RECORDS FILE COPY

OPEN FILE

1/1/115

August 1971
R.B. 71/147
DM 652/69

SOUTH AUSTRALIAN GOVERNMENT DEPARTMENT OF MINES

Amdel Report

No.793

EVALUATION OF KAPUNDA ORE

Part II

by

L.H. Goldney

Investigated by: Chemical Metallurgy Section
Officer in Charge: J.E.A. Gooden

F.R. Hartley. Director

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES
Adelaide South Australia

CONTENTS

SUMMARY

Background

In January 1969, the South Australian Department of Mines requested Amdel to examine the possibilities of treating the remnant ore (approx. 6 million tons at 0.7% Cu) from the copper deposits at the old Kapunda Mines in South Australia. The treatment methods were to include flotation or any other method that could be shown to be economically attractive.

Flotation studies were conducted at Amdel and reported in Amdel Report No.765. In addition, an article published in World Mining, October 1968, entitled "Bacterial Leaching Recovers Copper in Highland Valley" had created interest as being possibly applicable to Kapunda ore and it was decided that the author should inspect this operation in British Columbia,along with other operations in the Arizona and Utah areas of the United States,to study current methods of extraction.

The report covers the information gathered on this visit.

Objective

The primary objective of the overseas visit was to study treatment methods which could possibly be applied to the Kapunda ore. In general a survey of all current processing methods and such new technology as could be applicable was also required.

General Observations

The first impressions gained on this tour were those of the massive scale of some operations and the relatively low grade of ores being economically processed. Mining costs have been reduced by using large-scale equipment and all treatment plants inspected were located to allow gravity disposal of tailings. With water at a premium in most areas, water is recovered primarily from the flotation plant tailings thickeners but also from the thickener underflow tailings dams.

Although the production of copper, in the main, is by conventional milling, flotation and smelting, an increasing amount is being produced by acid leaching followed by cementation or solvent extraction and direct electrowinning. Most ores are relatively hard, with low carbonate content, and are particularly amenable to acid leaching by percolation in large dumps.

Although copper dissolution is slow and the ultimate recovery may only be 50%, most of the low-grade ores that are dump leached could not be economically processed by established conventional methods. Dump leaching is successfully applied to sulphide and silicate ores, in addition to oxides and carbonates. Bacterial assistance is obtained in some instances.

A large proportion of the copper is recovered from acid leach liquors by cementation in vats or launders but for large-scale operations, Kennecott's patented continuous cone precipitators could well be applied to give a higher grade cement copper with a lower iron consumption. An increasing interest is being shown in solvent extraction from leach liquors followed by electrowinning from the upgraded solution to produce high grade cathode copper. This process has been proved economical and eliminates smelting and refining which have associated air pollution problems. Air pollution is a major problem in some areas and current thinking is that hydrometallurgical extraction methods will be used increasingly.

Summary of Operations

Types of Ore. In Arizona the oxidised copper is predominantly chrysocolla with some malachite and azurite, while the sulphide copper is generally chalcocite and chalcopyrite with some bornite at Bagdad and other northern areas. Further north in Utah, Kennecott mines essentially sulphide ores. The British Columbia operations are also mainly with sulphides with predominantly bornite at Ashcroft and chalcopyrite at Merritt and Britannia Beach. Ore grades are low, generally with less than 1% copper.

Mining. All operations are open pits which in many cases will eventually extend to depths of up to 900 ft. Kennecotts open pit, one of the largest in the world, is about a half mile deep. Some orebodies extend to depths of several thousand feet and underground mining may be considered later but at present the economics are doubtful, considering the ore grades. The open cuts are usually in 50-ft benches.

Some areas have known reserves for up to 100-years production. Mines in Arizona and Utah with limited higher grade reserves will continue to operate by dump leaching already mined lower grade stockpiles.

Mining is generally on a large scale with tonnages ranging from about 15,000 tons per day to nearly 400,000 tons per day at Kennecott in Utah. A high degree of mechanisation is evident and the equipment used in some locations is massive. The overburden plus dump material strip ratios are in the range of 1 to 2 tons strip per ton of ore for milling, although in most cases the strip is not necessarily waste material as much is subsequently leached in dumps.

Milling. The copper content of ore to the sulphide flotation circuits is in the range 0.4 to 1.0% Cu. A small percentage of molybdenum is associated wiht some ores and is a bonus.

There are generally three stages of primary crushing, usually with gyratories, followed by rod and ball mills in closed circuit.

Flotation. Most flotation cells are mechanically agitated types, of capacities up to 100 cu ft, although one plant, Bagdad Copper, has 300 cu ft cells. Another plant, Miami Copper, has air agitated troughs 50-ft long (Hunt air lift cells). Concentrate grades are usually in the range 25 to 30% Cu although Bethleham Copper produces 35 to 37% Cu concentrates from bornite ores. Where molybdenum is recovered, a bulk copper molybdenum concentrate is produced followed by refloating for molybdenum with copper depression.

Smelting and Refining. Concentrates from the majority of plants either go to a custom refinery or are exported. Two plants visited, Inspiration in Arizona and Kennecott in Utah, have their own smelters with reverberatories and converters, the blister copper being electro-refined on site.

Vat Leaching. Vat leaching is operated at Inspiration, Arizona for the recovery of copper from oxide ores containing up to 1% copper. The ore is crushed to minus ¼ inch, deslimed, then leached in a series of vats to give a pregnant liquor containing more than 20 grams Cu per litre with a leaching efficiency of approximately 90%. This solution goes to electro-winning.

Dump Leaching. Dump leaching operations are on a large scale, accounting for 25 to 30% of the total copper production in Arizona. In Utah, 25% (400,000 lb Cu/day) of Kennecott's total production is from this source. Operations are minor in British Columbia.

Oxidised copper ores and low grade sulphide ores (ranging from 0.2 to 0.5% Cu), are leaching in heaps which are up to 300 ft deep. These are surface-irrigated with dilute sulphuric acid solution which is recycled from the copper recovery plant. The heap bases are generally compacted natural terrain with no sealing. Some are located in gullies with hard rock bases.

Until recently copper was recovered from all dump leach liquors by cementation and most plants marketed a cement copper. Early in 1969, Ranchers in Arizona installed a full scale solvent extraction circuit and two other plants in Arizona have pilot installations.

Cementation. Pregnant liquor from the dumps usually contains about 1 gram

Cu per litre and is passed through a series of vats or launders containing shredded de-tinned cans suspended on stainless steel screens. The cement copper contains beteen 75 and 80% Cu and iron consumption is in the range 1.5 to 2.5 lb Fe per pound Cu. The product is dewatered either by filtration or by draining, followed by solar evaporation. Inspiriation, Arizona, upgrade their cement product by flotation to approximately 90% Cu.

Kennecott have developed and patented continuous cone precipitators which produce a higher grade cement copper containing 85 to 90% Cu with an iron consumption of about 1.5 lb Fe per pound Cu. The units produce 400,000 lb Cu per day using selected shredded non-rusty cans. A new type of cone has bee developed which will use powder iron. Operating with copper solutions with low ferric iron content, the powder iron consumption is expected to be about 1.2 lb Fe per pound Cu.

Hydrogen Reduction. Only one company, Bagdad Copper, Arizona, operates this process to produce high grade (99.98 to 99.995% Cu) copper powder by hydrogen reduction under pressure of a solution of cement copper in sulphuric acid. Excessive plant downtime through reactor corrosion and physical plugging of process lines with copper powder make the economics extremely doubtful.

Solvent Extraction. An increasing interest is being shown in the recovery of copper from dump leach liquors by solvent extraction, followed by electrowinning, to produce 99.995% cathode copper. Ranchers, Arizona, produce 30,000 lb Cu per day operating at a dump leach liquor flowrate of 1100 US gallons per minute. A 4000 US gallons per minute plant is being installed at Bagdad, Arizona.

Research. Both Kennecott and British Columbia Research are involved in independent studies of leaching with the assistance of bacteria. The US Bureau of Mines, Salt Lake City Division, is primarily involved in copper projects with secondary interest in the recovery of uranium from copper barren liquors and the recovery of sulphur from dilute smelter gases. At the University of British Columbia there has been considerable work on pressure leaching of copper concentrates with the recovery of elemental sulphur as an essential part of the process, but no immediate commercial application is seen.

Application to Kapunda Ore

Correspondence failed to locate the operation, Copper-Can Development Ltd, described in the article "Bacterial Leaching Recovers Copper in Highland Valle and by personal contact with British Columbia Research in July 1969, this operation was found to be non-existent. However, British Columbia Research are proceeding with experimental work on bacterial leaching of sulphide ores in agitation systems. To date no commercial application had been established.

Dump leaching with bacterial assistance is an established operation both in the United States and British Columbia but the type of ore must be amenable to percolation. From studies of Kapunda ore drill samples and of exposed faces of the old open-cut workings, percolation leaching would not be possible, due to the high clay content of the orebody. Unless there is a breakthrough in extraction methods for ores which are predominantly sulphide, by agitation leaching with acid with or without bacterial assistance or by other means, conventional methods such as flotation should only be considered at this stage. The treatment of Kapunda ore by flotation gave encouraging results, as reported in Amdel Report No.765. Further investigation using this technique is warranted.

1. INTRODUCTION

With the increasing interest shown in smaller scale copper operations in South Australia, a further investigation into the possibilities of treating the remnant ore from the old Kapunda Mines was considered warranted. The deposit is estimated to contain approximately 6 million tons of ore of 0.7% copper, approximately two-thirds being in the sulphide form.

In addition to the flotation studies reported by Weir (1971)*, a study of the extraction technology used by copper producers and research organisations in Arizona, Utah and British Columbia was undertaken by the author. The report contains the information gathered on this tour.

2. RÉSUMÉ OF VISITS

2.1 Arizona

All visits in Arizona were arranged by Mr Eldon DeMent of General Mills Incorporated, 2030E. Broadway, Suite 217, Tucson, Arizona, 85719 (phone code 602: 622-8891). These were as given below.

Duval Corporation - (June 23, 1969)

4715E Fort Lowell Road, Tucson, Arizona (phone code 602: 327-5941).

Contact: Messrs Ron Brown and Gordon Agar, Research Engineers.

Visit to Esperanza and Siereta mines conducted by Mr Brown.

Operations Inspected: Mining, milling (15,000 tpd), flotation, dump leaching, cementation, pilot solvent extraction and new 80,000 tpd mill in construction.

Pima Mining Company - (June 24, 1969)

Nogales Highway and Pima Mine Road, Tucson, Arizona (phone code - 602: 792-1200).

Contact: Mr Clem Chase, Director of Research.

Operations Inspected: Mining, milling (40,000 tpd), flotation, tailings line, dams and water recovery systems, experimental dump leaching.

American Smelting and Refining Company - Mission Unit (June 24, 1969)

West Pima Road, Tucson, Arizona.

Contacts: Messrs Mike Mason, and Don Wilson, Metallurgists.

Operations Inspected: Milling (24,000 tpd), flotation.

---

* Weir, L.J. (1971). Evaluation of Kapunda Ore - Part I, Amdel Report 765, March.

793

Ranchers Exploration & Development Corp. - (June 25, 1969)

Bluebird Mine, Miami, Arizona.

Contact: Mr Ken Powers, Plant Manager

Operations Inspected: Dump leaching, solvent extraction and electro-winning.

Inspiration Consolidated Copper Company - (June 25, 1969)

Inspiration, Arizona

Contact: Mr Bill Mitchell, Director of Research

Operations Inspected: Flotation (25,000 tpd ore), vat leaching (10,000 tpd), cementation, smelting, electro-winning and refining, wire drawing plant.

Miami Copper Company - (June 25, 1969)

Copper Cities Mine, Miami Arizona

Contact: Mr Wayne Gould, Metallurgical Superintendent

Operations Inspected: Milling, flotation, dump leaching, cementation.

Bagdad Copper Corporation - (June 26, 1969)

Bagdad, Arizona

Contact: Mr Ed. Howell, Director of Research

Operations Inspected: Milling (6,000 tpd), flotation, dump leaching, cementation, hydrogen reduction, pilot solvent extraction.

2.2 Utah

The visit to the US Bureau of Mines was arranged by personal contact with Mr Rosenbaum. Mr D'Arcy George, USBM arranged the Kennecott contact.

Kennecott Copper Corporation - (July 7 & 11, 1969)

Research Centre, 1515 Mineral Square, Salt Lake City, Utah (phone Utah 322-1533).

Contacts: Dr H. Rush Spedden, Director of Research
Mr Ed. E. Malouf, Project Development Engineer
Mr A. Ed. Back, Assistant to Director of Research

Operations Inspected: Open pit mining (400,000 tpd), dump leaching, cone precipitators, research centre.

US Bureau of Mines - (July 3, 7, 9 & 10, 1969)

Metallurgical Research Centre, 1600 East First South Street, Salt Lake City, Utah (phone Utah 524-5350).

Contacts: Mr Joe Rosenbaum, Director of Research
Mr Carl Schack, Assistant Director
Messrs D'Arcy George, Bill McKinney, Dick Ross & others.
Operations Inspected: Pilot scale dump leaching of copper, uranium recovery from copper barren liquors, sulphur recovery from dilute smelter gases and discussion of other copper projects.

2.3 British Columbia

Some visits in British Columbia were pre-arranged by correspondence, others were by personal contact on arrival.

British Columbia Research - (July 14, 1969)

3650 Wesbrook Crescent, Vancouver, BC (phone BC224-4331)
Contact: Dr Douglas W. Duncan, Biology Division
Operations Inspected: Bacterial leaching of sulphide ores.

The University of British Columbia - (July 15, 1969)

Department of Metallurgy, Vancouver 8, Canada.
Contacts: Dr Ian H. Warren, Professor Ernest Peters
Operations Discussed: General discussion of activities.

Bethlehem Copper Corporation Ltd - (July 16, 1969)

PO Box 520, Ashcroft, British Columbia.
Contacts: Mr Tom Liss, General Manager
Mr Cliff Overton, Manager Plant Production
Mr John Smith, Manager Property Engineering
(Mr Smith conducted the tour of operations).
Operations Inspected: Mining (45,000 tpd), milling (15,000 tpd), flotation, water recovery from tailing.

Craigmont Mines Limited - (July 18, 1969)

Placer Development, 700 Burrard Building, 1030 West Georgia Street, Vancouver 5, BC (phone BC 682-7082)
Contact: Mr J.H. Eastman, Manager Engineering & Construction
Operations Discussed: Mining, milling (5,000 tpd), flotation (Operations at Merritt, BC)

Anaconda Company - Britannia Beach Mine - (July 18, 1969)

Britannia Beach, British Columbia (phone - Britannia Beach 896-2221)
Contact: Mr Jack Anderson
Operations Discussed: In situ leaching, cementation.

3. PLANT DETAILS

The operations inspected at the various plants have been given in Section 2. As the scale and type of operations vary at these plants, individual details are presented in this Section. Weight units are generally in short tons (2000 lb) and volume units are US gallons (0.8 imperial gallon) both in the United States and Canada.

The Arizona and Utah areas are relatively arid with annual rainfalls of the order of 10 inches, and with a high evaporation rate. Process water is therefore at a premium in most areas, particularly in Arizona where the summer temperatures consistently exceed 90°F, with periods in the range 100 to 115°F.

With the extensive road travel involved to some areas, the time at some plants was limited to several hours and it was difficult to obtain a complete cover of both mining and treatment operations at all locations.

3.1 Duval Corporation, Arizona

Current mining is from the Esperanza and Siereta open pits and it was anticipated that the Siereta mine would be in full production when a new 80,000 tpd flotation plant was commissioned in January, 1970.

Mining. The ore is mined in 50 ft benches and loaded with 15 cu yd shovels into 120 ton trucks. The higher grade sulphide ore goes to milling and flotation at the rate of 15,000 tpd of average grade 0.4 to 0.5% Cu and approximately 0.2% Mo.

Milling. After primary size reduction, the ore is milled in 3 rod mills followed by 4 ball mills in closed circuit. All mills are 12 ft diameter by 16 ft long.

Flotation. Flotation cells are Denver and Agitair of 100 and 60 cu ft capacities. A bulk copper and molybdenum concentrate is produced which then goes through a steaming process to remove flotation reagents. The primary concentrate is then selectively floated for molybdenum with copper depression. The concentrates are filtered on disc vacuum filters.

Approximately 100,000 pounds of copper is produced daily as a 25% concentrate. The molybdenum sulphide concentrate is roasted in a multi-stage rabble arm furnace to molybdenum trioxide.

Tailings from the mill go to 150 ft diameter thickeners for water recovery, the underflow flowing by gravity to the tailings dam. The thickeners have no underflow pumps.

Dump Leaching. Several dumps are operating giving a total flow to cementation of approximately 1000 gallons per minute of 1.3 g per litre Cu liquor. The ore is as mined and contains lumps up to 3 ft. The copper content is 0.1% or less and is mainly sulphide. Although not cultivated, bacteria are present and all new dumps are started with some liquor from old dumps. The overall copper recovery is approximately 60%.

One dump inspected was about 200 ft deep. This was surface sprayed from irrigation type plastic sprinklers set at 60 ft centres and each delivering 3 gallons per minute from a 40 psi liquor supply. The sprinklers were connected to a network of 2 inch plastic pipes and each spray covered an area of approximately 3000 sq ft. The top feed is barren liquor from cementation, the flow rate to this dump being 200 gallons per minute with approximately 150 gallons per minute recovery from the base. The main volume loss is by surface evaporation, there being little soakage loss at the dump base. The spray liquor acidity is adjusted to give a product liquor pH of approximately 2.0, to avoid Fe precipitation in the liquor lines. Acid consumption is low, most of the requirement being produced from the sulphide ore. Sulphuric acid prices were highly confidential at all mines but a figure of approximately $35 per ton delivered can be assumed for the Tucson area.

Cementation. Pregnant liquor from the dumps goes to a storage-settling dam prior to cementation. The liquor pH in is 2.0 to 2.5 and out at 3.0 to 3.5. The liquor passes through a series of vats containing shredded de-tinned cans ($US55/ton) suspended on stainless steel screens. The liquor flow is down through the first vat, up through the second, down through the third etc. Cement copper is discharged to sealed aprons for drying by drainage and solar evaporation prior to sale.

Solvent Extraction. A separate small dump of approximately 1.5 million tons of ore is reserved to supply 100 gallons per minute of pregnant liquor to the pilot scale solvent extraction circuit. No change in leaching efficiency has been detected through any possible effect on the bacteria of entrained or dissolved solvent in the recycle liquor.

The circuit has four extraction and three stripping stages and except for the mixer turbines, is similar to the General Mill laboratory unit design (ie. the agitator pumps both aqueous and organic into the mixer). The units were originally designed with high speed agitators but Mr Eldon de Ment (General Mills) changed these to slow speed turbines after seeing Amdel's design in operation at Rum Jungle. Phase disengagement is better using slow speed turbines and solvent losses due to entrainment have been decreased. The tanks are of concrete construction with plasticised PVC lining. The liquor concentrations were:

Solvent (mixture of LIX64 and LIX64N)	4.5 to 5.0% LIX in kerosene	
Pregnant liquor	1.3	g/1 Cu
Raffinate	0.02	g/1 Cu
Loaded organic	0.8 - 0.9	g/1 Cu
Strip solution	150	g/1 H_2SO_4
High grade strip	40	g/1 Cu

The high grade strip goes to an electro-winning circuit then is recycled to the solvent extraction strip circuit. The barren liquor raffinate is recycled to the leaching dump.

The solvent loss is approximately 0.06 gallon mixed solvent per 1000 gallons of raffinate. With LIX at $US2.5 per pound ($US1000/400 lb drum), the solvent extraction reagent operating cost is approximately US0.6 cents per pound of copper.

The cost of the 100 gpm pilot plant was approximately $US200,000 including the electrowinning tank house.

3.2 Pima Mining Company, Arizona

The company is jointly owned by Cyprus Mines Corporation (50%), Utah Construction and Mining Company (24%) and Union Oil Company (25%). The mine pit boundary connects with the Mission pit of American Smelting and Refining Company.

Mining. The mining rate is approximately 80,000 tpd with 40,000 tpd of sulphide ore going to the flotation circuit.

Milling. The head grade is adjusted to approximately 0.5% Cu by ore blending from a 0.35% Cu stockpile when the mining grade is higher.

Crushing is in three stages, all gyratories. Ten rod mills and 20 ball mills in closed circuit with cyclones are installed.

A mill section comprises 1 rod mill, 2 ball mills with a rougher section of 6 banks of 10 cells and a cleaning section of 3 banks of 10 cells which services 2 rougher sections. The cells are 66 cu ft Fagergren.

The flotation agent is Dow Z6. Some reagent is added to the ball mills but there are no separate conditioners. The pulp density is 35 to 40% solids in the rougher. Lime is added to pH 11.5. Scavenger concentrates from the rougher sections and cleaner tailings are reground.

Approximately 50% of the total water requirement is recovered from the tailings thickeners with a further 10 to 25% coming from the tailings dam via decant towers.

Tailings Line. There is a considerable fall from the tailings thickeners to the tailings dams and drop boxes with vertical drops of 6 to 8 ft are installed at intervals along the whole line. The line slope is 0.53% and concrete pipe sections of diameter approximately three times the line diameter are used for the drops. These sections have side entry holes near the top and side exit holes near the bottom. After positioning, the bottom and top of the drop box is sealed with concrete. The exit position is located so that a layer of solids will remain on the base of the drop box to minimise concrete wear. The older lines have the drop box pipes installed vertically but the later lines have a box angle of approximately 70 degrees to the horizontal.

Dump Leaching. Large dumps containing 0.6% total Cu with approximately 0.3% as oxide copper, predominantly chrysocolla, are available for dump leaching. Laboratory tests in 5-inch glass columns have shown that over 50% of the total copper can be recovered by acid leaching. The acid consumption is about 10 lb H_2SO_4 per pound of copper and the acid addition is adjusted to give a pH of 1.5 to 2.5 in the final pregnant liquor. The laboratory columns are in series with alternate down and up flow. All columns have CO_2 vents.

Other oxide type dumps have up to 2% $CaCO_3$ and will require a high acid addition unless in-situ acid can be developed from the sulphides present.

3.3 American Smelting and Refining Company — Mission Unit, Arizona

Mining. No mine visit was made. Oxidised copper ore containing 0.9% total Cu is stockpiled for later dump leaching. Sulphide copper containing 0.7 to 0.8% Cu and 0.02% Mo goes to the mill at the rate of 24,000 tpd.

Milling. Primary size reduction is with three stages of gyratory crushers. Allis Chalmers rod and ball mills, 10.5 ft dia. by 15 ft long driven by 900 hp synchronous motors are installed for fine grinding (details of mills - page 8).

The re-grind mills are 6 ft dia. by 13 ft long and produce an 80% minus 325 mesh (Tyler) product.

Each ball mill operates in closed circuit with four Krebs D20B cyclones which are 20 inches in diameter with a 2.75 inch apex and 7 inch dia. by 14 in.

vortex finder. The slurry solids concentrations in the cyclone circuit are: feed 55%, overflow to flotation circuit 32 to 40% and underflow to re-grind 74%.

The fine grinding mill details are:

	Speed		Liners	Rod/ball Consumption per ton ore	Grind Size mesh Tyler	Solids % Concentrate
	rpm	% of critical				
Rod	15.8	65.5	Single wave Cr-Mo and Ni-hard	0.7 lb of 4 in. rods	14% + 8	74
Ball	16.7	69.3	Double wave Ni-hard	1.5 lb of 2 in. balls (forged and cast)	12% + 65	72

Flotation. Lime is added to the rod mills and the cleaner regrind mills at rates of 0.15 lb per ton solids and 0.5 lb per ton solids respectively. Other reagent consumptions are:

		lb/ton ore
Float reagent	Cyanamid zanthate Z6	0.015
Promoter	Dow Aerofloc 238	0.02
Frother	Pine oil	0.04

The rougher cells are 60 inch Wemco and Fagergren and the cleaners 24 inch Denver. Molybdenum is recovered from the copper-molybdenum concentrate by re-floating, with copper depression by sodium sulphite. Molybdenum recovery is approximately 35% of the feed. Copper recovery is 91% of the feed with a concentrate grade of 28% Cu.

Water is recovered in 3 by 275 ft diameter tailings thickeners. The middlings thickeners are 125 ft diameter.

3.4 Ranchers Exploration and Development Corporation, Arizona

This operation is entirely dump leaching, solvent extraction and electrowinning, producing 30,000 lb per day of 99.995% copper cathode. The ore is mainly chrysocolla, with some malachite and azurite, and contains about 0.5% Cu.

Dump Leaching. Ripped ore is heaped in layers 20-ft thick. The first 20-ft bench is leached for approximately 60 days then another 20-ft layer is added and leaching continued for a further 30 days. Further 20-ft layers are added with intermediate leaching until the heap is approximately 200-ft deep. The heap bases are generally unsealed but a cement mix is occasionally used to compact the base in porous areas. There are eight areas leaching with a total liquor feed of 1200 gallons per minute at 50 grams H_2SO_4 per litre and an off-take pregnant liquor volume averaging 1100 gallons per minute. One typical heap had a surface area of 60,000 square feet and was irrigated at the rate of 200 gallons per minute.

Each area feeds from a 4-inch main from which 2-inch PVC pipes, at 90° to the main, are connected at 8 feet centres. The 2-inch pipes have small plastic needle valves (costing 5 cents each) at 2 feet intervals and these are manually adjusted to give approximately uniform flow over the whole area. The heap surface is prepared with furroughs between each 2 inch line. After leaching the first 20-ft layer, the surface is allowed to dry and is re-ploughed before adding the next layer. The leaching efficiency is approximately 40% and the total acid consumption in the whole process is 5 to 7 lb H_2SO_4 per pound of copper.

Before each heap is constructed, two 6-inch diameter PVC pipes, perforated with ½-inch holes, are laid in trenches the full length of the heap and covered with coarse gravel.

Solvent Extraction. Liquor from the dumps goes to a storage dam then to a precoat pressure filter. Liquor clarity from the storage area is good and normally the filter would not be required but is on line as a safeguard. The filter is followed by a wooden 50,000 gallon storage tank which gives holding capacity between filter cleaning and pre-coating cycles. A steam heated heat exchanger is installed for liquor heating prior to extraction. This appears quite unnecessary and is not normally used.

The circuit consists of three extraction stages and two stripping stages. The extraction mixers are 15,000 gallon capacity (nominal 2 minute retention time) and the settler area is 1 sq ft per 2 gallons per minute total flow. The settlers are lined concrete tanks while the mixers are separate circular tanks with pipe connections to the settlers. The mixers are not well designed (built by Bechtel Corporation - General Mills were not involved) in that the inlets and outlet are poorly positioned, giving the possibility of short circuiting. However, as the plant is producing to design capacity, Ranchers are not interested in a shut down for modifications.

At the time of the visit the pregnant liquor feed contained approximately 4 grams per litre Cu and 5 to 10 grams per litre H_2SO_4 and the flow rate was 1100 US gallons per minute (approx. 880 Imp. gallons per minute). The organic which is a 6 to 7% solution of mixed LIX 64 and 64N in power type kerosene (flash point 173^{o}F) is loaded to 1.2 grams per litre Cu. The solvent loss in raffinate is 0.12 gallon mixed solvent per 1000 gallons of aqueous which is higher than the Duval pilot plant. Mixed solvent costs about $US1.70 per gallon (US). The strip sulphuric acid circuit produces approximately 30 grams per litre Cu solution which goes to electro-winning. The return solution from the tank house contains 26 to 27 grams per litre Cu.

Electro-winning. In this circuit there are 48 tanks, in series electrically, with 40 cathodes per tank. The cathodes are thin sheets of copper weighing about 15 lb initially. These are pulled on a regular cycle, 6 tanks per day, when the cathode weight is approximately 150 lb. The ½-inch thick anodes are antimonial lead and the weight loss is 10 to 15% per year, or approximately 1 lb Pb per 100 lb Cu produced. The tanks are covered with a layer of kerosene to stop misting. Current efficiency averages about 80% but up to 90% has been obtained. The current density is approximately 22 amp per sq ft.

Costs and Capacity. The filtration, liquor heating, solvent extraction plant and the electro-winning circuit cost between 2.5 and 2.75 million $US. The solvent inventory amounts to $US180,000. An installed figure of $US3 x 10^6 was published in World Mining, April, 1969.

The design capacity was 5000 tons Cu per year. The actual average production rate is 36,000 lb Cu per day or 4800 tons Cu per year, assuming 300 days per year of operation. The approximate labour force is: mining 30, processing 20 and maintenance 25.

3.5 Inspiration Consolidated Copper Company, Arizona

This plant has a wide range of operations including vat leaching, cementation, flotation, smelting, electro-winning and refining and wire drawing. No mine visit could be made in the limited time available.

Vat Leaching. 10,000 tpd of oxide ore (up to 1% Cu) is leached in a series of thirteen rectangular lead lined concrete tanks without agitation. The ore is crushed to minus ¼ inch, the slimes being removed and sent to a flotation circuit. Each tank holds 10,000 tons of ore and six tanks are leached in series, with up-flow of liquor, for a period of 4 days, followed by a 4-day washing cycle, then 2-days' draining followed by a dig-out period of 1 day. Loading is by a mobile belt conveyor and a traversing excavator is used for emptying the vats. A pregnant liquor of 20-plus grams per litre Cu is produced with a leaching efficiency of approximately 90%. This solution goes to electro-winning.

Cementation. Dilute copper liquors from various sources including 3 grams per litre Cu washings from the leaching vats, go to cementation where shredded cans are used. The cement product is approximately 75% Cu and this is upgraded by flotation to approximately 90% Cu.

Flotation. 25,000 tpd of sulphide ore (0.75% Cu, 0.01% Mo) are floated with a copper recovery of about 85% with a concentrate grade of 30% Cu. The ore is relatively soft and ball mills only are used for grinding. The feed to the roughers is approximately 35% solids and a 10% Cu rougher concentrate is produced. This is up-graded to 30% Cu in two cleaning stages. The copper concentrate goes to an eight stage cleaning circuit where molybdenum is floated as a 55% Mo concentrate. The copper tailing is filtered and dried to a moisture content of 10%. This product goes to the smelter.

Smelting. Mixed copper concentrates including custom ores, ranging in grade from 20 to 25% Cu are smelted at the rate of 1000 tons of concentrates per day. The matte from the 32 x 110 ft reverberatory furnace goes to the converters which produce a 99.6% blister copper.

Electro-winning and Refining. The electro-winning section takes liquor from the vat leach circuit, the dilute liquor from the tank house, containing approximately 10 grams per litre Cu, being returned to leaching. Anodes are lead containing about 6% antimony. The starting cathode sheets are made by plating copper on stainless steel sheets for 24 hours to give a sheet weight of 15 lb. These are peeled from the stainless steel and carefully straightened by hand. The tanks are pulled on a 7-day cycle when the cathode weight is approximately 250 lb. The tanks have a thin layer of light oil on the surface to minimise misting.

In the electro-refining section the anodes are 99.6% Cu from the smelters. There are 66 anodes and 68 cathodes per tank. The electrolyte is heated to 60°C.

The total production of refined electrolytic copper is about 200 tpd. All tanks are concrete with lead lining.

Wire Drawing Plant. Cathode copper is melted in a gas fired furnace under reducing conditions then formed into bars which are drawn into 5/16 inch rod. Annealed rod is produced at the rate of 1700 ft per minute and is sold direct to electrical cable manufacturers.

3.6 Miami Copper Company, Arizona

The higher-grade sulphide ores of average grade 0.5% Cu go to milling and flotation and the low-grade ores containing about 0.2% Cu which is essentially sulphide, are dump leached with copper recovery by cementation.

Milling. This was the only plant visited using jaws as primary crushers. Second and third stage crushing is with gyratories.

Flotation. All rougher flotation cells are air agitated 50-ft long troughs (Hunt air lift cells). The air pipes are at 4-inch centres and air is supplied at a pressure of 3 psi from centrifugal fans. The cleaner cells are mechanically agitated. The concentrate grade from the roughers is 12 to 17% Cu and from the cleaners, 25% Cu.

Dump Leaching. The dumps are irrigated with orchard type sprinklers set at surface level. Leaching is with bacterial assistance and there is sufficient sulphide in the ore to make acid addition unnecessary. Liquor from the dumps containing less than 1 gram per litre Cu goes to cementation. The cementation pits are stepped to give gravity flow. Liquor flow is horizontal through a series of tanks.

3.7 Bagdad Copper Corporation, Arizona

In addition to flotation, dump leaching, cementation and pilot scale solvent extraction and electro-winning, Bagdad has a hydrogen reduction plant producing copper powder from acid copper solutions. A new 4000 gallons per minute solvent extraction plant should be in operation in January 1970.

Milling. The ore is relatively soft and after primary crushing, ball mills only are used for fine grinding. The 8-9 Marcy mills are in closed circuit with spiral and rake-bucket classifiers - all other plants visited used cyclones. The milling rate is 6000 tpd.

Flotation. The flotation cells are 300 cu ft Wemco and copper concentrate is produced at the rate of 15,000 lb Cu per day.

Dump Leaching. Dumps up to 300 ft deep, containing as-mined ore (0.5 to 0.5% Cu) with lumps up to 3 ft in diameter, are leached. Liquor distribution is by ½-inch flapping hoses connected to 2-inch plastic mains. The dump bases are not sealed. The overall volume recovery of liquor is 94% and the copper content of the pregnant liquor is approximately 1.2 grams per litre. Most of this liquor goes to cementation to produce 84% Cu cement at the rate of 30,000 lb Cu per day.

Solvent Extraction. This pilot circuit is similar to Duval's pilot plant with mixers 3½ by 3½ by 4½ ft deep. Bagdad has oversize (40 in. dia.) slow speed turbines installed. Aqueous and organic streams are pumped by the turbine. The plant was operating with a pregnant liquor flow rate of 60 to 80 gallons per minute (below design capacity) with a 5% LIX64 organic. General Mills will probably change the solvent to LIX64N to improve the plant performance. High grade strip goes to electro-winning.

A 4000 gallons per minute solvent extraction plant with four trains of 1000 gallons per minute is to be installed. This will have four extraction and three stripping stages (Ranchers operate successfully with three extraction and two stripping stages which is General Mills recommendation).

Acid Plant. A 200 tpd sulphuric acid plant is installed. Sulphur is delivered molten in tankers.

Hydrogen Reduction (Chemmetals Corporation Process). Cement copper is dissolved in sulphuric acid and reprecipitated as high grade copper powder, 99.98 to 99.995% Cu, by hydrogen reduction at about 300°F at a pressure of 400 psi. Hydrogen is obtained by reforming natural gas. The copper concentrations of the solutions in and out of the reactors are 90 grams per litre and approximately 20 grams per litre respectively. The powder copper product is filtered, sintered and sized.

The reaction vessels are titanium lined but corrosion is still severe. Plugging of lines with fine copper powder also adds to the operations difficulties. The plant has been installed for about 4 years and although a high grade product can be produced, excessive down time through corrosion and other troubles is making the economics extremely doubtful. The actual production is probably less than 20% of design capacity.

3.8 Kennecott Copper Corporation, Utah

Kennecott have the largest open pit mine in the world at Bingham Canyon near Salt Lake City. The pit is oval shaped, being about 2 miles wide. The ore is essentially sulphide and the plus 0.4% Cu goes to flotation at the rate of approximately 150,000 tpd, the average float feed being about 0.7% Cu. The low grade ore goes to dumps at the rate of approximately 250,000 tpd of average grade 0.2% Cu, ie. 1 million pounds of copper per day. The current production from dump leaching is 400,000 lb Cu per day which is about 25% of the overall production.

The mine is highly mechanised with an electric railway system on all benches. Visits were arranged to the dump leaching and cementation sections only.

Dump Leaching. The ore as mined is dumped in canyons which have near impervious bases. The heaps are several hundred feet high. Liquor distribution is by flooding areas at intervals to eventually give a complete cover of the whole heap. Air is aspirated into the heaps as the liquor flows downward. This is a much simpler system than the net work of sprays or perforated pipes used in the Arizona areas. Liquor may be pumped to one heap for several weeks, then the heap is left for several weeks and another is worked. The heaps yield 50 million gallons of approximately 1 gram per litre Cu at pH 2.6 per day.

The pH of the barren liquor from cementation is adjusted to 2.1, by the addition of 500 tons of H_2SO_4 per day (2½ lb acid per lb of copper recovered), then the liquor is recycled to the heaps. Kennecott have their own acid plants using the richest gases from the copper converters. Leaching is with bacterial assistance - the bacteria operate near the surface of the heaps oxidising ferrous to ferric sulphate. Chemical action presumably takes place lower in the heaps as temperature measurements taken near the heap centres can be of the order of 160°F - too high for bacterial life at this point. The bacteria is a naturally occurring strain which is registered in a US patent. The British Columbia Research Council was engaged for a 3 year research period initially but now Kennecott do all their own research on bacterial action. This is apparently quite an involved study and is part of Kennecott's 'know-how' available at a price. Some nutrients are added to stimulate bacterial growth - phosphate was mentioned as one.

Liquor from the heaps is collected in ponds, which act as settling chambers, and relative clear liquor is pumped to the cone precipitators. Iron is removed from the recycle leach liquor at intervals. The barren liquor from cementation is passed through aerators to oxidise the iron to ferric hydroxide, the precipitate being removed in settling ponds. The relatively iron free liquor is then acidified and recycled to the dumps. The effect of temperature and acidity on copper extraction are discussed in Appendix A.

Cementation. There are two banks each of 13 cones, each cone being enclosed in a wooden tank 20 ft dia. by 20 ft high. The cone design and operation is described in detail in Appendix B - "Use of Cone-type Copper Precipitators to Recover Copper from Copper-Bearing Solution" - Spedden, Malouf and Prater (1966)

The 26 cones are operated in parallel and produce 400,000 lb of cement copper per day with an iron consumption of about 1½ lb Fe per pound Cu.

The iron is detinned (NaOH treated) shredded cans. The cans are shredded in a hammer mill then heated in a small rotary kiln to burn organic matter. Selected non-rusty cans costing between $50 and $60 ton are used. A cement copper containing up to 92% Cu has been produced (92% Cu, 6% O_2, 1% Fe, 1% other insolubles).

Copper slurry from the cone precipitators contains about 5% solids and is filtered in plate and frame presses with stainless steel mesh cloth. The presses operate up to 70 to 80 psi for filtration, followed by a hot air cycle to dry the cake to approximately 20% moisture. An operator opens the presses, with hydraulic assistance, and the cake falls off cleanly into a collection chute. No cloth scraping is necessary.

The cones normally produce a cement copper containing 85 to 90% Cu compared with a launder cement copper containing 75 to 80% Cu.

With the cone installation, the labour requirement is 7 men per shift to produce 400,000 lb Cu per day. In the old launder plant producing only 30,000 lb Cu per day, the labour force was 30 men per day.

New Cone Development. The development of a new cone system is described in Appendix C - "Use of Particulate Iron in the Precipitation of Copper from Dilute Solutions" - Back, 1967. This cone must be operated near volume capacity to maintain fluidisation of the powder iron bed.

The estimated cost for a single cone installation, including cone, liquor storage, filter plant and pumps but excluding liquor pipelines to dumps, is $US100,000.

Mr Back considers that the minimum economic size is an installation to produce about 35,000 lb Cu per day, which would be a two cone installation with 1 gram per litre Cu liquor. Using powder iron and with liquor of low ferric iron content, iron consumption should be about 1.2 lb Fe per pound Cu.

Kennecott are interested in licencing their precipitators and leaching details for a fee. Enquiries should be directed to:

John Sniadow
Director of Licencing and Patents
Kennecott
New York.

3.9 US Bureau of Mines, Utah

The US Bureau of Mines is financed by the Federal Government and the

Metallurgical Research Centre in Salt Lake City is engaged in long-term research projects. The projects are usually of 1 to 10 years duration, the average being about 4 years. There is no contract industrial work and industrial problems are only undertaken if they are of general application, the results being publicly available.

Some of the research projects are listed below and the Bureau would be pleased to supply detailed information on any of their work. Their current basic interest is in copper with secondary interest in the recovery of uranium from copper barren liquors and the recovery of sulphur from dilute smelter gases. There is also work on Cu-Pb-Zn concentrates and on sulphate removal from seawater prior to distillation - to avoid boiler scale.

Current Projects:

1. Copper concentration in dump leach rock by selective sizing. From a dump sample of average grade 0.2% Cu, a minus ¼-inch fraction was screened which contained 0.4% Cu. Mo, Au and Ag were also concentrated in the fine fraction.

2. As an extension to the above, partial size reduction of dump material followed by screening at ¼ inch is also being investigated.

3. Oxidising agents for dump leaching of chalcopyrite. Dump leaching of chalcopyrite is very difficult. Oxidising with HNO_3, Cl, O_2 and SO_2 have been tried in the laboratory, Cl giving the only real positive result but with a consumption of 15 to 20 lb of chlorine per pound of copper in a closed system. This is uneconomic and no further investigation is planned at this stage.

4. Fundamental studies of chalcopyrite leaching - long-term.

5. Cement copper refining:
   Cement copper from launder oepration contains about 77% Cu, 6% Fe and 3% insoluble. By screening at 10 mesh, followed by flotation, a 93% Cu product can be obtained. The estimated production cost for this up-grading is 5.5 cents (US) per pound Cu plus 20% for return on capital investment for a 10 tpd plant.

6. Electric smelting and gaseous refining of cement copper

precipitate:
US Bureau of Mines, 1961, RI5899.

7. Smelting upgraded flotation copper in a reducing atmosphere.

8. Electro-refining of cement copper dissolved in H_2SO_4 (with aeration).

9. Copper cementation using auto scrap in a rotating drum (Reported in RI7182).

10. Copper recovery from converter slag:
Work is proceeding on both slow and rapidly quenched slags. The copper is recovered by flotation, the slow quenched slag giving the better recovery.

11. Pressure leaching of copper concentrates.

12. Acid leaching of Cu-Zn-Pb concentrates:
Cu and Zn is dissolved with H_2SO_4 and recovered by solvent extraction with Ca-EHPA.

13. Sulphur recovery from smelter gases:
The gases pass through a solution of 1 M citric acid and 1.0 to 1.5 M NaOH (solution pH approx. 3.5) in a packed absorption column. The solution is reacted with H_2S in a separate vessel to precipitate elemental sulphur. The sulphur is recovered by filtration as a fine crystalline powder. This process looks interesting and is the subject of patent.

14. Sulphur from gypsum.

15. Uranium from copper barren liquors:
Some barren liquors from cementation or solvent extraction contain approximately 10 ppm U. This is an attractive source as the solution is available at no cost. The type of equipment is of particular interest.
This is an eluex process. Current experiments are with liquor from cementation and the iron is present in the reduced (ferrous) form and not loaded by the resin. The resin is relatively coarse, 16 to 20 mesh, the type used in resin-in-pulp applications (Dow 21K or Rohm and Haas 425 - strong base quarternary amines).

The loading column consists of 8 by 2 ft sections of 2-inch ID glass, each 2-ft section being separated by an orifice plate. Currently installed are ¾-inch orifice plates but indications are that the orifice area should be only about 5% of the column area. Pregnant liquor is up-flowed at approximately 25 gallons (US) per square foot per minute. At this condition the resin is teetering (and not mixing) in each 2-ft section and not up-flowing, and each section is approaching a theoretical stage.

At regular intervals, the pregnant liquor flow is stopped and a volume of loaded resin is drained from the lower section. During this operation approximately equal volumes of resin transfer through the orifice plates to the successive lower column sections and a charge of stripped resin is added to the top section. The resin drain time is relatively small compared with the absorption cycle and the system approaches continuous ion exchange with a high liquor flow capacity.

The loading and stripping columns are not matched capacity-wise and currently a 4-inch diameter column approximately 12 ft long is being used for stripping experiments. Loaded resin is fed continuously to the top of the column and stripped resin discharges via a control jackleg from the bottom. Eluant, which can be either SO_4^{--}, Cl^{--} or NO_3^{-} - SO_4^{--} for subsequent amine SX, is up-flowed at a maximum rate of five parts aqueous to 1 part resin. This is the maximum rate in this system - higher rates cause resin backmixing and loss in counter-current efficiency. The eluate goes to an amine solvent extraction circuit where stripping is either with NH_3 - $(NH_4)_2SO_4$ at pH 4.5 or 1.5 M NaCl plus 10% $NaHCO_3$.

16. Sulphate removal from seawater prior to distillation:

No details of this process were given but apparently barium chloride is used to precipitate sulphates from solution prior to distillation.

<u>General Information</u> (from Mr D'Arcy George).

1. LIX64 considered a good reagent for Cu extraction but the loading capacity is a limitation.

Economics will depend on the degree of solvent loss in raffinate. Ranchers were initially using a flotation stage on raffinate to recover organic.

2. Algae in amine circuits:
   One South African uranium plant had algae troubles until they changed from kerosene to Solveso, an Esso oil product.

3. Zinc stripping from EHPA:
   No problems at Bureau providing sufficient H_2SO_4 is present:
   a. Use 1 M H_2SO_4 in three stages - 1 M $ZnSO_4$ and no free acid.
   b. Also strip with recycle electrolyte 60 to 80 grams per litre Zn - 125 to 150 grams per litre Zn to electrolysis.
   c. Zn loading with 0.6 N (0.3 M) EHPA - load to 18 grams per litre Zn.

3.10 British Columbia Research, Vancouver

British Columbia Research is a contract research organisation with 25% of the total budget being guaranteed by the provincial government. All work on bacterial leaching has been self-supporting with contracts from industry.

The biology division works mainly in the following fields:

1. Mineral microbiology
2. Water pollution - primarily with pulp mill effluents
3. Fish biology
4. Marine borer research - breeding patterns of borers are monitored along the north and west coasts. They also have diving teams equipped with sonic devices to detect borer holes in underwater timbers (piling etc.)
5. Industrial fermentation - the growth of bacteria in kerosene, natural gas and other petroleum products is studied.

About 75% of the total budget is associated with long-term projects (several years). Of the total number of projects, approximately 60% are short-term, ranging from a few days to several months.

Bacterial Leaching. The basic work has been on chalcopyrite but sulphide attack is of general application. The main exceptions to mineral solubilisation are lead, which goes to insoluble lead sulphate by either chemical or bacterial attack, and molybdenum which produces insoluble ferrites.

Preliminary tests, with the material ground to essentially minus 400 mesh, are carried out in shake flasks with controlled temperature and bacteria addition. Some tests are operated with automatic pH control. These tests will give a positive result in a matter of days compared with months or possibly years in simulated dump leaching tests using larger material in percolated columns. However, longer-term confirmatory tests are made in columns.

Larger scale tests, with finely ground ore, are then conducted in 20 inch by 20 inch by 16-inch deep tanks where the slurry is agitated and aerated with a special type of agitator. The agitator, Simon Carves (Simcar) aerator, is designed to throw a stream of slurry above the surface to accelerate air dissolution and entrainment. The slurry concentration is usually in the range 20 to 25% solids. As aeration depends on the water content, operation with higher solids concentrations and consequent lower water content, is impracticable.

The agitation process is being investigated on a larger scale to obtain scale-up information and operating costs. The full-scale pilot plant tanks will be 20 ft by 20 ft by 16-ft operating depth, constructed of concrete. Ten sponsors, each contributing $10,000 are required for this project. There are currently six definite sponsors, including Bagdad, Duval, ASARCO - American Smelting and Refining and AMAX - American Metal Climax.

Dr Duncan considers there is a bright future in bacterial leaching of sulphide concentrates followed by solvent extraction and electro-winning.

General Comments:

1. Leaching of very low grade ores (which will not percolate) by agitation leaching cannot be considered.

2. Bacterial leaching of copper ores is not practised in Canada except for one very small operation at Britannia Beach Mine where the bacteria are naturally occurring and no control is exercised.

3. One mine with fine ore difficult to percolate used perforated polythene pipe in drill holes for liquor distribution but these pipes periodically blocked and the operation was not very satisfactory.

4. Leaching of relatively high acid consuming ore can be practicable in some cases where sufficient total sulphide and iron is present for bacterial oxidation to produce ferric sulphate. A high acid

addition may be required initially on a dump of new ore, but once the bacteria are operating, excess acid is produced and some of this can be bled as starting liquor for another dump.

3.11 The University of British Columbia, Vancouver

The University is located on spacious grounds, a site of several thousand acres of natural timber country, and the erection of new buildings is proceeding on a large scale. Metallurgy is housed in a new five storey building and is very well equipped. There is a separate Chemical Engineering Faculty. Post graduate study is encouraged and six appointments are made each year in the Department of Metallurgy, with a grant of $3000 per year for Ph.D. studies.

There has been considerable work on pressure leaching of copper concentrates with elemental sulphur recovery an essential part of the process. No immediate application is seen. Hematite and goethite pressure leaching studies are also being made. The AEC is sponsoring fundamental work on the study of the reaction of steam on graphite.

The production of cathode lead from lead sulphides by chemical reaction followed by electrolysis has also been studied:

$$PbS + PbO_2 + 4NH_2SO_3H - 2Pb(NH_2SO_3)_2 + 2H_2O + S$$

$$2Pb(NH_2SO_3)_2 + 2H_2O - \underset{\text{cathode}}{Pb} + \underset{\text{anode}}{PbO_2} + 4NH_2SO_3H$$

Comment on Kennecott Dump Leaching. During the 8 months strike in 1968, Kennecott recycled liquors through the dumps without copper removal by cementation. During this period the copper concentration in the liquor did not increase (some iron precipitated in the dumps as the liquor pH rose). They probably have naturally occurring ion exchange materials in the rock which come to an equilibrium loading with the solution. Copper can exchange to the liquor only when the concentration in the liquor is lowered, achieved in this case when copper is removed by cementation.

3.12 Bethlehem Copper Corporation Ltd, British Columbia

Ashcroft is 220 miles north-west of Vancouver with Bethlehem Copper about 30 miles south of Ashcroft. The 250 mile trip is through mountain country which is snow covered in the winter season.

Open Pit Mine. The total mining rate is 45,000 tpd with 14,000 to 15,000 tpd of average grade 0.55% Cu going to the mill. The ore is predominantly bornite with some chalcopyrite. Pyrite concentration is low. Acid consuming material is relatively high and no dump leaching of low grade dump rock is practicable - acid consumption 100 to 150 lb H_2SO_4 per ton. Two boring rigs are on site for blasting operations. The loading and transport equipment consists of fifteen 50-ton trucks (costing $110,000 each - Canadian), one 15-20 ton front-end loader, three 5½ cubic yard shovels and one small front-end loader.

Another company, Valley Copper, has discovered copper in the same area and this deposit extends into the Bethlehem Copper lease. Apart from their existing open pit, Bethlehem also have two other areas being prepared for open pit mining.

Milling and Flotation. The initial milling rate was 3500 tpd in 1961 but this has been expanded to 15,000 tpd. The mill is compact, equipment is relatively new and housekeeping is excellent. Well equipped laboratories and maintenance facilities, including a rubber lining bay, are on site.

Primary crushing is with a 42 by 65 A-C gyratory followed by two secondary stages using 7-ft Symons cone crushers.

The primary grinding is in two 12 ft 6 in. diameter by 15-ft rod mills, followed by secondary grinding in two 10 ft 6 in. diameter by 14-ft ball mills and tertiary grinding in three ball mills (two by 11 ft dia. by 14 ft and one by 12 ft 6 in. dia. by 15 ft).

The slurry to the rougher circuit has a solids concentration in the range 35 to 40% solids. Denver, Agitair and Britannia aircells are used in the overall flotation circuit. With the low pyrite concentration in the ore there is no deliberate depression of pyrite. Lime consumption is 1 lb per ton ore and frother is at the rate of 0.022 lb per ton ore. The concentrate grade ranges from 25 to 48% Cu with the normal average in the range 35 to 37% Cu. The higher concentrate grades are obtained when the ore feed is essentially bornite and the lower grades when increased amounts of chalcopyrite are present. Filtered concentrate is transported by road to Vancouver at the rate of 200 tpd. Copper production in 1968 was 50 million pounds of Cu with an overall mill recovery of 84%.

Three tailings thickeners were used for water recovery but the operating cost, particularly flocculant cost, was excessive and tailings now go direct to a dam where the water is recovered with a pump barge at the rate of 6500 US gallons per minute with a 500 hp vertical pump. Although additional head, of the order of 200 to 300 ft, is involved pumping from the tailings dam, water recovery costs have decreased considerably since bypassing the thickeners. This was the only plant visited not using tailings thickeners and the delection of thickeners should be considered if a tailings dam can be conveniently sited at a low head differential with the mill. Make-up water is pumped from bores at the rate of 1500 US gallons per minute.

Production Costs. These are given in detail in the Canadian Mining Manual, 1968. Mining costs were 39 cents (Canadian) per ton of ore. Crushing, grinding and conveying cost 36 cents per ton. Milling, drying, tailings disposal and laboratory and maintenance facilities cost 33 cents per ton.

The total of the two latter items, 69 cents, per ton, has now increased to 78 cents per ton.

In general, costs have been kept to a minimum by the installation of larger type equipment, particularly in the mining field. The equipment sizes are comparable with those used in much larger tonnage operations.

3.13 Craigmont Mines Limited, British Columbia

No visit was made as the road from Bethlehem Copper to Craigmont at Merritt was not passable for conventional vehicles and the alternative was a long detour. Time was not available as the round trip that day was already 500 miles. Bethlehem Copper personnel commented that the grade to milling was not up to expectations as there was excessive ore dilution with the cave mining techniques. However, the problem was being overcome.

Placer Development were contacted in Vancouver on 18 July, 1969. Mr Harold Steane, General Mill Superintendent was out of town but Mr Eastman, Manager, Engineering and Construction gave detailed information.

Mining and Milling. The ore is mainly chalcopyrite with some magnetite and hematite. About 55% of the mill feed is from a low grade (0.57% Cu) stockpile obtained from previous open pit operation. The remaining 45% is from current underground mining operations. The milling rate is approximately 5000 tpd of average grade 0.97% Cu. Ore reserves are estinated at 3.7 million tons of 0.57% Cu ore in stockpile and 15.2 million tons of 2.03% Cu are underground. The orebody extends to 1500 - 2000 ft below the surface.

Size reduction is in three stages, the tertiary stage being in closed circuit. Milling is in an open circuit rod mill followed by a closed circuit (with cyclones) ball mill. The rougher flotation circuit is followed by three stages of cleaning with an intermediate regrinding stage. The concentrate, averaging 28% Cu, is filtered on a vacuum disc filter then railed to Vancouver for shipment to Japan.

The mining operation is heavily mechanised with diesel powered equipment.

3.14 Anaconda Company - Britannia Beach Mine, British Columbia

The mine was contacted but no visit could be arranged in the limited time available. The following information was obtained by phone contact. Further information can be obtained on request.

The main production is from underground mining followed by conventional milling and flotation. A small in situ leaching of an old underground working is of particular interest. The old working consists of chalcopyrite plus pyrite of estimated grade 1% Cu. The mine is in moutain country and although the working depth was 3000 to 4000 feet below the surface, a gravity flow of water can be obtained from the surface through the old workings then discharging through channels at the base of the mountain peak. The water flow varies with natural rain and snow falls but the average flow through the two areas leached is:

1. 1000 gallons per minute (varying from 600 to 5000 gallons per minute.
2. 1500 gallons per minute.

The first area produces a liquor containing an average of 0.1 grams litre Cu and the second 0.03 grams per litre Cu. The liquor pH is approximately 3.0 and this passes over scrap iron in a launder with the exhaust liquor containing 0.018 grams per litre Cu. Iron consumption is in the range 2.0 to 2.5 lb Fe per pound Cu and a low grade, about 50% Cu, cement is produced. The operation is small but profitable. Labour is one man per day on launder operation plus approximately 1 hour per day of technical supervision. Naturally occurring bacteria are considered to be an essential part of the process. No acid addition is made. Refinements to the operation are being investigated.

4. DISCUSSION OF UNIT OPERATIONS

Mining, milling and flotation operations are well established and applications in Australia are similar to operations visited, although generally on a smaller scale. The particular operations inspected which are not so common or are non-existent here were dump leaching and solvent extraction followed by electro-winning. These operations are summarised.

4.1 Dump Leaching

Ore grades for dump leaching are generally of the order of 0.2 to 0.5% Cu but range from 0.1 to 1.0%. The ore is usually as-mined with rocks up to 3 ft in diameter and a minimum amount of fines. Fine clay type ores are not suitable for percolation. Some dumps are old material previously considered too low grade for other methods of extraction, but most dumps are specially formed from currently mined ore and may consist of low-grade sulphide ores which cannot be economically processed by flotation, or oxidised ores of various grades. The dumped ore is usually several hundred feet in depth.

The dumps are preferably sited in canyons or hollows with hard rock bases. The bases are usually unsealed but occasionally a cement mix is used to compact porous areas. Where a new heap is being constructed, it is preferable to lay perforated PVC pipes in trenches the full length of the heap and cover the pipes with coarse gravel.

Five methods of liquor distribution on the surface of the heaps were used:

a. Irrigation type sprays
b. Perforated plastic pipes
c. Plastic pipes with needle valves at regular intervals
d. Flapping hoses
e. Intermittent flooding.

People using sprays claimed that this method is the best for dissolution and entrainment of oxygen, necessary for the oxidation of ferrous iron to ferric iron, while others using intermittent flooding claimed that sufficient air is aspirated into the heaps between surface floodings. The flooding technique was certainly the cheapest and simplest method. The heap surfaces are usually periodically ploughed to maintain porosity. Ploughing is also practised before adding additional ore to an established working.

The acidity of the recycled liquor is adjusted to give a product liquor pH of approximately 2.0, to avoid iron precipitation in the heaps and in liquor lines. All heaps require sulphuric acid addition initially but in some cases where sufficient sulphides are present, acid is produced by bacterial action. Bacteria is cultivated in one area while in other areas the assistance of naturally occurring bacteria was recognised. In the latter cases, all new dumps were started with some liquor from old dumps. The copper concentration of the pregnant liquor from the dumps varied considerably and was usually of the order of 1 to 2 grams per litre Cu, but in one instance the concentration was 4 grams per litre Cu. Copper was recovered from the pregnant liquor either by cementation or by solvent extraction and electro-winning.

4.2 Solvent Extraction and Electro-winning

By this process, high grade cathode copper is produced directly from leach liquors. In all cases, the solvent used was General Mills LIX64 or LIX64N.

The leach liquor is contacted with a 5 to 7% LIX solution in kerosene in either three or four extraction stages. Barren liquor from the extraction circuit is recycled to dump leaching. The solvent extraction process has a big advantage in that no iron is added to the recycled leach liquor, as is the case with the cementation process.

In addition, the acid consumed in copper dissolution in the leaching operation is regenerated in the extraction stage and is recycled to leaching in the barren liquor.

The loaded solvent containing approximately 1 gram per litre Cu is then stripped with 150 grams per litre sulphuric acid solution in either two or three stages, the stripped solvent being recycled to the extraction circuit. The acid strip solution containing 30 to 40 grams per litre Cu goes to an electro-winning circuit.

To maintain a high current efficiency and to give a high purity cathode copper, only approximately 5 grams per litre Cu is plated from solution in one pass. The copper depleted electrolyte is recycled to the solvent stripping section where the copper concentration is increased by stripping copper from the loaded solvent.

5. APPLICATION TO KAPUNDA ORE

Following an article published in World Mining, October 1968, entitled "Bacterial leaching recovers copper in highland valley" (British Columbia), the South Australian Department of Mines was interested in the possible application to Kapunda ore. Correspondence failed to locate this operation, Copper-Can Development Limited, and by personal contact with British Columbia Research in July 1969, this operation was found to be non-existent. However, British Columbia Research are proceeding with experimental work on bacterial leaching of sulphide ores in agitation systems. To date no commercial application had been established.

Dump leaching with bacterial assistance is an established operation both in the United States and British Columbia but the type of ore must be amenable to percolation. From studies of Kapunda ore drill samples and of exposed faces of the old open cut workings, percolation leaching would not be possible, due to the high clay content of the orebody. Unless there is a breakthrough in extraction methods for ores which are predominantly sulphide, by agitation leaching with or without bacterial assistance or by other means, conventional methods such as flotation should only be considered at this stage. Flotation tests on this ore have given encouraging results.

6. ACKNOWLEDGEMENTS

The financial support of the South Australian Department of Mines for these visits is gratefully acknowledged.

General Mills Incorporated arranged all the plant visits in Arizona and special thanks is due to Messrs J.E. House, E.R. DeMent and C.R. Merigold.

The co-operation of all people, listed in Section 2 of this report, in conducting plant inspections and discussing all operations is sincerely appreciated.

APPENDIX A

NEW TECHNOLOGY OF LEACHING WASTE DUMPS

by

E.E. Malouf and J.D. Prater

(Mining Congress Journal, November 1962)

New Technology of Leaching Waste Dumps

Higher extraction of copper from mine dumps is achieved with warm leach solutions, indicating the desirability of recycling effluents heated in percolating through large waste dumps

By E. E. MALOUF
Project Development Engineer
and
J. D. PRATER
Chief, Hydrometallurgical Section
Kennecott Research Center
Kennecott Copper Corp.

HERETOFORE, leaching of copper from mine waste has been more of an art than a science. Long ago, the blue tinge of waters flowing from some copper mines and the freshets coming down the draws during spring runoffs where mine waste was dumped showed that water-soluble copper—$CuSO_4$—was going to waste. Moreover, where these copper-bearing waters flowed over tin cans or bits of scrap iron, the red color of precipitated metallic sponge copper was evident. What could be more logical insofar as mine waste dumps were concerned—simply supplement the natural spring runoff by pumping water into ponds on top of the mine waste dumps, collect the water flowing down the gulch from the bottom of the gulch, let it pass over metallic iron (detinned scrap) in a concrete launder, collect the copper precipitates and then pump the water back up to the ponds to be recycled. The practical result was a simple process by which some of the low-cost copper was produced with a small capital investment. Indeed, what could be simpler?

Object is to Replace a Random System

Some years ago, S. R. Zimmerley, research director at Kennecott Research Center, was concerned about the huge amounts of copper gradually accumulating in these very low-grade mine waste dumps as compared to the relatively small amount that was being retrieved annually by this simple, lucrative and empirical process—"sort of letting nature take its course." Also, he was concerned about a critical problem as to how insoluble copper sulfide minerals such as chalcopyrite and chalcocite were changed into $CuSO_4$, which was soluble in water and hence could be leached from the mine dumps. In this instance, his hunch that bacteria, remotely hinted to in previous scientific literature, are the catalyst for the necessary oxidation to render the copper soluble in water proved to be correct.

Further considerations of the overall project to permit long-range planning and to yield the maximum economic return, led to systematic studies in basic research, bench testing, mathematical models, pilot plant leaching and field experiments on experimental dumps at the mines. These investigations embraced the study of bacteria-accelerated reactions, chemical reactions, chemical kinetics including the temperature effect, the theories of permeability and tortuosity related to the flow of water through random and altered rock, the mineralogy of the copper-bearing waste material, and the effect of constituents such as limestone on the solubilization of copper. These efforts have been supplemented by studies whereby the leaching systems may be automated and the precipitated copper may be made at a lower cost and yield a higher grade product.

To date, we certainly do not have all the answers. We are in the midst of trying to convert an apparently simple, random process into one that can be controlled, engineered and designed to produce low-cost copper to create a maximum cash flow to the corporation. Our research indicates that our findings may not be confined to copper. Also, bacterial oxidation of pyrite might have some bearing on the prime problem of low sulfur in coal.

To return to our basic thesis: we are concerned with the development of a new technology for the recovery of copper from the low-grade, multimillion ton reserves resulting from the segregation of sub-marginal copper mine waste in dumps amenable to leaching processes.

Controlled pH Aids Process

By controlling the pH of the recycled solutions it was found that accretions in the pipelines were dissolved, the surface of the dump kept pervious, and the ready percolation of solutions into the dump permitted the inspiration of air, thereby accelerating the bacterial oxidation of ferrous to ferric iron.

Controlling the pH of the leach solutions not only keeps the pipelines and surface of the waste dump open, but also keeps the ferric iron in solution, thereby forming an effective lixiviant for the copper sulfide minerals.

Oxidizing Bacteria Found in Mine Waters

A study of various mine waters indicated the presence of oxidizing bacteria that actually thrive in high-acid copper solutions. These bacteria not only thrive in this environment, but accelerate the oxidation of sulfide minerals to form acid and soluble copper and iron sulfates. One strain of bacteria, thiobacillus thio-oxidans, utilizes the oxidation of the sulfur of the sulfides as its energy source, while another strain, thiobacillus ferro-oxidans, oxidizes ferrous iron to ferric iron as its energy source. These bacterial strains complement each other in that the one produces acid which keeps the ferric iron in solution that is produced by the other, thereby forming the powerful lixiviant for copper minerals of acidified ferric sulfate.

This bacterial oxidation of ferrous to ferric iron occurs a thousand-fold faster than does the chemical oxidation reaction by aeration at ambient temperatures under similar conditions without bacteria.

Bacterial oxidation of pyrite to form sulfuric acid and ferric sulfate can be used in "breeder' tanks on a waste dump to generate a low-cost lixiviant for the copper in the waste. This approach can be used where waste dumps are low in sulfide minerals or where a high limestone content of the waste prevents the development of the bacterial oxidation in the dump.

In waste dumps where the environment is suitable for the promulgation of the bacteria, it has been determined that under conditions of controlled pH of the solutions and with ferrous iron present, ferric iron is readily formed in the first 20 to 50 ft of waste through which the solutions trickle. Sufficient oxygen for the bacteria is obtained from the solutions trickling over the rocks in the dump. The cycle of the ferrous iron being oxidized to ferric iron, which in turn oxidizes the sulfide minerals, with the iron again being reduced to ferrous iron to again enter the cycle, can be repeated several times as the solutions permeate through the waste. This effect will continue as long as the bacteria obtain oxygen and the temperature in the dump does not exceed the pasteurization temperature.

Sunlight Makes Bacteria Inactive

Bacterial cultures become inactive. but are not destroyed, when exposed to direct sunlight. Experiments have been made to evaluate bacterial activity in open ponds with solution depths up to two ft. Virtually no bacterial oxidation was noted in these ponds, even though the solutions were in nighttime darkness during 50 percent of the time and viable bacterial cultures were present. The bacteria are extremely sensitive to ultraviolet light, for only short exposure to this radiation will completely sterilize a culture.

Table 1. Rate of dissolution of sulfide minerals and sulfide-bearing materials using bacterially generated lixiviant

Material Leached	Lixiviant	Leaching Time, Days	Percent of Total Extracted			
			Cu	Fe	Zn	Mo
Chalcopyrite sample	Recycled Solution	65	9.2			
Chalcopyrite-flotation concentrate	Recycled Solution	408	47.9			
Chalcopyrite-bearing mine waste	Recycled Solution	287	64.6			
Chalcocite-bearing ore	Recycled Solution	105	95.0			
Chalcocite-bearing flotation tailing	Recycled Solution	89	86.0			
Copper-bearing pyrite	Recycled Solution	42	95.0			
Copper-bearing pyrite*	Synthetic Solution	113	79.8	3.8		
Sphalerite	Synthetic Solution	337			19.6	
Sphalerite and pyrite	Synthetic Solution	337		100	48.6	
Molybdenite concentrate (copper bearing)	Synthetic Solution	123	28.2	9.8		< 0.1
Molybdenite concentrate (copper bearing)	Recycled Solution	78	6 6			

* Synthetic solution refers to a ferrous sulfate solution inoculated with oxidizing bacteria while the recycled solution refers to the acidified ferric sulfate solution resulting from the bacterial oxidation.

Temperature has been found to exert a very pronounced effect on the rate of the bacterial conversion of ferrous to ferric iron with maximum bacterial activity occurring at 35°C; temperatures above 50°C have been found to destroy the bacteria; below 35°C the rate of bacterial action decreases non-linearly as the temperature is reduced.

The oxidizing bacteria are active only in acid media. Bacterial action is most pronounced in a media having a pH between 2.0 and 3.5. Both above and below this pH range, the rate of bacterial oxidation decreases and at pH values above 6.0 bacterial action is almost completely inhibited. In alkaline media (pH 9) the bacteria are destroyed.

Warmed Solutions Increase Recovery

With the use of controlled pH leach solutions on the large mine waste dumps, it was noted that copper content of the effluent solutions increased two to four lb of copper per 1000 gal over and above that obtained previously. This was attributed not only to the formation of the ferric sulfate lixiviant, but also to maintaining a pervious condition on the waste dump which permitted the penetration of air necessary for the bacterial oxidation to proceed.

As these conditions were maintained on the waste dump, the temperature of the effluent solutions increased 25°F over the temperature of the solutions entering the waste dump. Solutions that were entering the waste at 85°F temperature were draining from the waste with a 110°F temperature.

On an experimental basis, the leach solutions were first heated to the 100 to 110°F range before being discharged on the waste dump to determine the effect of recycling these warm solutions through the dump. The effect of these warm solutions was to double the copper content of the solutions from the dump.

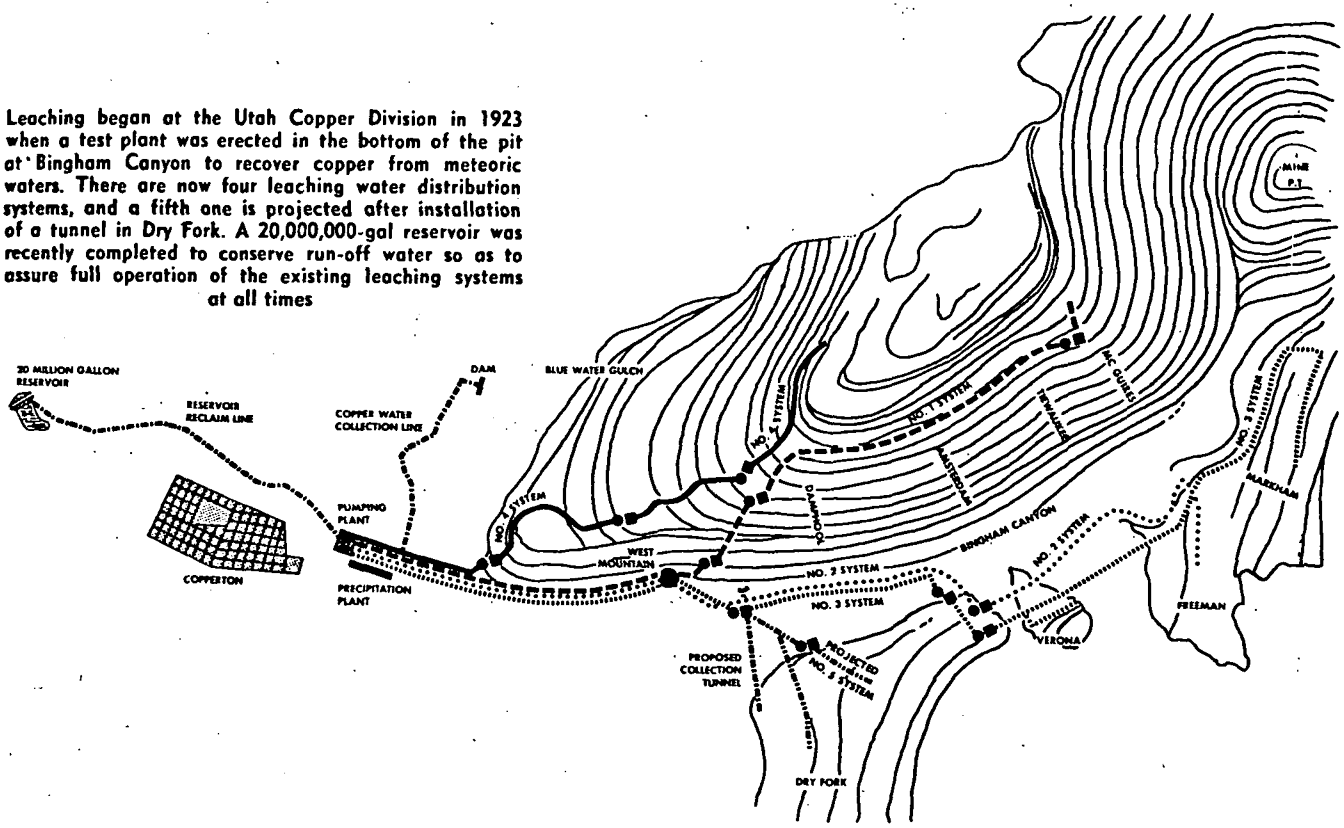
Even though the temperature of the leach solutions had approached the point where pasteurization of the bacteria could be expected, the rate of chemical dissolution at this higher temperature exceeded that of the bacterial action.

On small dumps, namely those up to 250 ft in depth, the energy generated by the oxidation processes is lost by radiation of heat as fast as it is generated. However, on larger dumps there is an accumulation of this energy sufficient to increase the temperature of the leaching solutions an additional 25°F.

Value of pH Control Demonstrated

Rate of dissolution of a number of sulfide and sulfide-bearing minerals has been determined in laboratory tests. Data obtained from these tests indicate that the copper sulfide minerals, in increasing resistance to dissolution, are chalcocite, covellite and chalcopyrite.

With copper-bearing pyrite, the



Leaching began at the Utah Copper Division in 1923 when a test plant was erected in the bottom of the pit at Bingham Canyon to recover copper from meteoric waters. There are now four leaching water distribution systems, and a fifth one is projected after installation of a tunnel in Dry Fork. A 20,000,000-gal reservoir was recently completed to conserve run-off water so as to assure full operation of the existing leaching systems at all times

copper was selectively extracted. In 113 days of percolation leaching of copper-bearing pyrite, containing four percent copper, using a synthetic nutrient solution inoculated with bacteria, 80 percent of the copper was extracted while only four percent of the iron was solubilized. However, when a pH controlled, bacterial generated lixiviant of ferric sulfate was used, virtually all the copper was extracted in 42 days. The second solution was much more effective because of its higher content of solubilized ferric sulfate.

Copper and iron minerals can be selectively leached from molybdenite concentrates. Twenty-eight percent of the copper, ten percent of the iron and virtually no molybdenum were extracted in 123 days of percolation leaching of a sample of molybdenite concentrate containing 2.17 percent copper and 3.30 percent iron.

Sphalerite was altered and the zinc solubilized when the sample was leached using a synthetic solution inoculated with the oxidizing bacteria. With mixtures of sphalerite and pyrite, the pyrite was preferentially attacked. In one laboratory test in which a mixture of ten grams of sphalerite and three grams of pyrite was used, the iron was completely solubilized together with 50 percent of the zinc in 337 days.

Table 1 lists various products leached and the comparative extraction at various leaching times.

In summary, the order of copper minerals in increasing resistance to attack was chalcocite, covellite, chalcopyrite. Copper was selectively leached from copper-bearing pyrite; copper and iron were preferentially extracted from molybdenite concentrates. Pyrite was attacked more rapidly than sphalerite when mixtures of the two minerals were leached simultaneously with lixiviant prepared by the use of oxidizing bacteria.

Dump Permeability Varies Widely

The capacity of a formation for transmitting water is measured by its coefficient of premeability which is defined by Meinzer as the rate of flow of water in gallons per day through a cross-sectional area of one sq ft under a hydraulic gradient of one ft per ft at a temperature of 60°F.

Flow of leach solutions through a mine waste dump is affected by the composition of waste, namely, fines or clay minerals that are present, size of the waste, depth and horizontal length of the dump, age of the dump which determines the degree of reconsolidation of the waste, as well as the breaking down of various gangue minerals to form clays or slimes which fill the interstices of the dump. The larger the dump, the faster the alteration due to internal temperatures and pressures.

Data obtained from leaching various size dumps indicate the coefficient of permeability of leach solutions of 30 gal per day per sq ft of area can be expected for dumps 30 or more years old which contain considerable slimes and clay, to 240 gal per day per sq ft of area for a waste dump which has not aged to the point where considerable reconsolidation has taken place, namely one under 30 years of age.

Leach solutions placed on a 100-ft high waste dump require two to three days before the solutions emerge from the toe of the dump. A 250-ft high dump requires three to four days, a 300-ft dump six days, while a 400 to 500-ft high dump requires 12 days. Of course, this time of percolation is dependent not only on the vertical distance, but also on the horizontal distances through which the solutions must percolate.

Ground water hydraulics can be applied in studying the leaching of mine waste. Although the first studies of the flow of water through capillary tubes by Hagen and Poiseuille indicated that the rate of flow is proportional to the hydraulic gradient, it was Darcy who confirmed and applied this law to the flow of water percolating through filter sands. Darcy's law is applied to the percolation of leach solutions through mine waste and is expressed as follows:

$$v = \frac{PI}{7.48p}$$

where: v = velocity in feet per day
P = coefficient of permeability in gallons per day per square foot
I = hydraulic gradient in feet per foot
p = porosity in percent

In most ground water problems, and this would apply to mine waste dump leaching, the total volume of flow is required rather than the velocity, and consequently Darcy's equation is modified to the following form:

$$Q_d = PIA$$

where: Q_d = discharge in gallons per day
P = coefficient of permeability in gallons per day per square foot
I = hydraulic gradient in feet per foot
A = area of flow cross section in square feet

This formula may be adapted for use with the more convenient coefficient of transmissibility by noting the distinction between its definition and that of the coefficient of permeability:

$$Q_d = TIW$$

where: Q_d = discharge in gallons per day
I = hydraulic gradient in feet per foot
T = coefficient of transmissibility in gallons per day per foot
W = width of flow cross section in feet

In applying Darcy's equation of ground water hydraulics to several cases of waste dump leaching where the waste dumps have had different permeabilities and heights, the results have been the same so that invariably the hydraulic gradient has been approximately one foot per foot of height.

Acid Added to Depleted Solutions

The bacterial oxidation of ferrous iron to the ferric state is beneficial because of the accelerated rate of solubilization of copper. However, the resultant oxidation of the iron with solutions of a pH value resulting from precipitation of copper with scrap iron caused undesirable hydrolysis and precipitation of basic iron sulfate. Lowered pump capacity due to partial plugging of the lines and sealing of the dump areas by precipitated salts resulted.

The addition of small amounts of sulfuric acid to the copper-depleted solution has been found to be effective in preventing hydrolysis and precipitation of basic iron salts and is now being used as a regular part of the leaching program in the Kennecott operation. Before pH control was practiced, permeability of the dump with lixiviant was very slow.

Field tests in an experimental dump at the Utah mine indicated that ponding of solutions could be practically eliminated by this addition of sulfuric acid. Later tests on one of the operating dumps indicated permeability was increased from two gal per sq ft per hour to 15 gal per sq ft per hour. Another advantage in the use of the acid was the increased time that an area could be maintained under leach without sealing of the dump. Over-all recovery of copper from the dump was substantially increased because of the longer time that an area could be leached.

The concentration of copper per unit volume of solution was increased about 30 percent in field tests when acid was added to solutions used as lixiviants as compared with solutions to which no acid was added after the copper was precipitated on scrap iron. Substantial savings in the cost of pumping solutions to the dumps were realized because of the larger payload resulting with each cycle.

Reaction is Exothermic

During the course of the experiments on a major mine waste dump, an increase in the temperature of the effluent solution was noted. Oxidation of copper sulfide minerals and of pyrite are exothermic reactions. Because of the increased reactions occurring within the dumps, the temperature of the effluent solutions was increased 25°F compared with background data obtained before acid was added to the solutions. The reactions are synergistic in that with the increased temperature the rate of reaction was increased, thereby liberating additional heat. Laboratory tests under controlled conditions indicated that the rate of solubilization of copper from mine waste was increased two to three fold by raising the temperature of the lixiviant from 70° F to 100°F. The value of the conservation of all heat possible in the leach solution becomes obvious.

Channels Replace Dump Ponds

As a result of the information gained in the tests on a major waste dump, a different method of preparing the dump for leaching has been established. Formerly, ponds were made on the dump surface to impound the water. This was necessary because of the relatively long time required for permeation of the solution into the dump. The preparation of the ponds was time consuming and costly. With the increased rate of penetration because of the addition of acid to the solutions, ponds are no longer necessary on most dumps. Instead, shallow channels are prepared with a bulldozer to provide a strip for leaching. The strips may either be parallel with the edge of the dump or at right angles to it, depending upon the terrain and size of dump under leach. For small dumps, the channels at right angles to the edge are generally preferred, but with large dumps either system may be used. Each system is being employed on different dumps at the Utah mine. Generally a strip 20 ft wide by 400 ft long will accommodate a flow of 2,000,000 gal per day.

Principal objectives of the leaching tests were (a) to effect higher extraction of copper and at more rapid rates than previously obtained, and (b) to determine a systematic and controlled method for leaching the various dumps. It has been found that differences in leaching characteristics exist between the several dumps with respect to composition of the waste, percent of copper contained in the waste, depth of waste, and time of placement or again of the waste in the dump. Systematic data are required so that the status of each dump under leach may be known. With this information the frequency of change from one strip to the next and the optimum distribution of solution to the different dumps may be planned to achieve the most effective use of the water and to insure constant copper heading to the precipitation plant. Leaching of mine waste thus has become a planned and integral part of the over-all operation contributing substantially to the total copper produced. Reduced costs plus increased recovery of copper at a rate more nearly current with the placing of the copper in the waste dump are goals which may be achieved.

Alternate Methods Under Study

With the increased extraction of copper from the waste, the problem of recovering the copper from solution becomes increasingly important. The historic method for recovering copper from mine and waste leach solutions has been by precipitation on scrap iron. Two alternative methods have been investigated by Kennecott.

According to the first, copper is precipitated in a cone by contacting the solution with a form of powdered iron. The precipitant is maintained in a dispersed state by the upflow velocity of the solution. Advantages of the process include (a) the reaction is rapid and precipitation is complete, (b) control of the process is precise, (c) the operation lends itself to automation, (d) iron consumption per unit of copper precipitated is lower than that in the usual precipitation by flowing the solution over scrap iron. Precipitation of relatively high purity copper powder, for direct fabrication without melting, by electrolytic reduction from mine water is also being studied as a possible means of recovery. Each method or combination of both show considerable promise as a substitute for the historic method.

APPENDIX B

USE OF CONE-TYPE COPPER PRECIPITATORS TO RECOVER COPPER FROM COPPER-BEARING SOLUTION

by

H.R. Spedden, E.E. Malouf, and J.D. Prater

(Transactions of Society of Mining Engineers, December 1966)

USE OF CONE-TYPE COPPER PRECIPITATORS TO RECOVER COPPER FROM COPPER-BEARING SOLUTION

by H. R. Spedden, E. E. Malouf, and J. D. Prater

With the expansion of copper leaching of the various mine wastes at the Kennecott properties, additional copper recovery facilities have been required. A research and development program has been pursued to determine a suitable economic approach to the processing of the additional leach solutions contemplated. Investigations on laboratory and pilot plant scales covered electrowinning, solvent extraction, launder and vat type cells, activated launders, precipitation drums, and precipitation cones. This paper contains information on the development and use of a new precipitation-cone-type copper recovery plant, which provides many advantages over older systems.

The application of research findings to the old art of leaching copper from copper-bearing mine waste has resulted in a significant contribution of copper to over-all copper production. For example, at the completion of the current expansion program at the Kennecott properties in the United States, copper produced from waste dump leaching will amount to about 25% of the total production. Formerly only 2% was derived from this source. This planned expansion of copper leaching focused attention on the problem of developing more efficient methods for the recovery of copper from greatly increased volumes of copper-bearing solutions.[1]

The detailed investigations which have been made in laboratory pilot plant and plant tests for the recovery of copper from copper-bearing solutions have included electrowinning, solvent extraction, ion exchange and cementation with iron in launders, precipitation drums, activated launders, and precipitation cones. The results of these investigations have led to the development of a new precipitation cone-type copper recovery system providing many advantages over older methods.

ELECTROWINNING OF COPPER FROM COPPER-BEARING SOLUTIONS

The direct electrowinning of copper from relatively dilute solutions of copper-bearing mine water, namely solutions containing 10 to 20 lbs of copper per 1000 gal, has always been an attractive possibility.[2,3] By this method, high-purity copper powder can be recovered with a power consumption of 3.5 kwh per lb of copper. However, once the solution strength de-

---

H. R. SPEDDEN is Research Director, E. E. MALOUF is Project Development Engineer, and J. D. PRATER is Chief, Hydrometallurgical Section, Western Mining Div. Research Center, Kennecott Copper Corp., Salt Lake City, Utah. TP 66B86. Manuscript, February 1966. New York Meeting, February 1966. Discussion of this paper, submitted in duplicate prior to Mar. 5, 1967, will appear in SME Transactions, June 1967, and AIME Transactions, 1967, vol. 238.

creases to approximately 5 lbs of copper per 1000 gal, the current efficiency drops rapidly, requiring the use of other methods to recover the remainder of the copper economically. These results combined with the high capital cost make this approach economically unsound under most conditions.

SOLVENT EXTRACTION OF COPPER FROM COPPER-BEARING SOLUTIONS

Another technically feasible process is using various organic reagents for the solvent extraction and concentration of copper from dilute acidic solutions.[4] The majority of solutions obtained from leaching rather heterogeneous mine waste dumps, however, contain substantial quantities of ions other than copper as well as suspended insoluble materials. These various ions and suspended gangue can result in a combination that may cause emulsification and costly loss of the solvent. Although substantially improved liquid ion-exchange or solvent extraction reagents which resist emulsification to extremely low levels are now available, enough experience with these new reagents has not yet been obtained to safely justify a major installation. Presently available reagents have a rather low loading factor and thus would require a large capital expenditure for plants of the size now being constructed. A small plant and, in particular, one which does not have a readily available, low-cost source of scrap iron would appear to be the logical next step in this development. The application of the process to copper metallurgy on a significant scale is, nevertheless, a most stimulating goal.

ION-EXCHANGE RECOVERY OF COPPER FROM COPPER-BEARING SOLUTIONS

The use of ion-exchange resins for the concentration of copper from copper-bearing mine solutions has not proven feasible. The non-selectivity of the resins and the fouling of the resins with iron and aluminum ions has precluded the use of this approach. Even the carboxylic-type resins, which are quite specific for copper, display an unsatisfactorily low loading capacity when used for extraction from acidic solutions; thus projected capital costs appear to be unduly high.[5]

CHEMICAL PRECIPITATION OF COPPER FROM COPPER-BEARING SOLUTIONS

Precipitation of copper from dilute copper-bearing solutions using various chemical precipitants has been a source of continuing investigations by several research groups.[6,7] Consideration has been given to precipitating the copper as a sulfide, as a cyanide, as a thiocyanate, and even as a hydroxide using lime. The recovery of an extremely fine chemical precipitate, with the inherent difficulties of settling and filtering, is a common problem in essentially all processes employing chemical precipitants. The products of chemical precipitation also usually require additional processing steps to obtain the copper in a form readily obtained by cementation on iron.

PRECIPITATION OF COPPER BY IRON

Gravity Launders: The most common method of precipitating copper from leach solutions, and the oldest method, has been the gravity-flow launder charged with scrap iron as the precipitant. As a general figure, this type of plant requires 500 ft of launder, 4 ft wide by 4 ft deep, to process 1000 gpm of copper-bearing solution. A launder of this size can effectively recover over 90% of the copper in solutions. However, iron consumption will generally vary between two to four times that amount theoretically required to precipitate the contained copper, depending on the ferric iron and free sulfuric acid contents of the solution. Fig. 1 is a photograph of the gravity launder-type copper precipitation plant at Kennecott's Utah Copper Div.

Launder plants, although simple to construct and operate, require much hand labor and produce an impure cement copper which is usually blended with concentrates as a feed to a smelter. Efforts to improve the method have let to numerous modifications, mostly involving variations in the mechanical handling of the scrap iron or the precipitates.

Drum Precipitators: Rotating drum precipitators have been used industrially in place of launders. The major problem encountered in this system is that of maintaining a large mechanical device in which the total mass of scrap iron is tumbled constantly. Our own test work has shown that the tumbling action breaks the copper precipitates into fine particles, much of it even colloidal in size, thus presenting a further operational problem. Labor requirements for charging scrap iron and for the periodic removal of unconsumed trash likewise make these units generally unsatisfactory.

Activated Launders: Gravity launders have been modified by laying one or more nozzle manifolds along the bottom of the launder to inject the copper-bearing solutions into the mass of iron.[8,9] Studies involving the precipitation of copper in this type of launder have indicated definite improvements over the gravity-type plant, both as to iron factor and volumes of solution treated. However, here again, cleaning of this type of unit requires removing the accumulation of copper precipitates, cans, and trash with consequently high hand labor requirements.

Development of a New Precipitation System: Although most of the recovery methods thus investigated are practical, each one seemed to be lacking in at least one of several desirable features for relatively large scale production. As with most chemical processing methods, an efficient copper recovery system should

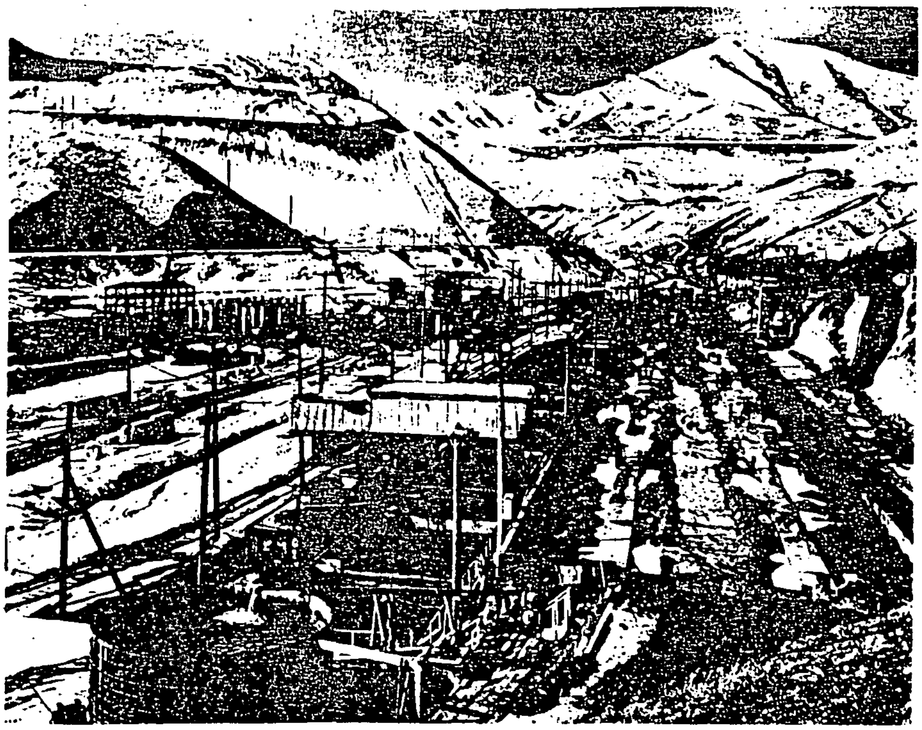


Fig. 1 – Photograph of gravity launder-type copper precipitation plant at Kennecott's Utah Copper Div., Bingham Canyon Mine.

provide for high volumetric capacity, be able to treat solutions of variable concentrations with high recovery, and be amenable to a substantial degree of mechanical handling and automatic control.

Three chemical reactions, each of which consumes iron, have long been recognized as of importance in copper cementation. These reactions may be shown by the following equations

$$CuSO_4 + Fe \longrightarrow Cu + FeSO_4 \qquad [1]$$

$$Fe_2(SO_4)_3 + Fe \longrightarrow 3\ FeSO_4 \qquad [2]$$

$$H_2SO_4 + Fe \longrightarrow FeSO_4 + H_2 \qquad [3]$$

Under quiescent conditions, as represented in a launder plant, these reactions will reach equilibrium. If, however, powdered iron of high surface area is used, the copper precipitation reaction is found to be predominant and may be essentially completed before excessive amounts of iron have been consumed by the other two reactions.

Since powdered iron is thus an effective precipitant and may be produced at reasonable cost as a by-product of a base-metal mining and smelting complex, a research program was initiated to develop an efficient vessel in which to utilize such a precipitant. The successful accomplishment of this objective has been reported separately by A. E. Back.[10]

A pressing need for more copper production, combined with the necessity for a prolonged delay to bring a source of powdered iron into production, dictated that a further effort be made to improve precipitation methods employing available scrap iron. Using the basic cone configuration, a new vessel was designed for a scrap iron feed; this vessel has now been found to have advantages not available in other precipitating processes. Its operation, design, and copper product differ from that of the powdered iron cone. It is a compact unit lending itself to automatic control, low iron consumption, and self-cleaning of copper precipitates. Furthermore, it results in the production of a higher purity cement copper than that resulting from a launder system. The operating characteristics of the cone precipitator are based on a high-velocity, rapid through-put of copper-bearing solutions and an intimate contact of the solution with clean active iron for precipitation of copper.

Our experience comparing the relative effectiveness of powdered iron and scrap iron as precipitants clearly indicated that the three basic reactions proceed at different rates. Furthermore, these reactions are temperature dependent, another characteristic of rate reactions. Fig. 2 shows the effect of temperature on increasing the copper precipitation rate. Thus, a rapid contact of solution with iron surfaces promotes reaction 1, the copper precipitation reaction, by removing the diffusion layer. The resulting suppression of the acid on iron reaction (Eq. 3) brings about a very real and profitable reduction in iron consumption.

Wadsworth and co-workers have defined the mechanisms involved in copper precipitation on iron[11] as:

1) diffusion of reactants to the surface,
2) adsorption of reactants on the surface,
3) chemical reaction at the surface,
4) desorption of products from the surface, and
5) diffusion of products away from the surface.

They have quantified item 1) for laboratory conditions in the terms of stirring speed of a mechanical agitator. We have likewise determined an apparent optimum range of conditions in the terms of rate of

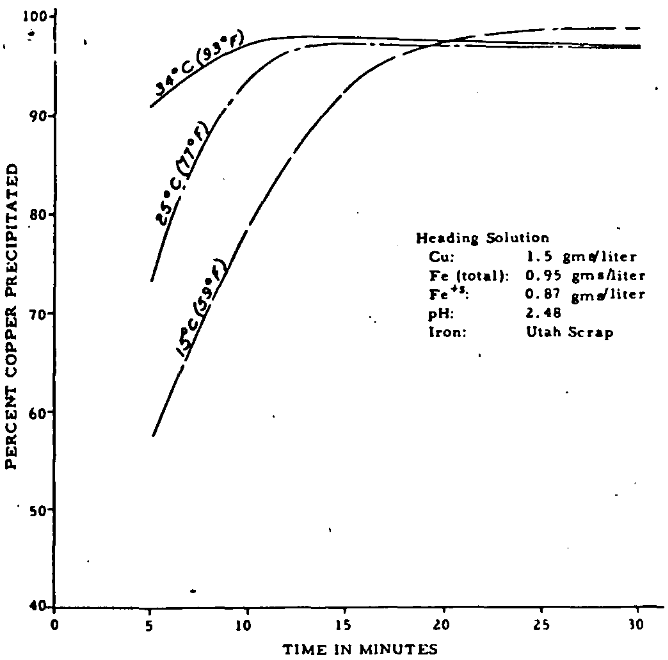


Fig. 2 — Copper cementation as a function of temperature.
**Conversion factor 1 g = 8.345 lbs per 1000 gal of solution.*

solution flow through full production size vessels of varying sizes. They further have found that the copper precipitation reaction is a first order rate reaction and that even under some conditions the reaction seems to follow zero-order kinetics. Thus, with adequate agitation, rapid precipitation will occur and acid consumption by iron will be minimized.

The nucleation mechanism has also been studied by Wadsworth[11] with the following conclusion: "The precipitated copper adheres to the iron as a spongy layer at low speeds, peels off in the form of bright strips at medium speeds, and as a fine powder at high speeds." The low speed agitation results defined by Wadsworth correspond to the action in a gravity launder. Medium speed results are of the same magnitude as those we have found optimum for promoting the precipitation reaction, and thus a dual advantage is gained by dynamic contact of solution with iron.

The theory of diffusional control is further enhanced by experiments employing iron turnings as a precipitant. Clean turnings, extensively laced with submicron cracks, present a larger surface area to the small hydrogen ion than to the much larger copper ion. Thus, acid attack by pore diffusion is possible in the cracks. Turnings in this form yielded copper precipitation at a consumption of over 3 lbs iron per lb copper, as compared with 1.5 lbs per lb of copper when using shredded iron in a precipitation cone.

Another portion of these turnings was crushed in a hammer mill to powder size. In a similar copper-bearing solution, the iron consumption was only 1.2 lbs iron per lb copper. The cracks had thus been opened, exposing all surfaces to the faster copper precipitation reaction.

The granular and dense type of copper produced under dynamic precipitation conditions results in a product that can be filtered readily to a low moisture content. This is in contrast to the thixotropic-type of precipitates produced in the launder-type plant which, after filtration, may contain 35% to 40% moisture.

The experimental precipitation cone as developed to employ these principles at the Utah Copper Div. of the Kennecott Copper Corp. is capable of processing high volumes of copper-bearing solutions. Fig. 3 is a photograph of a cross-sectional area of the precipitation cone. The vessel consists of a 14-ft diam tank, 24 ft tall, into which is mounted an inverted cone 10 ft in diam by 10 ft high. The outer 14 ft diam tank contains a 45-degree sloped false-bottom floor from one side of the tank to a bottom side discharge at the opposite side. The annular space between the inner cone and the tank is covered by a heavy gauge stainless steel screen. The screen is mounted as a continuation of the cone and is anchored to the cone and tank. The cone supports a pressure manifold that consists of six vertical legs with each leg containing a series of nozzles directed inward from the tangent to the cone and upward from the angle of the legs of the manifold. The nozzles are arranged in such a manner as to create a vortex when the copper-bearing solutions are pumped through the manifold into the cone. The inner cone and the area of the tank above

Fig. 3 — Photograph of a cross-sectional area of Kennecott's cone precipitator (patent pending). (Photograph courtesy Bechtel Corp.)

will contain 26 cone precipitator units. The plant will be of modular construction arranged in a manner to permit the solution flow to pass through two cones in series. This will provide an operational safety factor for optimum stripping. Fig. 6 is a photograph of a model of the new copper precipitation plant under construction at Kennecott's Utah Copper Div. The structure rising above the cone tanks is a movable scrap iron feeder fed by a conveyor belt from the scrap storage yard. Shredded scrap iron is added to each cone intermittently as required.

Table I. Typical Data (Daily Average) Comparing The Operating Results of an Experimental Precipitation Cone with a Gravity Launder

Heading	Cone Precipitator		Heading	Launder Precipitator	
Lbs Cu/ 1000 Gal	Cu Recovery, %	Soluble Fe Factor	Lbs Cu/ 1000 Gal	Cu Recovery, %	Soluble Fe Factor
17.6	92.4	1.13	15.1	98.3	2.85
14.8	93.2	1.45	15.4	89.4	2.15
14.8	88.2	1.83	16.1	85.1	2.04
13.3	94.3	1.60	15.7	87.5	2.26
12.9	91.0	1.49	14.4	85.8	2.30
13.3	92.4	1.77	14.8	90.7	2.17
15.8	93.6	1.86	14.6	93.7	2.16
16.2	93.3	1.74	14.7	91.3	2.22
15.9	95.3	1.54	15.1	91.5	2.25
16.4	96.4	1.74	14.6	84.6	2.40
16.2	95.4	1.10	13.7	84.4	2.47
13.9	94.0	1.72	14.0	95.8	2.70
Average					
15.1	93.3	1.58	14.9	89.9	2.33

Two cones have been operated in series to determine the effect of the second cone on total iron consumption when only partial stripping has been accomplished in the first cone. This two stage treatment can provide a better over-all control without increasing iron consumption beyond that required by a single stage treatment yielding the same recovery. Variations in the copper content of the influent solutions to the second cone have ranged, for test purposes, from 1.5 lbs of copper per 1000 gal to as high as 15 lbs per 1000 gal, yielding the same low content of copper in the tailing from this second cone. The copper-bearing solutions are chemically conditioned in passing through the first cone, thus resulting in rapid and effective stripping of the remainder of the copper in the second unit.

A further variation of the two-stage system is effectively employed in the recently expanded precipitation plant at the Kennecott Chino Mines Div. Larger cone units, each 20 ft in diam and 24 ft high, and thus capable of processing substantially greater volumes of solution, are used to recover over 80% of the copper in the first stage. The conditioned cone tailing is then passed through the pre-existing launder plant for final stripping. A single launder cell, which previously had a capacity of 300 gpm, can now strip copper from 1000 gpm of the conditioned solution at a relatively low iron consumption. Thus, the combination of cones and launders has provided the technical advantages of the cone precipitator in a plant of greatly expanded capacity at minimum cost.

Two, single-stage cone precipitators of the same basic design as the Utah cones, are also producing copper at Kennecott's Nevada Mines Div. In addition,

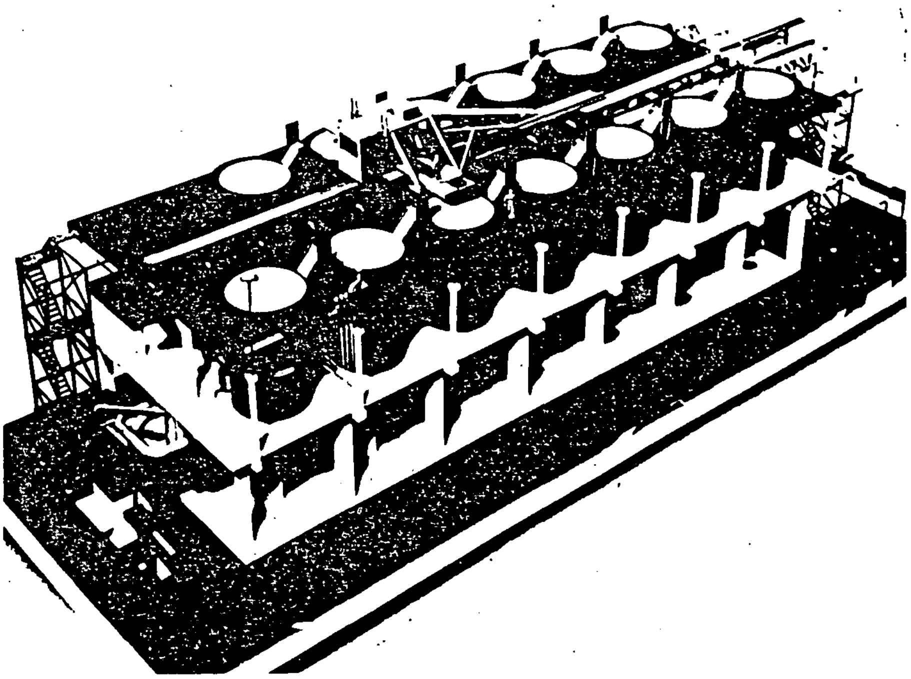


Fig. 6 — Photograph of a model of the cone precipitation plant now being constructed. Each module contains 13 cone precipitators. (Photograph courtesy Bechtel Corp.)

the stainless steel screens are filled with shredded detinned iron scrap, such as is commonly used in the precipitation of copper. The shredded iron is 'coned' to the top of the tank. This large mass of iron in a confined vessel has proved to be an effective heat retaining medium, thus enhancing the reaction kinetics.

Copper-bearing solutions are pumped through the manifold with the nozzles injecting the copper-bearing solutions into the mass of iron. The injection of the solutions has the effect of not only rapidly precipitating copper, but also removing the metallic copper from the iron surface, thereby exposing clean, fresh iron. Fig. 4 is a diagrammatic sketch of the cone precipitator showing solution inflow and copper precipitate discharge.

The precipitation cone is a continuously operated unit that is self-cleaning as to copper precipitates and eliminates the need for the conventional approach of labor with fire hoses to wash the copper precipitates from the precipitator, as is the practice in many launder-type plants. The pressure and velocity of the solutions in the lower conical section tend to move the copper precipitates in the same manner as an elutriation column, upward and out of the cone into the reduced velocity zone created by the larger diam of the holding tank. The copper precipitates settle down through the stainless steel screen and accumulate on the sloped false-bottom of the tank. The copper can then be discharged intermittently with the use of a pneumatically operated valve on a time cycle or bled continuously through a small diameter pipe into a thickener or holding basin. Fig. 5 is a photograph of the experimental cone-type copper precipitator at the Kennecott Copper Corp., Utah Copper Div. The copper precipitates produced in this manner are of substantially higher grade than the conventional cement copper produced in a launder-type plant. They typically will analyze 90% to 95% copper, 0.1% to 0.2% iron, 0.1% to 0.2% silica, and 0.1% to 0.2% alumina with the balance of the impurity being primarily oxygen.

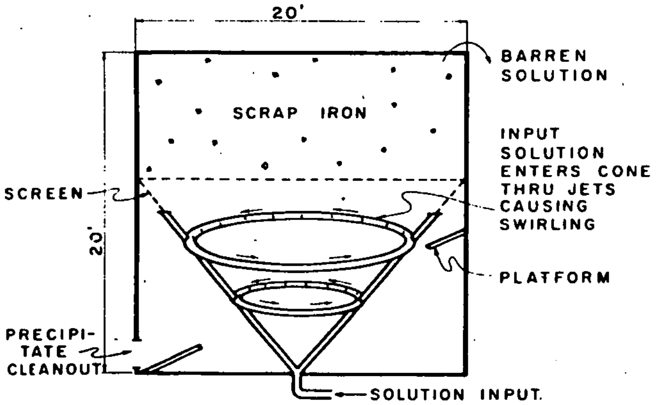


Fig. 4 – Sketch of cone precipitator showing solution inflow and copper precipitate discharge.

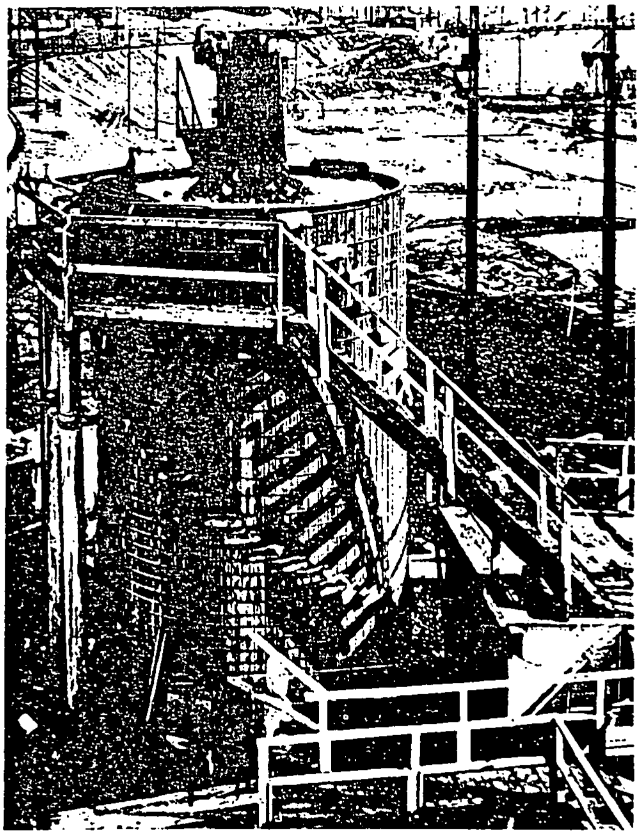


Fig. 5 – Photograph of experimental precipitator at Kennecott Copper Corp., Utah Copper Div.

The prototype precipitation cone was operated continuously for seven weeks in one test. After this test run, using commercial shredded iron identical to the material used in the launder plant, the unit was shut down and examined. A total of 18 in. of trash remained in the bottom of the inner cone. This residue consisted of pieces of concrete, rocks, granular copper, and some chunks of copper-plated steel shafting. This small amount of residue was striking evidence that most of the trash inherent in the scrap iron had been masticated by the dynamic action in the cone and discharged in the tailing solutions. In comparison, a launder plant usually requires daily washing with high pressure hoses.

During seven weeks of continuous operation of the experimental cone, copper recovery averaged 89.7% with inclusion of data for periods in which wide fluctuations in the addition of shredded iron was experienced. Sustained periods during optimum operating conditions resulted in copper recoveries exceeding 95% in the single cone.

Results comparing the performance of an experimental precipitation cone operating at approximately double the flowrate of the standard-sized gravity launder with which it is compared are presented in Table I. These results represent daily averages for the same operating periods.

As a result of the successful operation of the experimental cone at the Utah Copper Div., a cone-type precipitation plant is now being constructed which

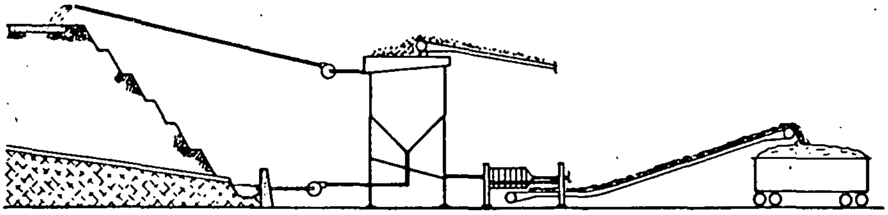


Fig. 7 – Diagrammatic sketch of the copper leaching and precipitation system as used at Kennecott Copper Corp.

a modified cone, using gravity flow from a hillside head tank at a relatively low injection pressure is under development at the Kennecott Ray Mines Div. A further variation in this particular design provides for cyclical operation with periodic dumping of the copper through the bottom clean-out valve.

In summary, operating experience with precipitation cones has demonstrated that the application of kinetic principles results in the production of precipitate copper of a more granular, higher purity form and at a lower iron consumption than is possible with the older launder methods. High capacity, versatile, precipitating vessels are now available with features permitting automatic control and mechanized materials handling. An old art has been modernized. Fig. 7 is a diagrammatic sketch of this new copper leaching and precipitation system.

REFERENCES

[1] John C. Kinnear, Jr.: *New Developments In The Large Scale Porphyry Copper Operations of Kennecott Copper*, presented January 11, 1966, to joint meeting Minnesota Section AIME and University of Minnesota Mining Symposium.

[2] S. J. Wallden, S. T. Henrickson, et. al.: Electrolytic Copper Refining at High Current Densities, *Journal of Metals*, 1959, vol. 11, pp. 528-534.

[3] C. B. Kenahan, D. Schlain: Deposition of Copper and Zinc from Sulfate and Cyanide Electrolytes, *U.S. Bureau of Mines Report of Investigation 5890*, 1961.

[4] D. W. Agers, J. E. House, R. R. Swanson, and J. L. Drobnick: A New Reagent for Liquid Ion Exchange Recovery of Copper, *Mining Engineering*, Dec. 1964, vol. 17, No. 12, p. 76.

[5] T. A. A. Quarm: Recovery of Copper from Mine Drainage Water by Ion Exchange, *Transactions of Institute of Mining and Metallurgy*, 1954-55, vol. 64, pp. 109-117.

[6] C. H. Keller: *Recovery of Copper from Copper-Bearing Solutions*, U. S. Patent 2,390,540, assigned to The Dow Chemical Co.

[7] P. J. McGauley: *Treatment of Sulfide Minerals*, U.S. Patent 3,053,651.

[8] F. M. Monninger: Precipitation of Copper on Iron, *Mining Congress Journal*, October 1963, vol. 49, No. 10, pp. 48-51.

[9] John Huttl: Anaconda Adds 5000 TPD Concentrator to Yerington, Enterprise at Weed Heights, *Engineering and Mining Journal*, March 1962, vol. 163, p. 74.

[10] A. E. Back: *Use of Particulate Iron In The Precipitation of Copper from Dilute Solutions*, presented 95th Annual Meeting of AIME, New York, N.Y.

[11] R. M. Nadkarni, C. E. Jelden, K. C. Bowles, H. E. Flanders, and M. E. Wadsworth: paper presented 95th Annual Meeting of AIME, New York, N. Y.

APPENDIX C

USE OF PARTICULATE IRON IN THE PRECIPITATION OF COPPER FROM DILUTE SOLUTIONS

by

A.E. Back

(Transactions of SME, March 1967, Vol.238)

USE OF PARTICULATE IRON IN THE PRECIPITATION OF COPPER FROM DILUTE SOLUTIONS

by A. E. Back

A method is described in which particulate iron, as distinguished from high purity iron powders used in powder metallurgy, is a precipitant for copper contained in dilute solutions. A new precipitation apparatus utilizes particulate iron precipitants to recover copper more efficiently than is possible in a conventional launder precipitation plant.

In the search for lower cost methods of recovering copper from mine water, the use of sponge iron or particulate iron (as distinguished from iron powder used in powder metallurgy) as precipitants in place of tin cans, detinned scrap iron, or scrap iron is an intriguing possibility. The relatively faster copper precipitation rate obtained with particulate iron as compared to scrap iron promises economic and processing advantages when, and if, particulate iron becomes competitive cost-wise with available scrap iron. Kennecott has developed a precipitation cone,[1] see Fig. 1, utilizing various particulate iron precipitants and has demonstrated the process successfully in a prototype at a flow rate of approximately 1000 gal of solution per min. Essentially complete precipitation of copper is obtained, the cone overflow solution is clear and contains no particulate copper, and the iron factor is more favorable than in a conventional launder plant.

Initially, sponge iron produced at the Ray Mines Div. Kennecott Copper Corp. for use in a Leach-Precipitation-Flotation process was used in exploratory tests to develop a suitable apparatus to take advantage of the rapid precipitation rate and efficient iron utilization for the recovery of copper from dilute solutions. Here it was demonstrated that, when sponge iron was added to a launder, the particles of iron collected in the bottom of the launder and tended to cement together. This resulted in incomplete precipitation of the copper and inefficient utilization of the iron. Next, sponge iron was suspended in glass columns to study the dynamics of the precipitation reaction. Again it was observed that the particles tended to cement together at low solution flow rates, while at higher flow rates large amounts of the precipitant overflowed the column, resulting in loss of precipitant and an inefficient process. Next, the precipitant was suspended in an inverted cone in a rising column of solution. Dynamic suspension of the solids was maintained at relatively rapid solution flow rates with instantaneous and complete precipitation of copper and with improved iron utilization. The first tests were conducted on a batch basis and it was determined that a dynamically suspended bed of precipitant was necessary for satisfactory operation. More than 99% of the metallic iron could be converted to metallic copper by careful control of the residence time in the cone and copper precipitation was essentially complete, even when the available iron precipitant was almost depleted.

Several cones were built, ranging in capacity from 1 to 200 gal of feed solution per min, and finally a prototype cone was constructed which was 20-ft high

---

A. E. BACK is Assistant to the Research Director, Western Mining Div., Kennecott Copper Corp., Research Center, Salt Lake City, Utah. TP 66B87. Manuscript, February 1966. New York Meeting, February 1966. Discussion of this paper, submitted in duplicate prior to June 15, 1967, will appear in SME Transactions, September 1967, and AIME Transactions, 1967, vol. 238.

Reprinted from Transactions of SME, March 1967, Vol. 238

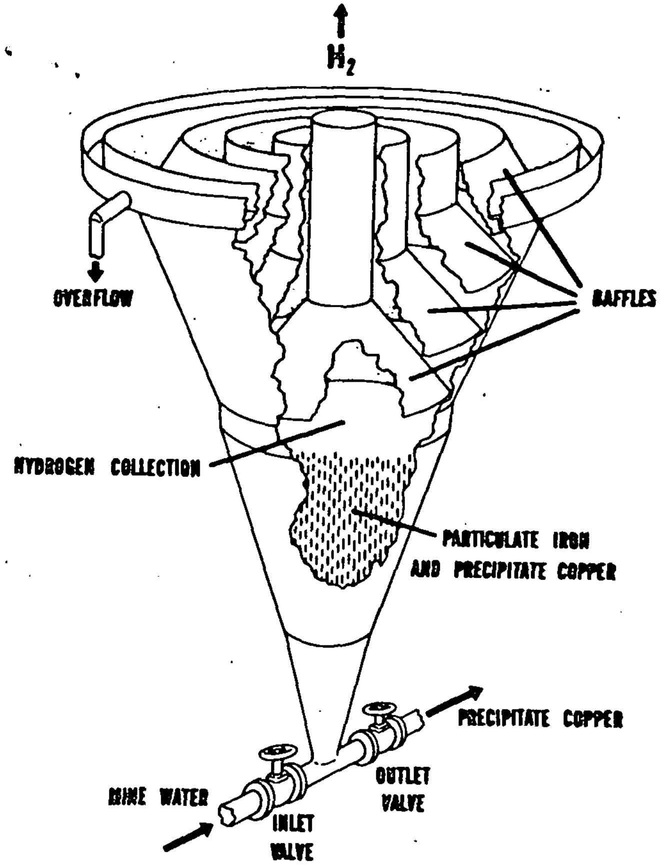


Fig. 1 – Precipitation cone, developed by Kennecott.

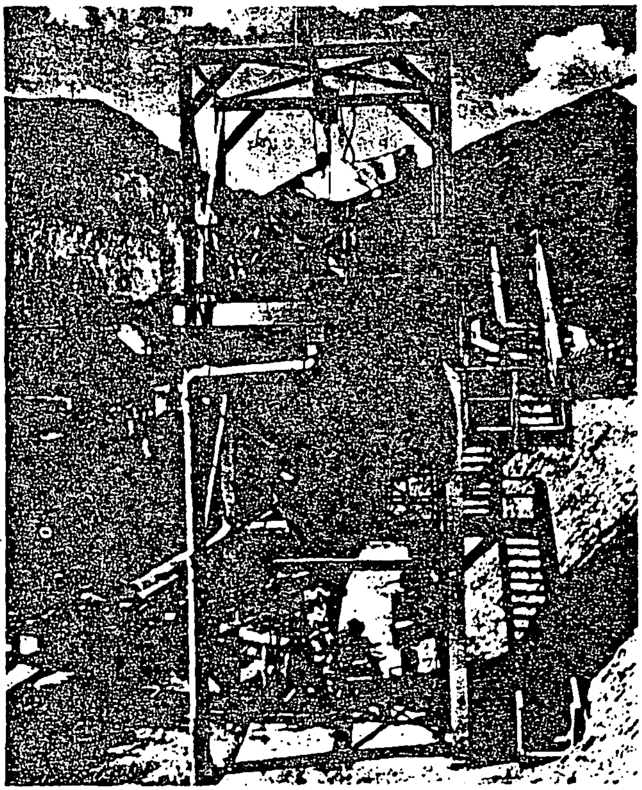


Fig. 2 – Prototype cone; approximate capacity, 1000 gal of solution per min.

and 20 ft in diam, with an approximate capacity of 1000 gal of solution per min. This cone is illustrated in Fig. 2. Tonnage samples of particulate iron have been evaluated in the prototype cone to confirm the results of small scale cone tests. The effectiveness of a precipitant using 10-lb samples in a small cone precipitator can be determined with the same confidence as testing tonnage lots in the prototype precipitator.

DESCRIPTION OF COPPER CEMENTATION PROCESS

The following competitive reactions occur in the precipitation of copper from dilute acid solutions using metallic iron as the precipitant

$$\text{Fe (metal)} + CuSO_4 \longrightarrow \text{Cu (metal)} + FeSO_4 \qquad [1]$$

$$\text{Fe (metal)} + Fe_2(SO_4)_3 \longrightarrow 3\ FeSO_4 \qquad [2]$$

$$\text{Fe (metal)} + H_2SO_4 \longrightarrow H_2 + FeSO_4 \qquad [3]$$

Reactions [1] and [2] proceed almost simultaneously in the cone precipitator because the metallic copper produced in [1] catalyzes reaction [2]. Reaction [3] proceeds more slowly than [1] and [2]. In the solutions studied, the bulk of the iron is used to precipitate copper because of the relatively small concentration of ferric iron present. Because of the intimate contact in the cone between the metallic iron and the solution, copper precipitation is complete with less neutralization of the acid [3] by the iron than in a conventional launder plant. Thus, the cementation reaction is more efficient with respect to iron consumption and an iron factor (weight ratio of iron consumed to copper precipitated) approaching more closely the theoretical value of 0.9 is obtained. In addition, advantage is taken of reaction [3] to monitor the course of the precipitation reaction [1] by measuring the change in the rate of hydrogen evolution. Hydrogen evolution decreases as the amount of available metallic iron is diminished, and provides a sensitive measure for process control without chemical analysis. Addition of a measured batch of precipitant to the cone, with continuous discharge of barren solution at the overflow and intermittent discharge of the precipitates from the apex of the cone, was found to be most desirable.

During the period of this investigation, some 14 different samples of particulate iron from various companies and individuals have been evaluated as copper precipitants. Various samples of directly reduced iron, iron powder, iron turnings and chips, and granulated iron have been submitted. Raw material source, reductant used, and typical analyses are given in Table I.

The samples contained variable amounts of copper, sulfur, carbon, and silicon which are not listed. The grade of precipitate theoretically recoverable is

Table I. Typical Analysis of Particulate Iron Samples Tested and Grade of Copper Precipitate

Raw Material	Reductant	Analysis, % Fe (Metal)	Analysis, % Fe (Total)	Reduction, %	Calculated Grade of Copper Precipitates with 0.9 Iron Factor, %
Commercial iron powder	—	97.6	99.2	98.4	97.8
Commercial iron powder	—	96.0	97.1	98.9	96.4
Magnetite concentrates	Reformed gas	93.8	94.4	99.4	94.4
Magnetite concentrates	Solid carbonaceous	93.8	96.0	97.7	94.4
Iron turnings	—	93.0	98.0	94.9	93.7
Pelletized magnetite concentrates	Solid carbonaceous	92.0	94.4	97.5	92.8
Pelletized magnetite concentrates	Solid carbonaceous	86.8	90.7	95.7	88.0
Pelletized magnetite concentrates	Solid carbonaceous	86.6	93.0	93.1	87.8
Pelletized magnetite concentrates	Solid carbonaceous	83.6	97.4	85.8	86.6
Magnetite concentrates	Solid carbonaceous	79.4	87.5	90.6	81.0
Reverberatory slag	Carbon	65.4	83.4	78.4	67.7
Pyrite cinders	Solid carbonaceous	55.2	71.6	77.1	57.8
Refinery residues	Reformed gas	48.8	64.2	76.0	51.4

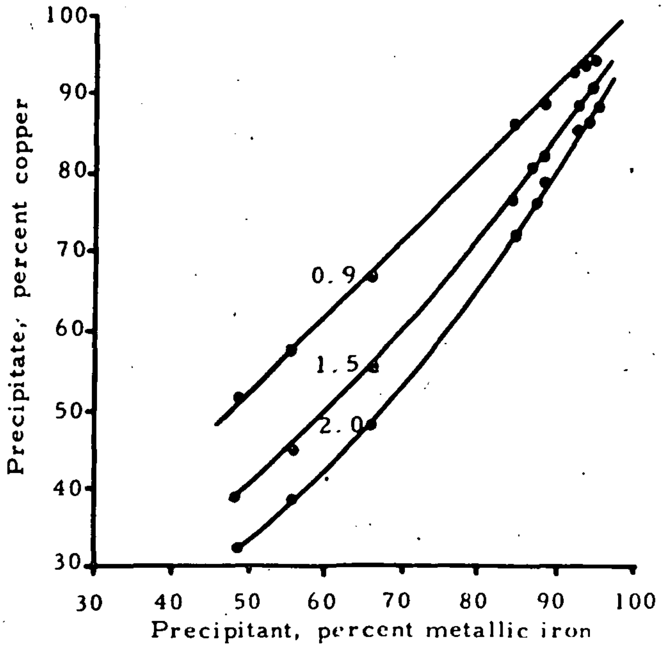


Fig. 3 — Relationship of grades of precipitant and precipitate at different iron factors.

shown graphically in Fig. 3 with theoretical 0.9, 1.5, and 2.0 iron factors. Any copper present in the precipitant is recovered in the resulting precipitates at a grade higher than that indicated in Table I. This means that the copper specification required for iron blast furnace raw materials is not critical, which could result in a lower cost precipitant, and a high grade precipitate would be smelted, which could decrease the amount of inert material required during smelting.

In general, each of the particulate iron precipitants is satisfactory for the recovery of copper from dilute solutions in a precipitation cone. The dense iron samples require finer grinding, i.e. to −35 mesh, than do the spongy iron samples, which are suitable if ground to −10 mesh. An excessive amount of −200 mesh material is undesirable because this favors reaction [3] slightly over reaction [1]. However, as long as the bed of precipitant is maintained in dynamic suspension by adjusting the rate of solution flow into the cone, variations in particle size can be accommodated and reaction [1] is favored. A high degree of reduction of the iron precipitant is desirable because of the resultant high grade precipitate. However, the cost of obtaining such a precipitant often prices it out of the market and a compromise between reduction costs for the precipitant and smelting costs for the precipitate must be resolved.

The prototype cone that is now in operation at the Utah Div. precipitation plant, Fig. 2, is equipped with a variable speed feeder so that the precipitant feed rate can be varied to meet the requirements of the process. The facility is automated so that the progress of the consumption of the available iron can be followed by the decrease in the evolution of hydrogen from the reaction,

$$H_2SO_4 + \text{Fe (metal)} \longrightarrow FeSO_4 + H_2. \qquad [3]$$

As the hydrogen evolution decreases and comes to a predetermined low value, a system of valves becomes operative thus shutting off the incoming solution and opening a discharge valve for approximately 30 sec to remove the precipitate from the lower portion of the cone. Then, the valve closes and fresh iron is fed into the cone simultaneously with the influx of fresh copper-bearing solution. The precipitates are filtered and then transferred to a conveyor for subsequent treatment. Channeling the hydrogen or reaction gases toward the center of the cone into a central exhaust system is essential because particles of copper adhere to the hydrogen bubbles and, if they are near the overflow of the cone, the particulate copper is transported into the tailings. Thus, it is necessary to install a battery of baffles in the upper part of the cone to divert the gas towards the center and allow the bubbles to burst and release the particulate copper to settle back into the precipitation zone. As this copper settles back into the apex of the cone, it is subjected to attrition and peening so that it is necessary to discharge the bed of the cone periodically to prevent attrition of the particulate copper to form a colloid which would not settle and would report as a loss in the tailings overflow.

It has been found that to obtain the best operation of the precipitation cone on a batch-continuous basis approximately 50% of the iron requirement should be fed into the cone during a 5 min time interval. The remainder of the iron is fed during the next fifteen minute period at a gradually decreasing rate. In this manner, a bed is built rapidly within the cone so that effective fluidization of the particulate iron and efficient utilization of the metallic iron as a copper precipitant are obtained. Under these conditions, the competing reactions between copper, acid, and metallic iron are regulated, the precipitation reaction being favored over the iron dissolution-hydrogen evolution reaction. In this manner, the pH range of the effluent solution in a precipitation cone is from 2.5 to 3.0 pH while in a conventional launder plant it might range from 3.0 to 3.5 pH.

ADVANTAGES OF CONE PRECIPITATOR USING PARTICULATE IRON

The advantages of a cone precipitation plant over a launder precipitation plant are many. For example, a cone precipitation plant capable of treating 10 million gal of solution per day would require an area of 10,000 sq ft whereas a launder plant similar to the Utah precipitation plant at Bingham Canyon would require an area approximately 10 times as great. Handling of the precipitant is time consuming in a launder plant because of the large bulk per unit weight of iron. The bulk density of the particulate iron samples tested varies from approximately 75 to 200 lbs per cu ft, and the material is free flowing so that it can be handled with conventional materials handling equipment. In a cone precipitation plant, the copper precipitates are discharged automatically into a filtration system while in the launder plant the precipitates are washed manually from the launders. It has been found generally that the iron efficiency in the cone plant is superior to the launder plant; in fact, in treating the same solution through the launder plant and through the prototype cone, it was found that savings of 0.5 to 1.0 lb of iron per lb of copper could be realized. In addition to this saving in iron, essentially all of the soluble copper is precipitated from the solutions so that a circulating load of dilute copper solution to the leaching system is eliminated.

In the operation of the prototype precipitation cone, it has been demonstrated conclusively that the decrease in the rate of hydrogen evolution is a sensitive indirect measure of the completeness of the precipitation reaction which can be used for process control. The only chemical analyses required are for accountability because of the completeness of the precipitation reaction within the cone. Another advantage in collecting the hydrogen in a central exhaust system is to operate under a slight pressure of atomic hydrogen, which seems to accelerate the copper precipitation rate and to prevent reoxidation of the ferrous iron.

It is interesting to note that when particulate iron is used as the precipitant, copper recoveries from solution are in the range of 95% before a bed of dynamically suspended precipitant is established and, after the bed has been established, recoveries of more than 99% are common regardless of the type of precipitant used and the amount of metallic iron available for precipitation. Thus, it is possible to precipitate 99% of the copper even when less than 1% metallic iron is available in the partially spent precipitant, which insures excellent iron efficiency. After a batch of precipitant has been added and is essentially utilized, a system of valves is actuated in which the solution inflow is shut off and the precipitate that collects in the lower portion of the cone flows out of the valve into a receiving tank from which it flows by gravity into a filter. The precipitate slurry is dewatered in a filter from approximately 15% to 50 to 60% solids and the filter cake is then discharged as a moist cake onto a conveyor for transport to the next processing step.

SOURCES OF PARTICULATE IRON

Sponge iron produced by the direct reduction of magnetite concentrates, iron ore, or pyrite cinders may be considered as sources of high grade reactive precipitants. Some ten different sources of this directly reduced iron have been made available for testing at the Research Center and at the Utah Copper Div. in the prototype cone. All of these precipitants were found to be suitable for the process. Factors of cost and tonnage availability remain to be resolved before any of these are used as the source precipitant. Commercial, high purity iron powder is also satisfactory; however, the cost of such a precipitant is not competitive with other indicated sources. Detinned scrap or burned tin cans are suitable precipitants for launder plants or for the cone precipitator described by Spedden, Malouf, and Prater.[2] Because of the necessity of keeping the entire bed of precipitant in dynamic suspension and maintaining a clear overflow, it is not possible to suspend large pieces of scrap iron satisfactorily. Cast iron or steel chips have been considered a potentially inexpensive source of precipitant; however, as produced, they contain grease and are relatively coarse. Degreasing of the chips and grinding them to approximately 10 mesh is required to prepare them for use in the precipitation cone, thus adding to the cost of iron. This type of precipitant has also been tested in the pilot plant.

At the present time, Kennecott is investigating the recovery of a suitable precipitant from reverberatory slag. Selective reduction of the slag with coke and pyrite yields a metallic product that is relatively brittle. Products of varying composition have been tested in the prototype cone, and again it was determined that this type of iron is a suitable precipitant with the same favorable characteristics of copper precipitation and iron utilization as the other iron samples that have been evaluated. Several potential producers of directly reduced iron have graciously supplied samples, some of them being tonnage samples for evaluation in the prototype cone. Other smaller samples have been evaluated in a cone precipitator with flow rates of about one liter per min. In these small scale tests, fluidization characteristics of the particulate iron samples are determined and the particle size of the precipitant is evaluated. Generally it is necessary to have a particulate iron of -35 mesh size. However, when sponge iron is used and a spongy product with a relatively high specific surface area is used, a 10 mesh product has been found satisfactory.

SUMMARY

At the present time, a prototype precipitation cone is being operated to establish operating parameters and to develop engineering data. A cone precipitation plant utilizing particulate iron as the precipitant to recover some 400,000 lbs of copper per day at the Utah Copper Div. of Kennecott Copper Corp. will be installed as soon as an economic source of particulate iron precipitant can be developed. Several potential suppliers have submitted samples which were found to be satisfactory. However, no firm contract has been made because of the difficulty of producing a particulate iron precipitant competitive with current scrap iron prices. Kennecott is now evaluating the previously mentioned process for the recovery of iron precipitant from copper reverberatory slag which, as a captive source, would make it unnecessary to rely upon outside suppliers.

REFERENCES

[1] A.E. Back, K.E. Fisher, and J. Kocherhans: *U.S. Patent* 3,154,411, Process and Apparatus for the Precipitation of Copper from Dilute Acid Solutions, October 27, 1964.

[2] H.R. Spedden, E.E. Malouf, and J.D. Prater: *Use of Cone-Type Copper Precipitators to Recover Copper from Copper-Bearing Solutions.* Presented at 95th Annual Meeting of AIME, 1966, New York.