White Pine mine was not determined). All mining in a section of the White Pine mine is conducted on one level. Connecting roads between rooms and sections of the mine are generally 25 feet wide and 11 feet high.

The Copper Range Company mining cycle includes: bolting, drilling, powdering, blasting, loading, hauling, crushing, and conveying (Copper Range Co. Undated). These activities are discussed in greater detail below.

Bolts are installed to stabilize walls and ceilings in a technique known as roof bolting. In roof bolting, holes are drilled into the roof at regular intervals and steel bolts with resin capsules are inserted into the holes. The resin then solidifies to form a bond between the bolt and rock to secure the roof (Copper Range Co. Undated).

Holes measuring 1 3/4 feet in diameter are drilled 12 to 15 feet into the ore column "face" with a drilling machine. The number of holes depends on the size of the room, and blast patterns vary, depending on location. The drilled holes are then loaded with explosives, and are blasted at the end of each shift (Copper Range Co. Undated). Copper Range Company blasts exclusively with ammonium nitrate fuel oil (ANFO). Each blast breaks about 350 tons of ore, with a total of 17,000 tons of ore removed from the mine daily.

Broken ore is scooped or mucked into trucks or carried directly by trams to portable primary (Stamler) crushers, which reduce the ore size to less than 12 inches (Copper Range Co. Undated). Crushed ore is then automatically deposited on conveyors for transport to the surface. As ore is conveyed to the surface, it is transferred (at Transfer Point #1) to one of two concrete coarse ore bins.

Overall, the mine workings at the White Pine mine were generally dry at the time of the EPA site visit. When blasting occurs, mine water flows are released and accumulate in the blast area. Portable suction pumps are used to collect this mine water and transport it to local underground sumps. Water used for dust control during blasting is also pumped to these sumps. From the local sumps, water flows into one of three central underground mine water sumps. From these central sumps, mine water is pumped to the surface and into the tailings management system.

Backfilling is not practiced at White Pine and there is no waste rock. All mined material is considered to be ore and is crushed and beneficiated. White Pine ore grade averages between 1 and 1.1 percent copper. According to Copper Range Company personnel, at the time of the site visit the facility was mining lower than average grade ore because it is easier to access.

4.2.2 Beneficiation Operations

The objective of beneficiation operations is to produce a high grade copper concentrate that can be smelted. Mined ore is crushed, ground in rod and ball mills, and floated to recover the copper minerals contained in the ore (Copper Range Co. Undated).

From 17,000 tons per day of ore extracted from the mine, the mill produces approximately 500 tons of concentrate per day, which assays at 30 percent copper. About 87 to 89 percent of the copper in the ore is recovered in the mill. The mill product is then filtered and dried prior to smelting (Copper Range Co. Undated).

4.2.2.1 Crushing and Grinding

At the surface, ore is received into two-1,500 live ton capacity coarse ore storage bins and withdrawn over double deck scalping screens. Greater than 3 inch size rock is crushed by two-7 foot Symons standard cone crushers. Scalping screen undersize is sent directly to fine ore storage, while the intermediate product is conveyed to four-1,000 live ton Shorthead crusher feed bins. Shorthead feed is processed through four parallel screening and crushing lines to make a final nominal crusher product size of 5/8 inch. Each bin has a capacity of 1,500 tons of ore. The crushing plant is equipped with one wet and one "Rotoclone N" dust collector.

Crusher product is conveyed and distributed to one of seven fine ore bins which feed the grinding circuit. There are three grinding sections consisting of two parallel lines per section and a fourth section with a single larger capacity set of mills. Rock is withdrawn from beneath the fine ore storage bins, conveyed to the rod mill feed chute and mixed with fresh water and a portion of the required flotation reagent. Each grinding line consists of an open circuit rod mill in series with a ball mill in closed circuit with cyclone classifiers.

A portion of the xanthate collector, necessary for flotation, is added to the ore in the ball mills. The discharge from each ball mill is fed through a primary hydrocyclone, where the overflow (undersize) material (5 percent greater than 100 mesh) is sent to primary flotation, while the underflow (oversize) material returns to the ball mill for further grinding.

4.2.2.2 Flotation

Flotation at Copper Range Company is accomplished in four stages. (Specific information concerning flotation cell sizes and capacities was not collected during the site visit.) A detailed flow diagram of the White Pine flotation circuit is shown in Figure 4-5.

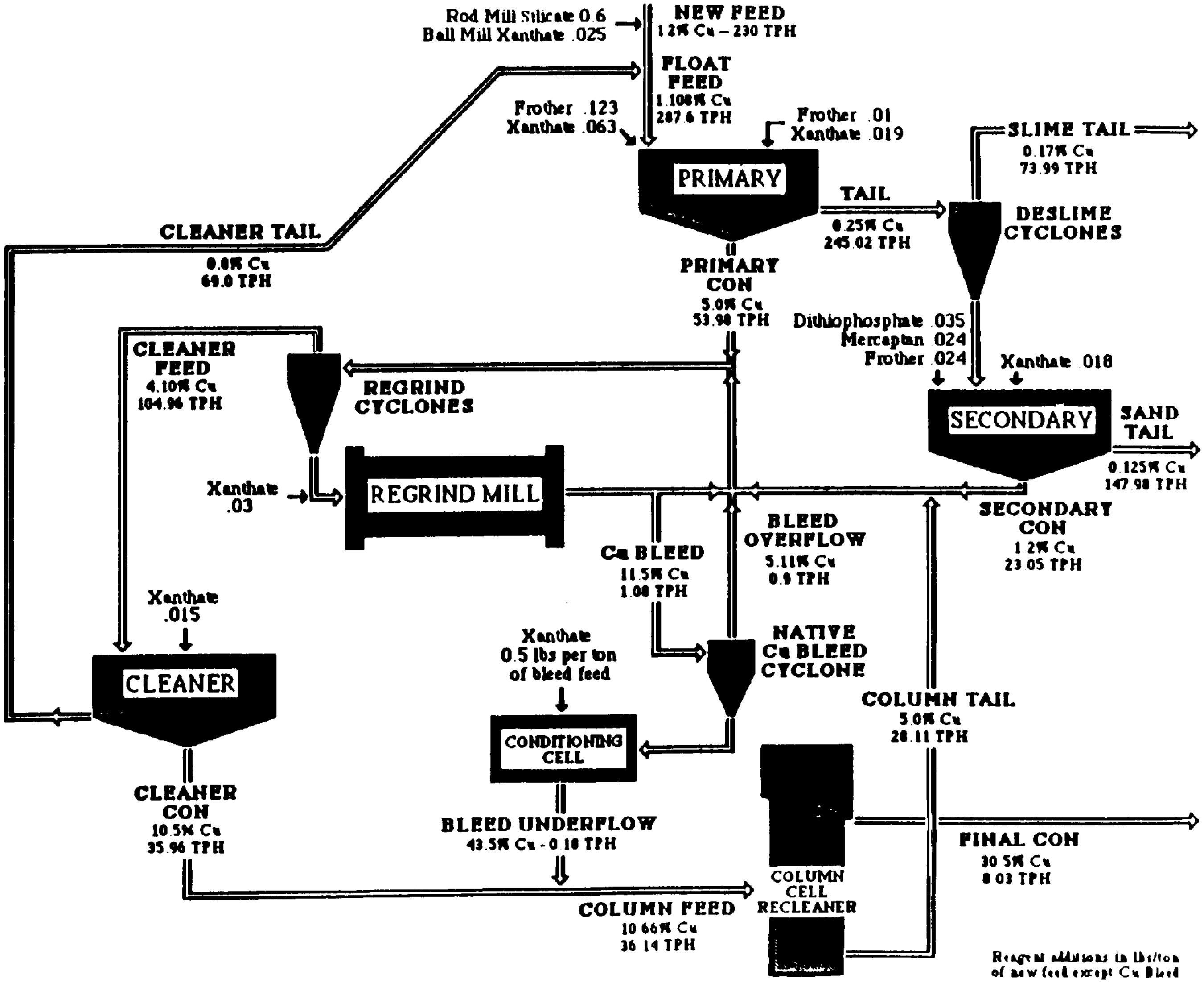
Rod Mill Silicate 0.6
Ball Mill Xanthate .025
NEW FEED
1.2% Cu – 230 TPH
FLOAT FEED
1.108% Cu
287.6 TPH
Frother .123
Xanthate .063
Frother .01
Xanthate .019
PRIMARY
SLIME TAIL
0.17% Cu
73.99 TPH
TAIL
0.25% Cu
245.02 TPH
DESLIME CYCLONES
CLEANER TAIL
0.8% Cu
69.0 TPH
PRIMARY CON
5.0% Cu
53.98 TPH
Dithiophosphate .035
Mercaptan .024
Frother .024
Xanthate .018
CLEANER FEED
4.10% Cu
104.96 TPH
REGRIND CYCLONES
SECONDARY
SAND TAIL
0.125% Cu
147.98 TPH
Xanthate .03
REGRIND MILL
SECONDARY CON
1.2% Cu
23.05 TPH
Cu BLEED
11.5% Cu
1.08 TPH
BLEED OVERFLOW
5.11% Cu
0.9 TPH
Xanthate .015
Xanthate 0.5 lbs per ton of bleed feed
NATIVE Cu BLEED CYCLONE
CLEANER
COLUMN TAIL
5.0% Cu
28.11 TPH
CONDITIONING CELL
CLEANER CON
10.5% Cu
35.96 TPH
BLEED UNDERFLOW
43.5% Cu - 0.18 TPH
FINAL CON
30.5% Cu
8.03 TPH
COLUMN CELL RECLEANER
COLUMN FEED
10.66% Cu
36.14 TPH
Reagent additions in lbs/ton of new feed except Cu Bleed

Table 4-1. Purpose and 1992 Application Rates of Flotation Reagent Classes

Reagent Class	Purpose	1992 Application Rate (pounds/ton)
Frothers	Promote the formation of froth having the desired characteristics (stiffness or fragility) and having little or no collecting property.	0.1085
Primary Collectors and Promoters	Selectively coat the particles to be floated with a water-repellant surface that will adhere to air bubbles.	0.1982
		0.0194

(Source: U.S. EPA Field Notes; Cummins and Given 1973)

Table 4-2. 1991 Annual Reagent Consumption and 1992 Application Rates at Copper Range Company's White Pine Mine

Reagent	1991 Annual Consumption (tons/year)	1992 Application Rate (lbs/ton)
Xanthate	987,865	0.1821
Test Collectors	87,812	0.0160
n-Dodecyl Mercaptan	158,592	0.0273
Flocculants	62,248	0.0061
Defoamers	7,614	0.0153

(Source: U.S. EPA Field Notes)

Reagent functions include frothers and collectors/promoters. The specific purpose of each class of reagent used at White Pine, as well as their collective 1992 application rates, are presented in Table 4-1. A complete list of all flotation reagents used at White Pine, along with their individual application rates, are listed in Table 4-2. Lime is also added to the ore at a rate of 0.0001 pounds per ton (25 tons per month) to buffer the rotary kiln dryer scrubber water, which is discharged to the concentrate thickeners (see below) and is characterized by low pH (the actual initial and final scrubber water pH were not determined).

Primary rougher flotation at Copper Range Company is a standard froth flotation system, using mechanical cells. Primary rougher flotation overflow is fed to the concentrate regrind mill, while the tails are sent to a desliming cyclone. The overflow from the desliming cyclone is discharged as slime tails, and the coarse sand underflow is sent to secondary scavenger flotation.

Secondary flotation at White Pine is also standard froth flotation. Secondary flotation cell overflow is fed to the regrind cyclone, while the underflow is discharged as sand tails.

Overflow from the regrind cyclone flows to the first stage of cleaner flotation. Underflow from the cyclone is fed to a regrind mill for additional size reduction. Reground ore is either discharged to a native copper bleed cyclone or to the secondary flotation circuit.

Copper Range Company operates four Deister column flotation cells (recleaners). Tails from column flotation are returned to the regrind system for additional size reduction. Final concentrate from recleaning flotation is sent, along with concentrate from the copper bleed flotation system, to thickeners. Tails from copper bleed flotation are fed to the column cells. Copper Range Company's final flotation concentrate is about 30 percent copper (Copper Range Co. Undated; MDNR 1992b). Final mill tailings are analyzed for copper content to assess mill performance and are discharged via a pumphouse to the tailings management system (see tailings discussion in Section 4.3.1.1 of this report). Copper Range Company's White Pine mine generates an average of 15,300 tons of tailings per day (approximately 5.6 million tons per year).

4.2.2.3 Concentrate Thickening

Copper Range Company operates two (one primary, one secondary) concrete concentrate settling ponds/thickeners in series, each measuring 100 feet in diameter (the depth and capacity of the thickeners were not determined). Overflow water from the thickeners is used as makeup water for the mill. Underflow from the thickeners is sent to the filter building.

4.2.2.4 Filtering and Drying

In the filter building, two rotary drum vacuum filters are used to produce a concentrate filter cake (at the time of the site visit, only one of the two filters was operating). The concentrate filter cake leaving the filter building has been dewatered to contain 20 - 22 percent moisture (MDNR 1992b).

From the filter building, the concentrate is placed on conveyors, fluxed with limestone and mixed with concentrates from other mines (application rates and total quantities used were not collected) and sent to a gas-fired rotary kiln dryer, equipped with a wet scrubber system. The scrubber water, characterized by low pH, is sent back to the concentrate thickeners. Fluxes used at the White Pine facility include limestone and precipitated dust from the facility's electrostatic precipitator (identified by Copper Range Company personnel as an unspecified recycled material). Limestone is stored in on-site piles under a shed roof. The kiln dryer reduces the moisture content of the concentrate to approximately 10 to 13 percent.

In the past, pyrite was added to high silicate White Pine ore to provide a source of sulfur (needed for smelting). Today, sufficient sulfur content in the smelter feed is obtained in ore concentrate received from other mines and mills (such as Phelps Dodge's Morenci operation), which is co-smelted with White Pine ore. The quantity of concentrate from other operations that is smelted by Copper Range Company varies on a daily basis.

4.2.3 Smelting and Refining

The effluent from the kiln dryer is fed into a reverbatory furnace, where it melts and separates into slag and matte. The liquid slag is skimmed off through tap holes while the matte, now 65 percent copper, is drawn off and charged into a converter furnace (Copper Range Co. Undated).

In the converter furnace, low pressure air is added to produce a 99 percent pure blister copper. A fire-refining furnace eliminates the remaining removable impurities, leaving 99.7 percent pure copper that is ready for casting. The copper is cast into anodes measuring 37 inches by 36 inches by 1.5 inches, and weighing over 600 pounds (Copper Range Co. Undated). Smelter stack emissions monitoring, including sulfur dioxide (SO_2) and arsenic, is discussed in Section 4.4 of this report.

Anodes go to an electrolytic refinery, where they are placed in cells containing electrolytic solution. Anodes are alternated in the cells with stainless steel blanks, which are the cathodes. When electrical current is sent through the system, copper is plated on the blanks, and the impurities sink to the bottoms of the cells. When the blanks are plated with copper on both sides, they are removed from the cells and stripped of copper. This final product, called cathode, weighs 90 pounds and measures 38 inches by 38 inches by three-sixteenth (3/16) inch. It is ready for shipment to market. Annual target cathode production output is approximately 120 million pounds (Copper Range Co. Undated).

Silver also occurs in the ore body as an important byproduct. It is collected in the slime at the bottom of the electrolytic cells and shipped for additional processing for silver recovery (Copper Range Co. Undated). The site team did not visit the smelting and refining portions of the site and information related to stack height, emission rates, and emission controls was not obtained.

4.2.4 Other Areas

4.2.4.1 Assay Laboratory

Copper Range Company operates a full, on-site wet laboratory located near the filter building. The laboratory analyzes samples from the mill, smelter, electrolytic refinery, and tailings.

4.2.4.2 Fuel Oil Storage Areas

Upon receipt at the facility, fuel oil and antifreeze (contained in 55-gallon drums) are stored in a central outdoor area. The drums are elevated above ground with no secondary containment. In facility shops and process areas, fuel oil, as well as lubricants and solvents, are stored (prior to use) without secondary containment. In some cases, drums of these materials are elevated above ground.

4.2.4.3 Power Plant

Copper Range Company has an on-site power plant, which is operated by 20 employees (including four wastewater treatment plant operators). The plant has three turbines with a nominal total capacity of 55 megawatts (MgW) (average peak load of 53 MgW). Two stoker-fired boilers feed two turbines (20 MgW capacity each). Waste heat recovered from the smelter feeds the third turbine (15 MgW capacity).

The boilers can be fired with coal, natural gas, or oil, depending on relative fuel prices. Natural gas has been used since February 1, 1992. When coal is used, low sulfur coal (less than 1.5 percent) is stockpiled in an outside storage area with very little surge capacity (most coal is delivered on a daily basis). Information was not obtained concerning power plant emissions or controls.

4.2.4.4 Shops

Maintenance and repair of surface department equipment is conducted in an on-site surface shop. Major repairs of mine equipment are also performed in surface shops. However, most mine equipment repairs are conducted in underground shops.

4.2.4.5 Underground Storage Tanks

At the time of the site visit, Copper Range Company had 15 State-permitted underground storage tanks at the White Pine site (several additional tanks had recently been removed). Only one of these tanks, a 20,000 gallon gasoline tank, was being used. This tank was inspected daily for leaks. An additional underground tank was currently empty, but can be used to store heating oil. The remaining 13 tanks, ranging in size up to 20,000 gallons, were empty and permanently inactive. All of the inactive tanks were previously used to store petroleum products.

4.2.4.6 Warehouse

Two on-site warehouses provide supplies to the mine and surface departments from a stock of 27,000 separate items. All chemicals, including flotation reagents, are delivered to a reagent storage area in the warehouse prior to being sent to specific areas for use.

4.2.4.7 Wastewater Treatment Systems

Copper Range Company operates two wastewater treatment plants. The main plant (located about 0.5 miles northwest of the smelter building) provides wastewater treatment for the town of White Pine, as well as most of the mine site. Treatment at this plant includes primary and secondary (i.e., biological) treatment. The main plant effluent is pumped to the tailings pumphouse and mixed with mill tailings. The lime in the tailings provides for additional phosphorous removal. Copper Range Company's smaller wastewater treatment plant (located at the southeast shaft) uses aeration treatment.

Effluent from both treatment plants is discharged to Copper Range Company's tailings management system via outfall 00A (see NPDES Permit discussion in Section 4.4.2.1 of this report). The average flow from these systems is approximately 0.004 MGD. Sewage sludges are sent to drying beds and are eventually used to promote revegetation of tailings impoundments areas.

4.2.4.8 Potable Water System

Copper Range Company pumps all of its fresh water (for potable and process uses) from Lake Superior. The water pump station is located at the Lake, 6 miles away from the mill site. The total fresh water volume pumped from the Lake to the facility averages approximately 20 MGD.

Of the 20 MGD input, an average of 0.3 MGD (1 MGD maximum) is sent to Copper Range Company's potable water plant. This plant supplies both the facility and the town of White Pine with potable water. Alum, soda ash, fluorine (in the form of fluosilicic acid), and chlorine are added to the pumped water, which is then filtered in three sand beds.

Complete volatile/semivolatile analyses are performed quarterly on treated potable water. According to Copper Range Company personnel, only chlorodibromomethane, chloroform, and dichlorobromomethane

Table 4-3. Results of Sampling for Constituents in Potable Water and Applicable MCLs

Compound	**7/24/91 ppm**	**12/12/91 ppm**	**3/23/92 ppm**
Chlorodibromomethane	0.0017	0.0005	0.0503
Chloroform	0.0237	0.0125	0.0177
Dichlorobromomethane	0.0068	0.0032	0.0023
Total Trichloromethanes*	0.322	0.0162	0.0703

* There are no MCLs for the individual compounds listed above. However, the MCL for total trihalomethanes is 0.1 ppm.

(Source: U.S. EPA Field Notes)

have been found in samples. These are thought to be byproducts of the chlorination and fluorination processes. Table 4-3 provides the results of the most recent analyses for these constituents.

Water pumped from Lake Superior to be used in facility processes is sent to a concrete-lined spray pond (3,800,000 gallon capacity) and combined with non-contact cooling water from the power plant. According to Copper Range Company personnel, no further treatment, with the exception of occasional chlorine addition, is employed prior to use in facility processes.

4.3 WASTE AND MATERIALS MANAGEMENT

A number of extraction and beneficiation wastes and materials are generated at the site and are managed in the facility's tailings impoundment system. Mineral processing wastes are also generated on-site. In addition, other wastes are generated on-site that are not uniquely related to mineral extraction, beneficiation, and processing. The hazardous portion of these wastes are typically shipped off-site for disposal. Nonhazardous wastes, such as tires and refuse, are managed both on- and off-site.

This section emphasizes management of extraction and beneficiation wastes and materials, and the units in which they are managed, as well as areas where processing wastes and materials are commingled with those from extraction and beneficiation. Although processing is generally beyond the scope of this report, limited information on these wastes and materials will be discussed below in order to characterize the material balance throughout the facility.

4.3.1 Extraction and Beneficiation Wastes and Materials

4.3.1.1 Tailings

Tailings samples are collected and analyzed daily in the on-site assay laboratory. These samples are primarily analyzed for copper and silver to assess mill performance. The results of analyses conducted immediately prior to the EPA site visit are summarized in Table 4-4 below.

Table 4-4. Results of Tailings Analyses (May 3 and 4, 1992)

Date	Percent Copper	Troy Oz. Silver/ Short Ton	Date	Percent Copper	Troy Oz. Silver/ Short Ton
May 3, 1992	0.166	0.031	May 4, 1992	0.147	0.027
	0.153			0.154	
	0.159			0.164	

(Source: U.S. EPA Field Notes)

Other tailings components may include sandstone, shale, trace metals and trace flotation reagents. Copper Range Company monitors the final effluent discharge from the tailings impoundment treatment system (outfall 001) and intermediate internal outfalls (00A and 00B) as required by their NPDES Permit.

In 1991, Copper Range Company's White Pine facility generated 5,583,483 tons of tailings in 1991. Tailings from the mill are discharged to a tailings pumphouse via two separate (sand and slime) launders. Lime additions in the mill increase precipitation and settling of solids in the tailings impoundment treatment system. Approximately one-tenth of a pound of lime per ton of tailings was added in 1991, resulting in a total addition of 604,000 pounds during the year.

At the pumphouse, sand and slime tailings flow to separate sump areas. The discharges from these areas normally are combined and pumped to the impoundment system via one of two steel pipelines. A launder and damper gate system within the pumphouse allows for separation of the sand tailings. They are then hydrocycloned in the sand plant portion of the pumphouse and pumped to North 2 impoundment via pipeline for use in construction of the core of the dam.

As shown in Figure 4-6

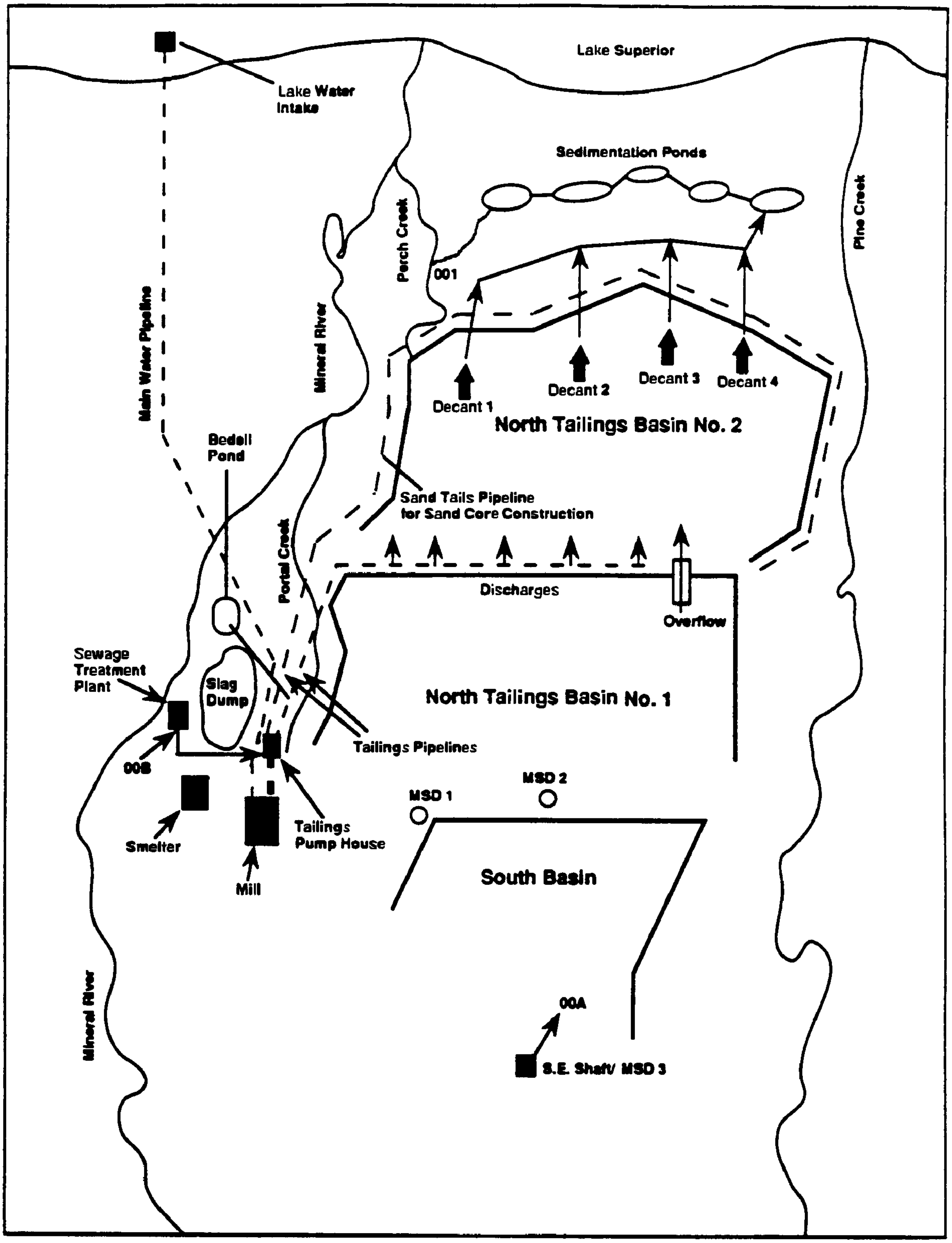

Figure 4-6. Map of Tailings System

(Source: U.S. EPA Field Notes)

, three tailings impoundments are located at Copper Range Company's White Pine mine: North 1 (1,850 acres), North 2 (2,450 acres), and the South Basin (1,300 acres). (Details about the height, capacities, and current tailings storage volumes of the facility's tailings impoundments were not determined.) The North 2 tailings dams (upstream and downstream) are constructed of clay with a central core of hydrocycloned sand tailings from the mill. The North 1 and South Basin dams, which were built by the facility for future tailings disposal, are constructed entirely of clay.

Tailings have been pumped to North 2 since October 1971. At the time of the site visit, it was estimated that North 2 had capacity for two more years of operation until the planned construction of an additional lift to the berm (see below). Tailings from the pumphouse flow to North 2 in steel pipes with no secondary containment. From the pipeline, tailings are discharged to the impoundment through one of five spigots (see Figure 4-6). The spigots are evenly spaced along the upstream dam of the impoundment. At the time of the EPA site visit, only the third spigot (central) was active. Copper Range Company also plans to utilize the first spigot (closest to plant) for tailings discharge when mill production increases. In addition to tailings, water collected in plant drains (including water from the mechanical shops, paint shops, and "spartan" steam cleaning) and sanitary wastewater from both facility water treatment plants are discharged to North 2. At the time of the EPA site visit, the surface of the impoundment was completely covered with water. The site visit team also noted the presence of standing trees in the North 2 impoundment.

Four decant towers, identified in Figure 4-6, are located in North 2 (evenly spaced near the downstream dam). These towers are used to decant clarified effluent from North 2 through a spillway and into the first of a series of five clay-lined basins (or polishing ponds) (see Figure 4-6). Water residence time in North 2 is 117 days. These ponds are among a number of old borrow pits that were originally used to obtain clay during initial construction of the impoundment dams. These ponds are not dredged by the facility.

Construction of the North 2 dam includes a french drain system to remove precipitation that enters the central sand core of the impoundment. These drains, along with runoff from the outside slopes of

North 2, discharge into a series of clay-lined ditches (spillways), which completely surround North 2 (except for the barrier between North 1 and North 2). The site visit team observed discharges from virtually all of the drainage pipes, which are spaced at intervals of 125 feet along the ditches. The drainage ditches flow into the polishing pond system.

No tailings water is reclaimed at White Pine. Rather, water is channeled through the polishing (sedimentation) ponds to provide final settling prior to discharge. No chemicals are added to the water in the ponds. The discharge from the last polishing pond passes over a concrete weir and rip-rap into Perch Creek (designated as NPDES outfall 001). Perch Creek flows into the Mineral River, which flows into Lake Superior. In the past, polymers were added to the decant pipe discharge to the spillway to promote settling and precipitation in the polishing pond system. However, this practice has been discontinued.

The average effluent discharge from outfall 001 during 1991 was 24 MGD. Freezing does not occur at outfall 001 due to the constant flow from the tailings system. The effluent discharge from outfall 001 accounts for essentially all of the flow in Perch Creek. Monitoring requirements and data for outfall 001 are described in Section 4.4.2 of this report.

At the time of the EPA site visit, Copper Range Company was experiencing a minor leakage of non-clarified tailings into the spillway system as a result of a malfunction in the bladder system of one of the decant towers. This malfunction caused concern that an increase in solids might clog the tailings decant system. Copper Range Company closed off the affected area of the spillway and has repaired the decant tower. According to Copper Range Company personnel, no change in the composition of outfall 001 effluent was experienced.

Prior to October 1971, North 1 was used for tailings disposal. The North 1 impoundment currently receives water directly from Mine Sump Nos. 1 and 2, indirectly from Sump No. 3 after prior settling in the South Basin, and direct precipitation. In the northeast corner of the unit, North 1 overflows to North 2. Water residence time in North 1 is 17 days. The EPA site visit team observed that North 1 was approximately one-third full (centered around the overflow area).

The North 1 impoundment dam is constructed entirely of clay. Therefore, according to Copper Range Company personnel, no seepage enters the confining berms. Sheet storm water runoff from the outer slopes of the west berm enters Portal Creek located immediately west of the unit. Sheet storm water runoff from the east berm enters Pine Creek located immediately east of the impoundment.

The South Basin currently receives mine water from Mine Sump No. 3, as well as direct precipitation. Water flows by gravity through the South Basin and then via pipe to North 1. Copper Range Company does not control runoff from the outside slopes of the South Basin. The EPA site visit team did not visit the South Basin.

According to Copper Range Company personnel, no reclamation is required for the tailings impoundments at White Pine. However, Copper Range Company has voluntarily seeded the outside slopes of the tailings impoundment(s) as well as the borrow pit areas, both for the control of erosion from natural elements and to provide cover and food for wildlife species which populate or migrate to the area. Substantial vegetation growth was noted by the EPA site visit team. Parts of the outside slopes of the North 2 impoundment have also been reseeded and lower areas of the slopes appear to be well vegetated.

Copper Range Company applied to the Michigan Department of Natural Resources (MDNR) for permission to add an additional lift to the North 2 dam under the State's Dam Construction Act. Specifically, the application calls for the addition of 200,000 cubic yards of clay to raise the dam three feet in height. The approval process involves only dam safety considerations. Copper Range Company also recently applied to MDNR to undertake additional construction work on the facility's South Basin in anticipation of future use.

Copper Range Company is not required to monitor ground water at the site.

4.3.1.2 Mine Water

As shown in Figure 4-6, Mine Sump Discharge (MSD) Nos. 1 and 2 discharge to the North 1 tailings basin (and subsequently North 2). MSD No. 3 discharges to the South Basin and thence by gravity flow, via pipeline, to North 1. The volumes of mine water generated were not determined. Analysis of mine water discharged to the impoundment system is performed on samples taken at the three sumps in the mine and/or at the MSD discharge pipes within the impoundment. According to Copper Range Company personnel, elevated total dissolved solids (TDS) levels detected in outfall 001 are caused by mine water since elevated TDS levels are characteristic of the connate water found in the Nonesuch formation, where mining is occurring.

4.3.2 Smelter Slag and Reverbatory Bricks

Although slag and bricks are generally beyond the scope of this report, they are briefly discussed here because runoff from piles of slag and brick storage areas is commingled with extraction and beneficiation wastes. Copper Range Company generated 207,843 tons of reverbatory furnace slag in 1991. Today, Copper Range Company operates a slag dump (constructed in 1953) located adjacent (just to the north) of the smelter. Historically, during the 1970's, the majority of the dump slag was crushed and underwent beneficiation (separation of copper-bearing slag from the waste slag) via a heavy media flotation process plant located within the slag dump. The residual from the plant (tailings) was graded and stockpiled at the north end of the dump. Periodically—during low mine production periods—copper-bearing slag is now processed in the mill to recover the copper. Piles of refractory bricks, from normal repair activity, are presently stored on the southern portion of the slag dump. These bricks are crushed at the site and recycled via the mill to recover the copper content. The piles of stored brick are covered with tarps. Scrap tires tied to lanyards are utilized to secure the tarps. Runoff from the slag dump flows to North 1 via the Bedell Pond.

4.3.3 Site Runoff

In general, all site runoff is collected and ultimately sent to the tailings pumphouse (with the exception of sheet storm water runoff from the outside slopes of the North 1 impoundment along Portal Creek to the west and Pine Creek to the east and the South Basin, as discussed above). Site runoff can be stored temporarily in a 50 million gallon reservoir (referred to as the Bedell Pond) prior to discharge to the tailings impoundment (see Figure 4-6).

4.3.4 Other Wastes and Materials

4.3.4.1 Laboratory

Analyses at the on-site lab generate a number of materials. A summary of them and their management is presented below:

- Electrolyte - recycled to the plant.
- Liquid samples (including tailings) - placed in barrels and disposed of off-site as hazardous waste by an outside firm. In 1991, 78 containers containing 9,864 kg of laboratory wastes were generated.
- Ceramics/crucibles - placed in barrels and disposed of off-site by an outside firm (no quantities available).

4.3.4.2 Sanitary Wastewater

Effluent from the sanitary wastewater treatment system located at the S.E. shaft dry is discharged through NPDES internal outfall 00A to the tailings management system (typical flows range from 0.003 to 0.005 MGD). Sewage treatment plant effluent from the main plant (NPDES internal outfall 00B), which also services the community of White Pine, is pumped to the tailings pumphouse and thence to the tailings management system.

4.3.4.3 Power Plant Wastes

Runoff from the coal storage pile, along with all plant runoff, is managed through the tailings impoundment system. Ash from the plant is sluiced with water from the spray pond and discharged to the tailings system. Approximately 216 gallons per day of ash sluice water (containing 30 percent solids) is generated at White Pine. The plant's boilers are equipped with wet scrubbers for dust control. Collected scrubber waste is combined with dust collected from the smelter flue and sent to the rotary kiln dryer (along with the fluxed feed from the filter plant).

4.3.4.4 Waste Oil

A 10,000-gallon aboveground tank is used to store vehicle and hydraulic waste oil. The tank is surrounded by a clay containment "moat." When the tank is filled, waste oil is manifested and shipped off-site by Oil Services, Inc., (Eveleth, Minnesota) for recycling and reuse in asphalt plants. In 1991, there were three pick-ups, totaling approximately 98,210 lbs (13,095 gallons) of used oil.

4.3.4.5 Refuse

Nonhazardous solid waste generated at Copper Range Company is classified in one of two categories: garbage (3,016 cubic yards generated in 1991) or inerts (3,741 cubic yards generated in 1991). Garbage includes lunchroom scraps, paper, cardboard, metal containers, glass, rags, and other rubbish. Inert materials include broken concrete, bricks, masonry, rocks, and uncontaminated soils. Both inerts and garbage are sent to a State Type II private landfill, identified as the K&W landfill, in Ontonagon, Michigan. It is not known why the facility manages these wastes separately prior to disposal.

Garbage is accumulated in designated areas and stored for pickup in containers by an outside firm. Inerts are temporarily managed in a large area adjacent to the facility road from the mill to the tailings impoundments. Throughout this storage area, the EPA site visit team noted stained soils and the presence of empty drums. In addition, explosives and caps are stored in tractor trailers in the inert storage area.

Scrap steel, wood pallets and other wooden items, and empty 55-gallon drums (which did not contain hazardous materials) are disposed of in an area identified by Copper Range Company as the "boneyard." These items are accumulated in the boneyard for future reuse, recycle, resale, or other management. According to Copper Range Company personnel, empty drums are managed according to the empty container requirements specified in 40 CFR Part 261.7.

4.3.4.6 Spent Solvents

In 1991, 172 containers containing 9,864 kilograms of spent solvent were disposed of off-site as hazardous wastes. Copper Range Company holds an EPA Hazardous Waste Generator permit (Permit No. MID086176658). A non-chlorinated solvent, "140 Stoddard," is the most commonly used solvent at Copper Range Company. Copper Range Company personnel indicated that an additional 10 containers (1,562 kg) of hazardous petroleum naphtha were also disposed of off-site. All shipments of hazardous waste are manifested. The EPA site visit team noted the presence of "Safety Kleen" solvent collection/storage containers throughout shop areas at the facility.

4.3.4.7 Batteries

Spent batteries are temporarily stored in the boneyard prior to pick-up by the original vendor. No information was available concerning the quantity of spent batteries generated at White Pine or the management of battery acids.

4.3.4.8 Scrap Tires

Copper Range Company has been involved in the recapping of tires since the early 1950's. Copper Range Company attempts to reuse tire casings to the greatest extent possible and has modified a number of vehicles to accept heavier ply tires to maximize recapping potential. To date, there are tires in use at White Pine that have been recapped over 20 times. On average, 1600x25 size tires are recapped five times and 1800x25 size tires are recapped eight times. On average, larger tires cannot be recapped as many times as smaller tires. Copper Range Company also works with rubber manufacturers on increasing recap life through the improvement of rubber compounds (Copper Range Co. 1992d).

Copper Range Company has also made improvements to extend the original tread life of its tires. Early tests have shown that tread life can be increased by 40 percent by installing cores in the tires of larger equipment. The use of cores also reduces waste, as only the damaged part must be removed, rather than an entire core or tire (Copper Range Co. 1992d).

Scrap tires are disposed of in a tire storage area permitted by the MDNR (it is not known if the area is lined). The tires managed in the area are equally proportioned between surface truck tires and mine equipment tires. The exact size and number of scrap tires in the tire storage area were not determined. The Copper Range Company is currently looking for a vendor to shred these scrap tires.

Recently, Copper Range Company began salvaging scrap tires from the pile. Eight tires have been removed from the scrap pile and sent for repairs. These tires are then run with tire cores in them. According to Copper Range Company, additional tires will be removed and reclaimed as time and resources allow (Copper Range Co. 1992d).

4.3.4.9 Asbestos

Copper Range Company personnel indicated that asbestos is found in buildings at the White Pine facility. An "Asbestos Team" (four employees) removes asbestos from locations at the site on an unspecified schedule. Asbestos materials are placed in bags, which are stored in a covered area adjacent to the hazardous waste storage area (no additional information was available concerning the hazardous waste storage area). When 100 bags have accumulated, the bags are shipped off-site to the permitted K&W landfill. The total quantity of asbestos material shipped off-site was not obtained.

4.3.4.10 Polychlorinated Biphenyls (PCBs)

Copper Range Company has eight electrical transformers containing PCBs located in the underground mine workings and a total of 49 PCB transformers located within the surface facilities. General PCB inspections are conducted on a quarterly basis. In addition, Copper Range Company conducted a PCB audit in the first quarter of 1990. This audit inspected all underground and surface operations. For units with the potential for PCB leaks, daily inspection and repair plans were developed. The EPA site visit team observed one power plant transformer containing PCBs. The transformer had secondary containment (i.e., it was surrounded by berms), and daily transformer inspection records were maintained.

4.4 REGULATORY REQUIREMENTS AND COMPLIANCE

A number of Michigan State agencies are responsible for regulating various aspects of Copper Range Company's White Pine operations. These agencies, the permits/authorizations issued to Copper Range Company, and the major permit requirements, are described below.

4.4.1 Mining Regulatory Requirements

The Michigan Mining Act does not regulate metals mining. Therefore, Copper Range Company is not required to develop or submit an operating/closure plan. There are no State requirements for tailings impoundment reclamation or bonding.

4.4.2 Surface Water Permits and Monitoring

4.4.2.1 NPDES Permit

NPDES permit No. MI0006114 was issued to Copper Range Company on January 1, 1991, by the Michigan Department of Natural Resources. The Copper Range Company NPDES permit addresses two discharges, outfalls 001 and 002. The limits for metals, total suspended solids (TSS), and pH are based on the effluent guidelines for the Ore Mining and Dressing Category (40 CFR Part 440). (It is unclear whether the permit limits are based on effluent guidelines for mine drainage and/or mill discharges.) For other parameters, limits are based on the State permit writer's best professional judgment. These limits are shown in Table 4-5.

As part of its 1990 NPDES permit renewal process, Copper Range Company was required to conduct biomonitoring (quarterly tests on two species using effluent from outfall 001), a Lake Superior mixing zone study, and as part of a Lake-wide initiative, examine new approaches to reducing their TDS/chlorides loadings to the Lake. These reports were submitted to the State in April 1992 and are discussed in greater detail in Sections 4.4.2.2 through 4.4.2.4 of this report (MDNR 1990).

Table 4-5. Monitoring Data for Outfall 001 (January - March 1992)

Parameter		Permit Limits	January 1992	February 1992	March 1992
Flow (MGD)	Monthly Avg	(report)	23.4	18.9	28
	Daily Max	51.1	27.3	21.3	30.6
pH (s.u.)	Range	6.5-9.0	7.4-7.6	7.4-7.5	7.1-7.6
TDS (mg/l)	Monthly Avg	(report)	1,900	1,650	2,000
	Daily Max	(report)	2,000	1,800	2,000
TSS (mg/l)	Monthly Avg	20	14	7	6
	Daily Max	30	18	10	12
Cadmium (µg/l)	Monthly Avg	50	0.6	0.6	0.7
	Daily Max	100	0.7	0.7	0.8
Copper (µg/l)	Monthly Avg	150	10	ND	25
	Daily Max	300	10	ND	40
Lead (µg/l)	Monthly Avg	300	ND	2	ND
	Daily Max	600	ND	3	ND
Zinc (µg/l)	Monthly Avg	500	5	ND	5
	Daily Max	1000	10	ND	10
Mercury (µg/l)	Monthly Avg	(report)	ND	ND	ND
	Daily Max	(report)	ND	ND	ND
Chloride (mg/l)	Monthly Avg	(report)	1,140	916	1,035
	Daily Max	(report)	1,215	917	1,114

ND = Not Detected

(Source: Copper Range Co. 1992e)

Outfall 001 is the discharge of effluent from the final clay polishing pond in the tailings water system into Perch Creek. The facility's permit requires monitoring for pH, chloride, metals, hardness, flow, TDS, and TSS. Sampling is conducted weekly for flow and pH and two times per month for all other parameters, with the results of monitoring submitted to the State on a monthly basis. Facility personnel indicated that a trace metals study was also conducted in the early 1980's (no additional details about the study were available).

Table 4-5 summarizes monitoring conducted by Copper Range Company at outfall 001 during January through March 1992.

Outfall 002 represents the discharge from the 3-Shaft sewage treatment plant. While Copper Range Company retains the permit for this outfall, the system is currently inactive, and no discharge actually occurs. Discharge monitoring reports for outfall 002 for the period January through March 1992 indicated no discharge during the period.

The NPDES permit also requires monitoring at two internal outfalls, designated as outfalls 00B (sewage treatment effluent from the main sewage treatment plant to the tailings pumphouse) and 00A (from Southeast Shaft sewage treatment plan to the South Basin).

Wastewater discharged from these outfalls eventually flows into the facility's tailings impoundment system. Specifically, outfall 00A is the discharge from the main wastewater treatment plant located north of the mill (includes sanitary wastewater from the town of White Pine) and outfall 00B is the discharge from Mine Water Sump No. 3 to the South Basin. Monitoring data for outfalls 00A and 00B for the period of January through March 1992 are summarized in Tables 4-6 and 4-7.

Table 4-6. Monitoring Data for Outfall 00A (January - March 1992)

Parameter		Permit Limit	January 1992	February 1992	March 1992
Flow (MGD)	Monthly Avg	(report)	0.004	0.004	0.003
	Daily Max	0.03	0.005	0.004	0.004
TSS (mg/l)	Monthly Avg	(report)	41	57	36
	Daily Max	(report)	58	86	44
BOD (mg/l)	Monthly Avg	(report)	117	109	75
	Daily Max	(report)	180	170	101

(Source: MDNR 1990)

Table 4-7. Monitoring Data for Outfall 00B (January - March 1992)

Parameter		Permit Limit	January 1992	February 1992	March 1992
Flow (MGD)	Monthly Avg	(report)	0.259	0.242	0.368
	Daily Max	0.9	0.265	0.385	0.508
TSS (mg/l)	Monthly Avg	(report)	13	13	22
	Daily Max	(report)	18	19	28
BOD (mg/l)	Monthly Avg	(report)	16	14	13
	Daily Max	(report)	26	16	15

(Source: MDNR 1990)

4.4.2.2 Lake Superior Mixing Zone Study

As part of the NPDES permit reapplication process, Copper Range Company was required to conduct a 1991 study to describe the dimensions and temporal behavior of the plume of the Mineral River discharge into Lake Superior, which is dominated by the effluent from Copper Range Company's White Pine mine. The results of the study were to be used to determine the need for limitations on dissolved solids, metals, and other pollutants in the discharge from outfall 001. The Copper Range Company study characterized the Mineral River discharge as having TDS concentrations between 2,000 and 3,000 mg/l and chloride concentrations between 1,000 and 2,500 mg/l. Flow and constituent concentrations varied both on a seasonal and daily basis (Copper Range Co. 1992a).

The nearshore area of the lake is dominated by a large bedrock outcrop, just west of the mouth of the Mineral River. The outcrop, as well as associated boulders and cobble, were found to influence the deposition of sand in flats and bars beginning at Gull Point and extending eastward to the Mineral River mouth. The sand shifts radically, periodically covering and exposing areas of bedrock and cobble. The dynamic nature of the bottom is controlled by wind and wave conditions. According to Copper Range Company, the precise configuration of nearshore sand bars influences the initial direction of the Mineral River discharge as it enters the lake (Copper Range Co. 1992a).

According to Copper Range Company, the Mineral River plume generally extends toward the northeast from the mouth for a distance of 500 to 4,000 feet, under the influence of steady eastward currents and longshore drift. The plume is dense and typically sinks to the bottom. Copper Range Company stated that incompletely diluted effluent is often confined to a layer within one foot of the bottom. The plume does, however, occasionally extend vertically throughout the water column within about 500 feet of the mouth, when localized onshore winds pile water up along the shore (Copper Range Co. 1992a).

According to Copper Range Company, most of the water column is generally free of the plume's influence, and the plume does not block the longshore migration of fish populations. In addition, according to Copper Range Company, the concentrations of chlorides in the plume are not toxic to fish (Copper Range Co. 1992a). (EPA chronic and acute water quality criteria for chloride are 230 mg/l and 860 mg/l, respectively.)

4.4.2.3 Environmental Assessment of the Effects of Chloride and TDS Discharge on Lake Superior

As part of the NPDES permit reapplication process, Copper Range Company was required to conduct a 1991 study to assess the local and long-term effects on Lake Superior of the discharge from Copper Range Company's White Pine mine. The Copper Range Company study concluded that the Mineral River discharge was not causing any measurable effects, either locally or over the entire lake. Copper Range Company made this conclusion based on the lack of impact on the benthic community, the insignificant magnitude of any future chloride increase in the lake attributable to mine discharges, and the absence of any detectable effects on local water intakes (from Silver City to Ontonagon) (Copper Range Co. 1992b).

According to Copper Range Company, chloride concentrations in Lake Superior have been constant (at about 1.2 mg/l) from 1885 to the time of the study (1992). The Copper Range Company stated that by far, the largest and most widespread sources of chloride loading to the lake are natural tributary inputs and atmospheric deposition, which combine to contribute an estimated 74 percent of the total loading. Point sources, among them the White Pine mine, were found by Copper Range Company to account for 26 percent (estimated) of the total chloride loading (Copper Range Co. 1992b). As shown in Table 4-5, chloride concentrations in the 20 MGD mine discharge averaged about 1,000 mg/l from January through March 1992.

4.4.2.4 Biomonitoring Testing

The Mineral River supports a fish and macroinvertebrate community year-round. These communities are limited by yearly extremes in hydrologic fluctuations, primarily the seasonal low flows. A June 1989 investigation by the Michigan Department of Natural Resources' Surface Water Quality Division found a diverse fish community both upstream and downstream of the Perch Creek confluence with the Mineral River (i.e., where the White Pine mine discharge enters the River). While the MDNR described the overall macroinvertebrate abundance and diversity as low, the macroinvertebrate community was much reduced downstream of the confluence as compared to the community upstream (MDNR 1989).

The reduction in macroinvertebrates downstream of the confluence may not be solely attributable to the effluent introduced via Perch Creek. Major physical and natural stream quality differences (i.e., slope, elevation, water velocities) exist along the Mineral River in the White Pine area. Below the Perch Creek confluence, the River is more susceptible to flow fluctuations and scouring during periods of high precipitation and runoff. These natural extremes in combination with heavy natural clay deposits on all stream substrates are thought be responsible for reducing the diversity of macroinvertebrates in the Mineral River (MDNR 1989).

As part of the NPDES permit reapplication process, Copper Range Company was required to perform acute and chronic biomonitoring on samples of the discharge from outfall 001. Cerodaphnia dubia (Cerodaphnia) and Pimephales promelas (fathead minnows) were selected as the test organisms for the sampling and analysis that were performed in April, June, August, and October 1991. The results of Copper Range Company's biomonitoring are summarized in Table 4-8.

Table 4-8. Results of Copper Range Company Biomonitoring Testing (April - October 1991)

Report Date	Cerodaphnia dubia 96-hour LC50	Cerodaphnia dubia 7-day NOEC	Pimephales promelas 96-hour LC50	Pimephales promelas 7-day NOEC
April 1991	100 percent effluent (no acute toxicity)	100 percent effluent (no chronic toxicity)	100 percent effluent (no acute toxicity)	100 percent effluent (no chronic toxicity)
June 1991	100 percent effluent (no acute toxicity)	100 percent effluent (no chronic toxicity)	100 percent effluent (no acute toxicity)	100 percent effluent (no chronic toxicity)
August 1991	Acute toxicity at 62 percent effluent	Chronic toxicity at 25 percent effluent (survival) and 12.5 percent effluent (reproduction)	100 percent effluent (no acute toxicity)	100 percent effluent (survival) and Chronic toxicity at 50 percent effluent (reproduction)
October 1991	Not Valid	Not Valid	100 percent effluent (no acute toxicity)	100 percent effluent (no chronic toxicity)

The LC50 represents the highest concentration of effluent (diluted with water) at which less than 50 percent organism mortality is observed. The no observed effect concentration (NOEC) represents the concentration of effluent (diluted with water) in which no chronic toxicity is observed (i.e., there were no observed differences between the test sample and the control sample).

Table 4-8 indicates that, during August 1991 sampling, acute and chronic toxicity were observed in diluted effluent from outfall 001.

4.4.3 Ground Water Monitoring

Based on Michigan State law, Copper Range Company is not required to conduct ground water monitoring in any area at the site, including the tailings impoundments. It should be noted that the uppermost aquifer (25 feet) underlying the facility recharges Lake Superior.

4.4.4 Air Permits and Monitoring

The Minnesota Air Pollution Control Commission has issued permits for air use equipment operating within the facility's Power Plant, Rotary Kiln Concentrate Dryer, Electrolytic Refinery and Nickel Sulfate Plant.

Although the smelter stack is the primary emission source at the site, the smelter facility was specifically "grandfathered" from the permit process under State law. The only requirements are that the opacity of emissions at the site must be less than 20 percent and particulate emissions must not exceed limitations specified by State law.

In 1969, Copper Range Company began conducting voluntary continuous monitoring for ambient sulfur dioxide (SO_2) and particulates at four locations (1 mile north and south, 3 miles east, and 6 miles west of the plant). Monitoring data were submitted to the State Air Quality Division on a monthly basis. When the plant closed in 1984, the State Agency granted the cessation of ambient monitoring activity.

The White Pine Refinery is unusual in that there is no bleed stream to remove impurities from the electrolyte during the electrowinning process. However, impurities are removed via the anode slimes which collect at the bottom of the cells and are further processed for sale as a byproduct. Similarly, nickel is removed from the process as nickel sulphate (in the Nickel Sulfate Plant) and is sold as a byproduct. Periodically, specially enclosed (Liberator) cells are used to adjust the copper content within the electrolyte. In the event of abnormal conditions, during such operations, an arsine gas detection device coupled to the liberator cells immediately shuts down the operation including the generation of arsine gas from the liberator cells. Abnormal conditions rarely occur during the operation of the liberator cells.

In 1986, EPA Region 5 required that the facility begin analyzing/reporting the inorganic arsenic content of the smelter matte under 40 CFR Part 61 (National Emission Standards for Inorganic Arsenic from Primary Copper Smelters). Arsenic is a volatile gas that may be liberated during the smelting process and released to the environment via smelter stack emissions.

The arsenic data collected by Copper Range Company are sent to both Region 5 and the State. Due to low levels of arsenic detected in matte samples during 1986-1987, Copper Range Company requested that the Region allow the facility to cease monitoring. According to Copper Range Company, no response was received and monitoring continues. Samples are collected and analyzed during each shift and compiled into monthly composite reports. Table 4-9 below summarizes arsenic monitoring conducted by Copper Range Company for 1991 (Copper Range Co. 1992c). Smelter stack emissions of arsenic are not monitored by Copper Range Company, and it is not clear how the matte data relate to arsenic air emissions.

Table 4-9. 1991 Annual Arsenic Monitoring Results

Month	Matte Inorganic Arsenic Analysis (%)	Converter Input Rate (Kg As/Hr)
January	0.047	6.3
February	0.049	7.4
March	0.048	6.4
April	0.051	6.4
May	0.040	5.1
June	0.028	3.9
July	0.030	4.1
August	0.024	36
September	0.022	3.2
October	0.020	3.3
November	0.021	3.3
December	0.028	4.0`

(Source: Copper Range Co. 1992c)

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APPENDIX 4-A

COMMENTS SUBMITTED BY COPPER RANGE COMPANY ON DRAFT SITE VISIT REPORT

[Comments on the draft site visit report were submitted by Copper Range Company in a letter dated December 21, 1992. This letter is not reproduced for this electronic version. Copies may be obtained from U.S. EPA, Office of Solid Wastes, Special Waste Branch.]

APPENDIX 4-B

EPA RESPONSE TO COMMENTS SUBMITTED BY COPPER RANGE COMPANY ON DRAFT SITE VISIT REPORT

EPA Response to Comments Submitted by Copper Range Company on Draft Site Visit Report

EPA has revised the report to incorporate all but one of the comments and suggestions made by Copper Range Company. Copper Range suggested that EPA delete the site visit team's observation that inert wastes are temporarily managed in a landfill between the mill and the tailings impoundment. EPA has retained this observation in the final report. In some cases, EPA made minor changes to wording suggested by Copper Range Company in order to attribute the changes to Copper Range Company or to enhance clarity.