

Bulletin

Nonferrous Metals

Technical Awareness

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This report is intended to provide state-of-the-art information on nonferrous metals developments. This overview document should be useful in providing a background perspective to assess environmental conditions for the industry. The "user" should be aware that some of the technical aspects may be changed in the final report.

NON-FERROUS METALS INDUSTRIES

IN THE NEWS



... a publication of the USEPA Metals and Inorganic Chemicals Branch, Office of Research and Development. It is designed to spotlight selected recent events, concerns, and technology within the nonferrous metals industry as reported in the open literature, and does not necessarily reflect the views of the USEPA.

July 1979 HIGHLIGHTS

- Aluminum facilities closed in 1977 by energy constraints are being reopened in response to strong market demands. (Items 3 and 4, Chemical Engineering and American Metal Market.)
- Inventory reduction, continued foreign production ills, and more attractive demands and prices for copper and molybdenum are sparking renewed activity in the domestic copper industry. (Items 8-12, Paydirt and other sources.)
- Noranda is the winning bidder who may reap benefits in taking over the Papago-owned Lakeshore copper facility recently written off by Hecla and El Paso. (Item 12, Northern Miner.)
- OSHA and EPA lead standards continue to foster debate and rhetoric between industry, government, and public interest groups, and appears to be far from a settled issue. EPA, Office of Research and Development has initiated a research program in this area. (Item 29) (All items under "Lead" numerous sources.)
- Demand and prices continue to encourage exploration and reactivation of gold and silver operations. Small operations, in particular, are proliferating. (See "Precious Metals"; The Mining Record and other sources.)

Individually contributed news items may be submitted to Metals and Inorganic Chemicals Branch, Industrial Environmental Research Laboratory, USEPA, 5555 Ridge Avenue, Cincinnati, Ohio 45268, 513-684-4491.

Prepared by Battelle's Columbus Laboratories under EPA Grant R805095.

Aluminum

(1) One of the nation's largest secondary aluminum smelters, Michigan Standard Alloys, Inc., of Benton Harbor, Michigan, will phase out its aluminum operations and its dross mill within 30 days because of problems with pollution regulations.

(American Metal Market)

(2) In an effort to "break the bauxite barrier", Alumet intends to open-pit mine alunite on 13,000 acres and build a processing complex on 5,000 acres of land in Beaver County, Utah. The company has been trying since mid-1975 to obtain an environmental impact statement (EIS) for the \$500-million project. They paid a \$93,250 assessment for preparation of the statement by the Bureau of Land Management (BLM) but have refused to pay a second assessment for \$166,750. After 2 years' litigation, the court enjoined the BLM from charging Alumet. The Department of Justice and Alumet will argue the case later this year.

(Engineering and Mining Journal)

(3) Reynolds Metals Company will reopen its primary reduction plant at Corpus Christi, Texas, because of an increase in demand for aluminum. Production from a 57,000-ton/yr potline will begin in May, bringing Reynolds to 94 percent of its primary aluminum capacity. A second potline will be activated as soon as the first returns to production.

(Chemical Engineering)

(4) Alcoa is also reactivating its Port Comfort, Texas, smelter. Two potlines are scheduled to be reactivated in May and the third in July. The company's primary aluminum production will be increased by 90,000 short tons per year.

(American Metal Market)

(5) Ogden Metals will increase its production capacity for secondary aluminum ingot by 10 percent to 300 million pounds per year, with a \$1.6-million investment in its Wabash Alloys, Inc., subsidiary in Cleveland, Ohio, and in Wabash, Indiana. The Wabash unit will also enter the scrap market for aluminum and bimetallic cans.

(American Metal Market)

Beryllium

(6) Brush Wellman, Inc., will expand and upgrade its minerals processing plant at Delta, Utah. A \$0.5-million beryllium sludge treatment process will be installed in addition to a \$1-million plant for uranium production.

(USBM Minerals and Materials)

Cobalt

(7) An investigation is being undertaken into the feasibility of reopening the Blackbird mine at Cobalt, Idaho. An active producer in the 1950's, new technology will enable the recovery of more cobalt from the ore in the milling process. There is presently no domestic supply of mined cobalt, and less than 4 percent of the domestic demand for refined cobalt is supplied by U.S. producers.
(The Mining Record)

Copper

(8) The Duval Corporation resumed operations at its Esperanza copper-molybdenum mine near Tucson in Pima County, Arizona, in April. It has been closed since October, 1977. Production is being resumed because of the recent firming trend in the copper market and the strong, world-wide demand for molybdenum. Production capacity of the Esperanza concentrator is 33 million pounds of copper and 3 million pounds of molybdenum annually.
(New Mexico Paydirt, Skillings' Mining Review)

(9) Cypress Mines Corporation will resume ore stripping operations in May and restart the concentrator early in the third quarter of 1979 at the Pima mine near Tucson, Arizona, down since September, 1977. Production will reopen on a limited scale because of the surge in the price of copper and molybdenum. Molybdenum is a by-product contained in substantial quantities in the Pima ore. Limited production of copper is expected to total about 24,500 stpy (of contained copper) or one-third the 1977 production level.
(Arizona Paydirt)

(10) Closed since the summer of 1977, Inspiration Consolidated Copper Company will reopen its Christmas mine near Phoenix, Arizona, in about 10 weeks. Full production capacity is about 6,000 stpd of ore.
(American Metal Market)

(11) The Oracle Ridge property near Tucson, Arizona, is being developed jointly by Continental Materials Corporation and Union Miniere as an underground copper mine capable of supplying 14,000 st of ore to the concentrator, weekly. A copper recovery of 89 percent and a concentrate grade in excess of 32 percent copper were consistently obtained during the test program. However, the project has been substantially curtailed until completion of a comprehensive study to more precisely define the ore zones and reassess the underground mining conditions. The study is expected to be completed during 1979.
(Engineering and Mining Journal, Skillings' Mining Review)

(12) The Lakeshore copper mine near Casa Grande, Arizona, has been leased by the Papago tribe to Noranda Exploration, Inc. Noranda will begin underground mining in 6 months of the 472-million-ton deposit averaging 0.75 percent copper. By September, a production of 6,000 stpd of ore is expected to be reached. Noranda will pay \$10 million for the plant written off

by Hecla and El Paso to the tune of hundreds of millions of dollars. An initial \$1.6-million lease and preroyalty payment is also involved.
(The Northern Miner, Engineering Mining Journal)

(13) The Nevada mines division of Kennecott Copper Corporation has begun engineering work on the planned McGill tailings retreatment project. Accumulating for over 70 years from the Ruth mine which closed in 1978, natural classification has resulted in a deposit of minable grade copper-bearing material. Whether or not Kennecott will proceed with this will depend upon forthcoming engineering data and economic analysis.
(Skillings' Mining Review)

(14) Smelting operations at the Kennecott Copper Corporation's McGill, Nevada, plant will be suspended for at least 3 months, beginning in February, to make major repairs on reverberatory furnaces and boilers.
(New Mexico Paydirt)

(15) Anaconda Company will temporarily curtail operations during April at its copper smelter in Anaconda, Montana, due to lack of feed. Increase in freight rates to ship concentrates from Canada is blamed. Carr Fork will provide feed, commencing late in 1979.
(USBM Minerals and Materials)

(16) The Duval Corporation has permanently closed down the sulfide copper ore operation at its Battle Mountain, Nevada, plant. The sulfide milling operation was converted to process precious metal ores after the sulfide ore production tapered off because of low ore grades.
(New Mexico Paydirt)

(17) Park City Ventures, a joint venture of Anaconda and Asarco has abandoned, at this time, plans to reopen the Ontario No. 3 mine. It was closed on February 15, 1978, because of high water and unstable rock conditions.
(Engineering Mining Journal)

(18) Asarco's Amarillo, Texas, copper refinery has begun to refine copper scrap as well as blister copper. The scrap feed facility was built in late 1978. Blister copper is being sent to Amarillo from the Tacoma, Washington smelter following the shutdown of the Tacoma refinery. The Amarillo refinery produced 270,000 tons of refined copper last year and is expected to be producing at a rate of 378,000 tpy by the end of 1979.
(Arizona Paydirt)

(19) Kennecott Corporation's Flambeau Mining Company has no plans to mine copper in the near future at the site near Ladysmith, Wisconsin. The Wisconsin Department of Natural Resources rejected a 1976 application for a mining permit, and Kennecott will not appeal this.
(Engineering and Mining Journal)

(20) Hydrometallurgical and pyrometallurgical processes for copper smelting and refining are being examined at the University of Utah in a \$233,000 renewal of a Department of Energy study. Energy usage is being

assessed from the time the ore is mined until it becomes high-purity, refined copper. The goal of the research is to determine the least energy-intensive copper producing methods.
(Mining Engineering)

(21) A report has been released by the Inform organization, strongly criticizing copper companies who operate 16 smelters in the U.S. for failing to adequately protect their workers from exposure to cancer-causing agents. Half of the 2,800 workers examined were exposed to hazardous levels of arsenic and a fourth to hazardous levels of sulfur dioxide. Many others are exposed to dangerous levels of copper dusts, fumes, cadmium, silica, and noise. Reactions from industry: "replete with errors and misstatements"; none of the findings is correct"; "unqualified sources, hearsay, gossip, and conjecture", etc.
(American Metal Market)

(22) Reversing its previous stand on the subject, the U.S. in late February proposed an international agreement to stabilize the price of copper by fixing a median price to be maintained by means of a buffer stock of at least 1 million metric tons.
(Arizona Paydirt)

(23) The cost of implementing federal regulations in the copper industry, over the next 10 years, could close three major copper smelters and force those remaining to face a "no-growth situation" according to a review by the Department of Commerce. Those which possibly will close are operated by Kennecott at McGill, Nevada, by Phelps Dodge at Douglas, Arizona, and by Asarco at Tacoma, Washington. Adherence to the regulations will require investments of \$1.8 billion and total expenditures of \$3.5 billion, and will increase the price for copper.
(American Metal Market)

(24) In 1974, Kennecott's Ray Mines Division in Arizona discontinued the practice of charging copper-rich converter slag to the reverb to allay air pollution, and instead, now returns copper values to the head of the line (crushing) operation. This has caused logistics problems. RMD is adapting to slag inventory buildup by additional truck haulage and the installation of a new grizzly to handle this excess inventory.
(New Mexico Paydirt)

Government Regulations

(25) In order to meet national air quality standards by the end of 1982, the New Mexico Environmental Improvement Board has adopted a plan which focuses particular attention on 10 areas that fail to meet air quality standards. One of these areas is near Silver City, location of the Hurley smelter. To meet this plan, Kennecott will probably have to spend \$140 million to control sulfur dioxide and particulate matter at Hurley. (Added in proof: Kennecott has big plans for Chino Mines Division, including Hurley smelter--see next Bulletin.)
(New Mexico Paydirt)

Lead

(26) Mr. D. B. Craig, president of St. Joe Minerals Corporation, has suggested the following points on which both government and the lead industry must agree to reach long- and short-term approaches to regulatory enforcement. These are: environmental improvements made with existing technology without exorbitant costs; research and development for alternative conversion methods; and financing both programs so that industry should not have to bear the entire cost. He also called for new technology for development of scrap and stressed the need to make workers in the user and production industries aware of the problem.
(American Metal Market)

(27) The District Court of Appeals of Washington, D.C. has stayed the most costly provisions of the OSHA standard for lead exposure. Requirements for engineering controls, compliance plans, and construction of new hygiene facilities were stayed, while the standard which eventually requires a limit of 50 micrograms of lead/m³ of air over an 8-hour period went into effect on March 1, 1979.
(American Metal Market)

(28) Three years after OSHA proposed a standard limiting worker exposure to lead, the controversy concerning it still rages. The prime target is the 50 microgram/m³ of air standard for exposure. Industry claims the standard should be based on actual lead in the blood, which should be kept below 80 micrograms per gram. Legal challenges have been filed by several groups, and the processes will drag on for some time. (A Supreme Court decision concerning benzene this year, in which it will decide whether OSHA must come up with strict risk/benefit analyses before issuing a standard, could destroy the lead standard rule.)
(American Metal Market, Lead and Zinc Supplement)

(29) The EPA ambient lead standard is the principal area of concern of six primary lead smelters and 50 secondary ones. EPA states there has been no positive response from the industry in the form of engineering data to be used as guidelines. EPA and NIOSH have initiated joint studies of the occupational & ambient lead problem. Contact Fred Craig, 513-684-4491 or Jim Gideon at 513-684-4295. (American Metal Market Lead & Zinc Supplement)

(30) ILZRO (International Lead and Zinc Research Organization) last year spent \$480,000 in research and development in the environmental field. ILZRO states three concepts which are particularly onerous to industry: (1) the zero threshold concept which holds that there are no safe concentrations of toxic substances; (2) the proof of safety, in which the burden of proof is shifted from those contending a substance is harmful to those contending a particular substance is safe; and (3) the adversary law requiring scientists to be identified with one side or the other in a particular issue. At stake are 9,300 direct jobs and 13,950 indirect jobs; the closing of all primary smelters; 40 of 50 secondary smelters and 6 of 200 battery manufacturing companies.
(American Metal Market Lead and Zinc Supplement)

(31) The strike by the production and maintenance employees of Kennecott's Ozark Lead Company, Sweetwater, Missouri, continues after rejection of an eleventh-hour proposal. Negotiations for contract renewal began in January; however, the present contract expired March 1 with no settlement. No date has been set for resumption of negotiations.
(American Metal Market)

Miscellaneous

(32) Laser Analytics, Inc., of Lexington, Massachusetts, has been granted a \$60,000 contract by EPA's Environmental Sciences Research Laboratory for development of a flue gas sulfuric acid vapor monitor. A monitoring instrument is needed to detect sulfuric acid in power plant and industrial smokestack emissions; however, conventional infrared techniques cannot separate gases whose wavelengths are too close in the spectrum. An infrared system using a tunable diode laser as a light source is sought by the EPA.
(Environment Reporter)

Molybdenum

(33) The development stage, estimated to cost over \$200 million, has been initiated by Cyprus Mines Corporation in their Thompson Creek, Idaho, molybdenum mine. Located near Clayton in Custer County, the deposit has reserves estimated at 233 million mt assaying 0.18 percent MoS₂ or recoverable reserves of 155 million mt averaging 0.201 percent MoS₂. Japanese steel interests are studying the possibility of financial assistance to Cyprus in developing this project.
(Engineering and Mining Journal, The Mining Record)

(34) The second largest molybdenum deposit in the world, containing over 700 million st, with grades ranging from 0.15 to 0.22 percent MoS₂, lies in Quartz Hill, Alaska, which has recently been placed in the Misty Fjords National Monument, an unminable preserve. With a potential value of \$7 billion, U.S. Borax and Chemical Corporation has already invested \$7 million in work at the site. The company feels that it has demonstrated willingness to protect the area by rehabilitating the adjacent Keta River. In March, the House Interior Committee approved an Alaska lands bill that would move the boundary line for the wilderness area 15 miles north to exclude the deposit. U.S. Borax is optimistic and will proceed with \$3 million of additional exploration work this year.
(Engineering and Mining Journal)

Nickel

(35) The Amax Nickel Refining Company, Inc., at Port Nickel, Louisiana, received the first nickel matte shipment from the Agnew mine in Western Australia on February 14. This initial 400 mt shipment is part of a 10-year

contract with the Agnew mine for its entire production up to an annual maximum of 15,800 mt. The matte contains 72 percent nickel, 6 percent copper, and 0.6 percent cobalt.
(Skillings' Mining Review)

Precious Metals

(36) The reconditioned Nancy Lee mill began processing stockpiled ore on April 5. The 2,000 tons of ore were mined at the Nancy Lee mine near Superior, Montana, and consists of copper, silver, and gold. Concentrates will be shipped to East Helena for refining.
(The Mining Record)

(37) Treasure Hill Exploration, Inc., has paid the Silver King Mines an advance royalty payment of \$100,000 on its East Hamilton property near Ely, Nevada. Treasure Hill plans to process 100,000 tons of ore per year at its plant near the property. About 225,000 tons of ore are stockpiled, with another 200,000 to 300,000 tons near the surface, averaging 6.3 ounces of silver per ton.
(Skillings' Mining Review)

(38) Gulf Oil is continuing exploration drilling on Silver King Mines' Ward Properties (14,500 acres) near Ely, Nevada. Indicated reserves of 15 million tons of copper, silver, and zinc have been found in which Gulf can earn a 51 percent interest by expending \$10.5 million over a 7-year period.
(Skillings' Mining Review)

(39) Canadian Superior Mining is conducting further drilling and metallurgical work for Ranchers Exploration and Development Corporation on Rancher's leased property near Stibnite, Idaho. Determination of the extent of an oxidized gold deposit on the property is the objective of the work by CSM. Production by open-pit mining, heap leaching, and precipitation of gold concentrates could begin soon if no metallurgical problems develop.
(Engineering and Mining Journal)

(40) The Mary Nevin Mine and mill made the first shipment of gold from the Cripple Creek, Colorado District since 1962 on February 14, 1979. On March 28, operations at the Mary Nevin Mill were expanded to a 24-hour per day schedule. The mill capacity is presently 25 tons of ore per day but will soon be expanded to 100 tpd. Concentrates are shipped to an Asarco buying station in Denver.
(The Mining Record)

(41) Deposits of gold, silver, zinc, and copper have been discovered in the abandoned Bluejacket mine near Grangeville, Idaho. Approximately 2.5-million tons of ore, with a value of \$70 million, lies in this area; however, the site is within Hells Canyon National Reservational area, so a renewal of mining may not be permitted. Attempts are being made in Congress to exclude the mine area from the wilderness classification, but it is being opposed by the Sierra Club.
(Engineering and Mining Journal)

(42) Plans to begin trial mining at the Escalante silver mine in south-west Utah are under way by the Ranchers Exploration and Development Corporation. The increase in silver prices has added impetus to this project as the mine is estimated to contain 20-million ounces. Trial mining on its Alaskan gold placers this summer is also planned by Ranchers.
(New Mexico Paydirt)

(43) The Mikado and perhaps the Little Squaw, two lode deposits belonging to the Little Squaw Gold Mining Company will begin production this year, financed and operated by Whelan Mining and Exploration, Inc. Areas along Big Creek, Little Squaw Creek, Tobin Creed, and Big Squaw Creek will be explored and mined where results are favorable. Each creek has small rich sections remaining unmined after producing 45,000 to 50,000 ounces of gold in the past.
(The Mining Record)

(44) Helena Silver Mines has acquired an electric flotation mill near Missoula, Montana. It will be used to custom mill at its present location if the Cape Nome-Hidden Treasure properties are reactivated, or to mill stockpiled material from Helena Silver's Gregory Mine. Summit Silver, Inc., would like to rent it for processing silver ore from its Baltimore Mine near Boulder, Montana.
(The Mining Record)

(45) The Sutro Tunnel at Virginia City, Nevada, is being reopened in a renewed search for gold and silver in the famous Comstock mining district. Restored and retimbered by the Houston Oil and Minerals Corporation, the gates were opened in January for crews working to restore the 4-mile-long tunnel for exploration and perhaps production. Completed in 1878, it was used until the mines closed down in 1942.
(The Mining Record)

(46) The eight largest U.S. mine producers of silver in 1977 and 1978 in millions of ounces were: Sunshine mine in Coeur d'Alene, Idaho, 3.79 and 4.95; Callahan's Galena mine in the Coeur d'Alene, operated by Asarco, 3.70 and 3.99; Hecla's Lucky Friday mine, in the Coeur d'Alene, 2.64 and 2.46; Coeur d'Alene Mines Corporation, Coeur mine, adjoining the Galena mine and operated by Asarco, 2.38 and 2.40; Anaconda's Berkeley pit copper mine at Butte, Montana, 3.19 and 2.35; Kennecott's open-pit copper mine at Bingham Canyon, Utah, 1.96 and 2.22; and Homestake's Bulldog mine in Colorado, 2.04 and 2.10.
(The Mining Record)

Titanium

(47) With the lead time at the end of last year of 40 to 50 weeks for titanium alloy, sheet and plate, and up to 70 weeks for forgings, producer's ability to deliver will depend on the availability of titanium sponge and scrap. RMI, the only U.S. titanium sponge producer selling externally, has "maximized its output", and will undertake a \$3.5-million expansion of its

sponge plant at Ashtabula, Ohio, to be completed by the spring of 1980. This will expand its current 15-million pounds annual capacity by about 25 percent.

(American Metal Market Aerospace Metals and Machines Supplement)

Vanadium

(48) Ranchers Exploration and Development Corporation has suspended plans to extract vanadium and uranium from old mine tailings at Durango, Colorado, because of delays in licensing by the Colorado Department of Health.

(USBM Minerals and Materials)

Zinc

(49) There is no expectation of a settlement in the near future of the 11-month-old strike at St. Joe Zinc Company's Balmat-Edwards mining operation in upstate New York. The Edwards mine is the largest zinc mine complex in the U.S., supplying 17 percent of the total U.S. mine production of zinc in 1977.

(American Metal Market)

(50) Exploration began about a year and a half ago on St. Joe Minerals Corporation's Smith County zinc property in Tennessee to assess its profitability. About \$5 million has been spent on shaft sinking near Carthage to determine the size and grade of the ore body. A decision is expected by late spring or early summer on opening the mine. If favorable, the joint venture of St. Joe Minerals and Freeport Zinc Company will spend \$40 million to develop the mine.

(USBM Minerals and Materials)

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RECENT PATENTS

4,141,159

METHOD AND APPARATUS FOR DEEP SEA MINING

Wilford V. Morris and George W. Sheary, III, assignors to Summa Corporation, Las Vegas, Nevada.

(This describes a sea-bed mining system comprising a nodule collector and crusher and a pneumatic lift system. Crushed nodules are pumped to a storage line from which they are fed batchwise to a pneumatic lift feed chamber from which they are pumped to the surface.)

4,141,721

METHOD AND APPARATUS FOR COMPLEX CONTINUOUS PROCESSING OF POLYMETALLIC RAW MATERIALS

Jury F. Frolov and 15 other USSR inventors.

(A continuous smelting process for complex zinc sulfide/oxide ores is claimed. This involves reduction of the smelt by a plasma at 4,000 to 5,000 C, with the bath surface maintained at 1,500 to 1,600 C!)

4,141,804

PROCESS FOR ELECTROWINNING METAL FROM METAL BEARING SOLUTIONS

Michael M. Avedesian and Anthony P. Holko, assignors to Noranda Mines Limited, Toronto, Canada.

(This describes the apparatus and electrolyte handling system for fluidized bed electrowinning of metal values from leach solutions.)

4,142,952

COPPER EXTRACTION WITH SALICYLALDOXIME-P-NONYLPHENOL MIXTURES

Raymond F. Dalton, assignor to Imperial Chemical Industries Limited, London, England.

RECENT PATENTS (Continued)

(A process is claimed for extracting and electrowinning copper using this class of lixiviant.)

4,143,865

FLASH SMELTING FURNACE

Thomas N. Antonioni, assignor to The International Nickel Company, Inc., New York, New York.

(A method for cooling a critical area--matte-slag interface--of refractories in the Inco flash smelting furnace is claimed.)

4,144,055

METHOD OF PRODUCING BLISTER COPPER

Stig A. Petersson and Bengt S. Eriksson, assignors to Boliden Aktiebolag, Stockholm, Sweden.

(The Boliden TBRC process for blister copper production is described and claimed. This includes operations of the TBRC, slag-treating, and converter units.)

4,144,056

PROCESS FOR RECOVERING NICKEL, COBALT AND MANGANESE FROM THEIR OXIDE AND SILICATE ORES

Paul R. Kruesi, assignor to Cato Research Corporation, Wheatridge, Colorado.

(Heating oxidic or silicate ores of manganese, nickel, and cobalt in the absence of air, with ferric chloride and other chloride salts to suppress volatilization, converts the desired metal values to soluble chlorides.)

4,144,310

HIGH SLURRY DENSITY SULFIDIC MINERAL LEACHING USING NITROGEN DIOXIDE

Theodore C. Frankiewicz and Robert E. Lueders, assignors to Kennecott Copper Corporation, New York, New York.

(Complex sulfidic base metal concentrates in a highly loaded slurry form are oxidized to return at least one leachable metal value. Nitric oxide is the oxidant, and is restored to its higher valence form by the presence of oxygen.)

4,145,187

TREATMENT OF MATERIAL WITH HYDROGEN CHLORIDE

Raymond E. Oliver and George McGuire, assignors to Matthey Rustenburg Refiners (Pty.) Ltd., Johannesburg, South Africa.

(A hydrogen-chlorine burning reactor is claimed for the conversion and separation of silver from silver-containing base metal concentrators. The reacted product is leached to remove soluble base metal chlorides.)

RECENT PATENTS (Continued)

4,145,212

PROCESS FOR RECOVERING SILVER AND OPTIONALLY GOLD FROM A SOLID STARTING MATERIAL CONTAINING SAID METALS

Fernand J. J. Bodson, assignor to Societe des Mines et Fonderies de Zinc de la Vielle Montagne, Angleur, Belgium.

(Team silver and gold resources, at least partly sulfidic, when oxidation leached in an aqueous thiourea solution, are converted to soluble complexes which can be cemented as a 90+ percent precious metal product.)

4,145,398

BAUXITE DIGESTION BY CAUSTIC ALKALI WITH IMPROVED HEAT TRANSFER IN TUBULAR REACTORS

Ludolf Plass, assignor to Vereinigte Aluminum-Werke A. G., Bonn, Federal Republic of Germany.

(This claims and describes the VAW energy-conserving tubular reactor for bauxite treatment--see Technology and Trends item 3.2.9/1.)

4,146,389

THERMAL REDUCTION PROCESS OF ALUMINUM

Bela Karlovitz, assignor to Bela Karlovitz and Bernard Lewis, both of Pittsburgh, Pennsylvania.

(This appears to be a plasma reactor/condenser system for the suspended-particle reduction of metal oxides.)

4,146,572

SIMULTANEOUS EXTRACTION OF METAL VALUES OTHER THAN COPPER FROM MANGANESE NODULES

Paul H. Cardwell and William S. Kane, assignors to Deepsea Ventures, Inc., Gloucester Point, Virginia.

(A variant of U.S.P. 4137291. A reduction roast converts the copper values to the cuprous state whereby they are not removed by ammonium complexing, and thus can be separated from other metal values in the first-stage leach.)

4,146,573

RED MUD TREATMENT

James Kane, assignor to Nalco Chemical Company, Oak Brook, Illinois.

(Less than 0.02 percent of an organic ammonium salt mixed with Bayer process red mud waste will "solidify" the red mud.)

4,147,390

NODULE DREDGING APPARATUS AND PROCESS

Jacques Deliege, Michel Giot, Vsevolod Obolensky, and Marc Lejeune, assignors to Union Miniere S.A., Brussels, Belgium.

RECENT PATENTS (Continued)

(A sea nodule dredging apparatus is claimed.)

4,147,614

AQUEOUS MIXTURE OF DIESEL OIL, PINE OIL AND DIAMINE FOR CONDITIONING OF CRUSHED MAGNESITE ORE IN MAGNETIC BENEFICIATION PROCESS

Theodor Gambopoulos and Antonios Frangiskos of Athens, Greece.

(Conditioning of crushed magnesite ore in an aqueous mixture of diesel oil, pine oil, and a diamine allows the occlusion of particulate ferromagnetic material on gangue material, but not on the magnesite mineral. Gangue may be magnetically separated, leaving a high-quality magnesite concentrate.)

4,147,644

COLLECTOR COMBINATION FOR NON-SULFIDE ORES

Samuel S. Wang, Eugene L. Smith, Jr., and Ernie F. Huliganga, assignors to American Cyanamid Company, Stamford, Connecticut.

(Nonsulfide ores may be treated with a fatty acid and an anionic perfluoroalkyl compound as claimed to permit concentration by flotation.)

4,148,630

DIRECT PRODUCTION OF COPPER METAL

Charles Arentzen, assignor to The Anaconda Company, Denver, Colorado.

(Claims a method for pneumatically injecting a slurry of sulfidic copper into a molten bath in a process for continuous, autogenous smelting and reduction to produce metallic copper.)

4,148,632

TREATMENT OF DISSOLVED BASIC NICKEL CARBONATE TO OBTAIN NICKEL

Willie Seibt and Donald R. Weir, assignors to Sherritt Gordon Mines Limited, Toronto, Canada.

(This appears to be a variant of the Sherritt process for nickel in which secondary nickel values are recovered from the unleached residue.)

4,148,813

NICKEL AND COBALT SOLVENT EXTRACTION WITH MERCAPTIDES

Alkis S. Rappas and J. Paul Pemsler, assignors to Kennecott Copper Corporation, New York, New York.

(Claims mercaptides as the complexing agent in solvent extraction of nickel and/or cobalt from hydrometallurgical leach liquors.)

4,148,815

AMINO-THIOL NICKEL AND COBALT SOLVENT EXTRACTION

Alkis S. Rappas and J. Paul Pemsler, assignors to Kennecott Copper Corporation, New York, New York.

RECENT PATENTS (Continued)

(A variant of 4,148,813.)

4,148,816

ALKALI METAL MERCAPTIDE SOLVENT EXTRACTION OF COBALT, NICKEL

Alkis S. Rappas and J. Paul Pemsler, assignors to Kennecott Copper Corporation, New York, New York.

(A variant of 4,148,813.)

4,148,862

HYDROMETALLURGICAL TREATMENT OF SOLUBLE SILICATE-BEARING ZINC MATERIALS

Sigmund P. Fugleberg and Jaakko T. I. Poijärvi, assignors to Outokumpu Oy, Outokumpu, Finland.

(Physical characteristics of the silicate burden in siliceous zinc ores are controlled by the rate of addition of silicates during sulfuric acid leaching to promote good settleability and filtration properties.)

4,149,880

RECOVERY OF COPPER FROM ARSENIC CONTAINING METALLURGICAL WASTE MATERIALS

John D. Prater and Barry A. Wells, assignors to Kennecott Copper Corporation, New York, New York.

(This claims a method for processing toxic flue dusts and or refinery sludges from copper smelters. Oxidative pressure leaching with sulfuric acid is conducted under conditions to solubilize copper and some arsenic, with most of the other toxicants and precious metals remaining as solids. Downstream cementation with iron recovers cement copper without the liberation of arsine. The iron-laden raffinate containing arsenic is recycled.)

4,149,947

PRODUCTION OF METALLIC LEAD

John C. Stauter and William K. Tolley, assignors to UOP, Inc., Des Plaines, Illinois.

(In a variant of USP 4124461 for converting crystallized lead halide first to carbonate, then to lead fluosilicate from which it is electrowon, the carbonate conversion may be accomplished by treating the halide with CO₂.)

4,150,091

MANGANESE ORE LEACHING PROCESS

Henry J. Peterson, assignor to Sun Ocean Ventures, Inc., Radnor, Pennsylvania.

(This claims an improved HCl leaching process for treating sea nodules. By regulating pH and chlorine pressure, the desired base metals may be converted to soluble chlorides, leaving insoluble manganese.)

FOREIGN TECHNOLOGY

(The articles listed are from foreign language journals with titles and abstracts printed in English.)

Vogg, H., H. Braun, and A. Lubecki, Ore Prospecting, Exploring, and Processing Techniques by Use of Radionuclides, Erzmetall, 32(1):8-12, January, 1979. (German) The application of radionuclide excited X-ray fluorescence, and neutron activation analysis using neutron isotope sources, with a view to in situ borehole analysis and on-line analysis of slurry systems is reviewed.

Wöbking, H., and H. Wörz, Theoretical Bases and Practical Impacts of the PCR Process in the Copper Refinery Electrolysis, Erzmetall, 32(2):53-57, February, 1979. (German) The operational results of the Brixxlegg-developed PCR process in relation to anodic passivation is reported.

Zanardi, G. M., Exploitation and Extraction of Copper Oxide Ores by Hydrometallurgical Methods in Cerro Verde, Peru, Erzmetall, 32(2):58-63, February, 1979. (German) Mining and processing of the copper oxide ore capping the sulfide ore body includes crushing, heap leaching, ion exchange, and electrolysis. The geology, mineralogy, and costs are also discussed.

Galitis, N., M. Clement, and R. Bertram, Investigations for the Leaching of Sulfidic Copper Ores with Agitation Reactors, Erzmetall, 32(2):74-77, February, 1979. (German) The influences of leachant/electrolyte concentration, the solid electrolyte rate, and temperature on graphite, titanium/platinum, and lead anodes and copper cathodes during simultaneous pocked bed reactor leaching and electrowinning of Mitterberg chalcopyrite and Rammelsberg copper concentrate are discussed.

Trytko, O., V. Masek, Emissions and Immissions of SO₂ in Metallurgical Plant, Hutnicke Listy, (2):127-130, February, 1979. (Czechoslovakian) This presents a study of workplace and ambient SO₂ levels associated with a "metallurgical" plant. For each ton of material "rolled or forged", 22 Kg SO₂ was emitted (apparently from the power plant. The abstract does not otherwise describe the "metallurgical plant").

Solozhenkin, P. M., and R. D. Kupeyeva, Investigating the Sorption Layer on Galena and Sphalerite and of Surface Conditions for Xanthogenate Desorption, Izvestiya Vuz Tsvetnaya Metallurgiya, (6):9-14, 1978. (Russian, title only)

Alkatsev, M. I., Production of Standard Conditioning Metal Deposits by Cementation, Izvestiya Vuz Tsvetnaya Metallurgiya, (6):21-27, 1978. (Russian, title only)

Kutvitsky, V. A., V. V. Mechev, and V. Yu. Taskin, Copper Refining from Nickel by Directed Crystallization in the Magnetic Field, Izvestiya Vuz Tsvetnaya Metallurgiya, (6):27-31, 1978. (Russian, title only)

Zapuskalova, N. A., and E. V. Margulis, The Influence of Separate Parameters on Hydrolytic Precipitation of Iron, Copper, and Zinc from Sulfate Solutions, Izvestiya Vuz Tsvetnaya Metallurgiya, (6):31-37, 1978. (Russian, title only)

Slutsky, I. Z., V. P. Kiseiyov, E. S. Yakubovski, and M. G. Tsyppkin, Pilot Plant Test of Electrolytic Refining the Aluminum-Silicon Alloy Resulting in Aluminum Production, Izvestiya Vuz Tsvetnaya Metallurgiya, (6):59-64, 1978. (Russian, title only)

Markov, Yu. F., V. A. Ivanov, and A. F. Gavrilenko, On Controlling Nickel Precipitation in the Flow Sheet of the Autoclave Treatment of the Nickel Magnetic Pyrite Concentrates, Izvestiya Vuz Tsvetnaya Metallurgiya, (6):115-121, 1978. (Russian, title only)

Sergiyevskaya, Ye. M., and I. A. Vorobyova, Thermodynamics and Kinetics of Nickel Oxide Dissolving in the Sulfuric Acid Solution, Izvestiya Vuz Tsvetnaya Metallurgiya, (6):126-129, 1978. (Russian, title only)

Kravets, M. V., and E. V. Margulis, Effect of Lead Compounds on the Indium Behavior in Zinc Sulfate Solutions, Izvestiya Vuz Tsvetnaya Metallurgiya, (6):129-131, 1978. (Russian, title only)

Novozhenov, V. M., T. I. Tataurova, and S. I. Kuznetsov, To the Question of Reducing the Chemical Losses of Alkali and Aluminum Oxide at Alumina Production by Bayer Method,, Izvestiya Vuz Tsvetnaya Metallurgiya, (1):40-45, 1979. (Russian, title only)

Volkov, V. V., and N. I. Eremin, The Thickening Rate Calculation of Nephelite Slime in Aluminate Solution, Izvestiya Vuz Tsvetnaya Metallurgiya, (1):45-48, 1979. (Russian, title only)

Polyakov, P. V., V. M. Mozhayev, V. V. Burnakin, V. A. Kryukovsky, and V. E. Nikolayenko, On Bubble Liberation at Cryolite-Alumina Bath Electrolysis, Izvestiya Vuz Tsvetnaya Metallurgiya, (1):55-61, 1979. (Russian, title only)

Leonova, M. L., B. L. Egorov, M. D. Ivanovsky, and M. A. Meretukov, Effect of Heat Treatment on Gold Extraction from Grinding Powders, Izvestiya Vuz Tsvetnaya Metallurgiya, (1):75-78, 1979. (Russian, title only)

Ivanov, V. A., L. B. Berliner, V. I. Kesisoglu, M. P. Shapirovsky, and E. I. Sterenberg, The Mathematical Description of Basic Regularities of Copper-Nickel Concentrate Sinter Roasting, Izvestiya Vuz Tsvetnaya Metallurgiya, (1):128-134, 1979. (Russian, title only)

Mukhina, T. M., and N. V. Krupkin, The Analysis of the Actual and Planned Efficiency of Introducing New Equipment into Non-ferrous Metallurgy, Izvestiya Vuz Tsvetnaya Metallurgiya, (1):134-139, 1979. (Russian, title only)

Molnar, I., Some Aspects of the Selection of the Technical Solution for a New Aluminum Electrolysis Plant, Banyaszati Es Kohazati Lapok-Kohaszat, 112(1):33-37, January, 1979. (Hungarian) An analysis of the energy and environmental protection aspects in the design of an aluminum electrolysis plant is presented.

Zambo, J., Technical Development in the Hungarian Alumina Refineries, Banyaszati Es Kohazati Lapok-Kohaszat, 112(1):43, January, 1979. (Hungarian) Energy economics and utilization of red mud will result in a major change in equipment for new Bayer plant and equipment design.

Paschen, P., Metal Refining by Selective Evaporation, Metall, 33(2):137-140, February, 1979. (German, title only)

Stepniak, B. D., Copper Extraction by 2-Hydroxy-5-Nonylobenzoic Aldoxime, Rudy Metale, 24(1):2-5, January, 1979. (Polish) A higher degree of copper extraction in a shorter period of time is obtained by the use of 2-hydroxy-5-nonylobenzoic oxime rather than 2-hydroxy-5-nonylobenzophenomic oxime.

Kuczumow, A., M. Majdan, and S. Radzki, The Role of Rare Earth Elements in Chemistry and Technology, Part 1. Resources, Winning and Distribution, Rudy Metale, 24(1):24-28, January, 1979. (Polish) Methods of isolating rare-earth elements by classical and fractional methods applied in industrial processes are reviewed. Local resources of rare-earth elements are also discussed.

Kubicki, J., and S. Kuczkowska, The Effect of Ammonium Sulfate on Intensification of the Copper Electrorefining Process, Rudy Metale, 24(2):56-60, February, 1979. (Polish) Ammonium sulfate added to copper electrolytes containing nickel improves the electrochemical refining process.

Krupkova, D., Cementation Purification of Industrial Zinc Sulfate Solutions, Rudy Metale, 24(2):65-75, February, 1979. (Polish) This discusses cementation purification of zinc electrolytes using zinc dust. In the first stage, cobalt, nickel, and other impurities are eliminated at 90 C with high lead-antimony zinc dust, while small quantities of cadmium are eliminated at 60 C in the second stage with "columnar" zinc dust.

Wanielista, K., Algorithm to Determine Criteria for Recoverability of Ore Deposits, Rudy Metale, 24(2):76-82, February, 1979. (Polish) A procedure is proposed to determine the minimum ore content of a deposit necessary for industrial utilization. Profitability is emphasized.

Subject: Conditioning Effects of Flotation of Finely Divided
Non-Sulphide Copper Ore

The effect of several physical and chemical parameters on the grade and percent of copper recovery from nonsulfide ores during the flotation process are described in this paper.

The ore--a siliceous, malachite-prominent ore from Zaire--was ground to 60 percent-9 μ . The following parameters were varied during the experiments: (1) stirring speed, (2) stirring time, (3) quantity of collector (K-amyl xanthate), (4) quantity of sodium sulfide, (5) feed grade of copper, (6) pulp density, and (7) the quantity of calcite, dolomite, or both, added. The results are shown graphically.

The results showed that flotation was successful only when the fines were conditioned at stirring speeds above 900 rev/min. A minimum satisfactory stirring time of 40 minutes at 2060 rev/min was defined. A stirring time of 60 minutes at 2060 rev/min was chosen to be used in most experiments. The minimum quantity of K-amyl xanthate necessary to obtain efficient separation was found to be 3.75 kg/t. The addition of sodium sulfide was found to be detrimental to the flotation process. As the feed grade of copper decreased from 8.5 to 0.4 percent, only a 10 percent decrease in recovery and a 12 percent reduction in grade was noted. However, there was a significant decrease in both grade and recovery when pulp density was increased to about 10 percent in weight. No effect on grade or recovery was found when calcite or dolomite made up 0-25 percent of the gangue.

Reference: Rubio, J., Institution of Mining and Metallurgy, Section C, 87:C284-287, December, 1978.

Subject: Metal and Sulfur Recovery from the Copper Sulphide
Precipitate in the Sherritt Gordon Process

The Sherritt Gordon pressurized ammonia leach process for the extraction and production of nickel and cobalt is employed at the Western Mining Corporation's Kwinana plant in western Australia. A by-product of this operation is about 2,600 annual tonnes of copper sulfide precipitate produced in the "copper boil" stage of the S.G. process. This precipitate carries with it some entrained nickel and cobalt values, and also significant quantities of precious metals (Pd, Ag, Au, Pt). In present practice, the Cu_xS product is washed, filtered, dried, and toll-smelted and refined by the Electrolytic Refining and Smelting Co. Pty Ltd, N.S.W., Australia. However, the freshly precipitated and repulped Cu_xS product is a very reactive, finely divided slurry that is ideally suited to further hydroprocessing. The economics of further processing will depend on the scale of operations, and have not yet been determined.

This technical article describes laboratory studies of the further hydrometallurgy of the copper boil product to recover metallic copper, return nickel and cobalt values to the main S.G. process circuit, and to concentrate precious metal values for refining. The preferred hydroprocess involves a low-temperature (<40 C), moderate-pressure (30 psig) oxidizing $\text{NH}_3/(\text{NH}_4)_2\text{SO}_4$ leach, which releases sulfur in elemental form. Precious metals precipitate as sulfides when the liquor is stripped of NH_3 . SO_2 treatment and heating of the stripped liquor crystallizes copper as Chevreul's salt, and nickel and cobalt values (still in solution) are returned to the S.G. circuit. Acetonitrile treatment under nitrogen at 55 C yields cuprous sulfate, which is steam disproportionated to yield copper powder of high purity. Various alternatives or options were also investigated, and are discussed. Elements of the process were individually examined, and a 500 gm demonstration was conducted. A flow sheet and material balance are presented in the article.

Reference: Filmer, A. O., A. J. Parker, M. Ruane, and L. G. B. Wadley,
Proceedings Australasian Institute of Mining and Metallurgy,
(268):39-46, December, 1978.

Subject: Studies on Segregation of Roasted Copper Concentrates:
Separate Na and Cl Additions and Effects of Sulfate

The segregation process for winning metallic copper from dead-roasted concentrates involves halide-activated reduction by carbon, with CO_2 and copper as the ultimate reaction products. This paper reports expanded laboratory studies of the role of halide source, reductant, and influence of residual sulfur (as sulfate) on the rates and extent of reaction at about 700 C.

Sodium chloride is the usual halide source. When cuprous chloride is used as the halide source, reaction rate is improved, but with petroleum coke as the reductant, a continuous layer of segregated copper blocks further reduction when only about half the copper value has been recovered. The coaddition of CuCl and active carbon (e.g., coconut charcoal) results in essentially complete reaction at suitably high rates.

Coadditions of CuCl and Na_2CO_3 with mol ratios of Na:Cl varying from 0.5 to 2 were studied. Excess sodium slowed the reaction, and excess chloride blocked it. At a molar ratio of 1, reaction proceeded as if NaCl had been the additive--moderate rate; complete reaction.

Experiments with calcine containing a higher sulfate content (as $\text{CuO} \cdot \text{CuSO}_4$?) suggested that the activity of petroleum coke is enhanced by a moderate amount of residual sulfate in the calcine. The presence of sulfate increases the reaction rate. However, the presence of halide is still necessary to allow reduction to go to completion.

(The dead-roasting/segregation process offers potential for abating the economic impact of SO_2 removal from conventional copper smelting. These reported experimental results may help to define the potential commercial operation of the segregation process.)

Reference: Marcuson, S. W., Inst. of Min. and Met. Trans. C., 87:(26)-C265, December, 1978.

Subject: Utah Copper and the \$280 Million Investment in Clean Air

Kennecott's essentially new, 270,000 st/yr, \$280-million copper smelter at Garfield, Utah, has been the subject of prior Awareness reports (see T&T 1.3.2/1, 1.3.2/15, selected news articles). This editorial feature from E&MJ presents an excellent overall summary of this new, two-stage, Noranda-process-based facility.

Two, 1,650 stpd (dry concentrate) reactors and three converters (excluding a spare reactor and a spare converter) form the working heart of the system. However, these are dwarfed by the dimensions of the gas handling and processing system, which require in excess of 33,000 horsepower to move the gas through various stations, finally exhausting at a rate of 1.3 million scfm through the new, 1,200-foot stack.

A lucid flowsheet and schematic layout are included in the article. The presentation centers about reasons for the original selection of the new system, reactor and converter processes, slag reprocessing, gas handling (including the four single-contact acid plants, scrubbers, mixers, cyclones, etc.), process control and data management.

This large investment by Kennecott was directed toward achieving the then-mandated air regs (which EPA subsequently intensified, much to Kennecott's chagrin). The smelter now captures 86 percent of the input sulfur, compared with a 55 percent prior capture.

Production capacity is about the same as previously. Despite considerable savings in direct process energy consumption, net energy savings are essentially offset by energy demands for pollution control and essential ancillary process demands.

Reference: Dayton, S., Engineering and Mining Journal, 180(4):72-83,
April, 1979.

Subject: Aqueous Oxidation of Chalcopyrite in Hydrochloric Acid

Process kinetics for the oxidation leaching of chalcopyrite by hydrochloric acid are explored in the laboratory, as described in this paper. The effects of oxygen pressure (to >2000 k Pa), temperature (90-130 C), agitation, and acid strength (CuFeS₂:HCl molar ratios from about 0.1-2) were explored. Optimum conditions were defined to be 110 C, 2010 k Pa of O₂, highly agitated, and molar ratio of chalcopyrite/HCl of 0.8-0.9.

The reaction rates are faster than with the similar H₂SO₄ process. Under optimum conditions, copper recovery was found to be only about 70 percent, but the pregnant liquor was very low in iron, which hydrolyzed and precipitated as Fe₂O₃ or FeOOH. Increased acid strengths resulted in the generation of copper ferrite or contamination of the leachate with iron. The sulfur reports as free sulfur, which at temperatures greater than 110 C was held responsible for decreased reaction rates due to coating of the particles with molten sulfur.

(This report of academic investigation was not concerned with economic or corrosion problems.)

Reference: Habashi, F., and T. Toor, Metallurgical Transactions, 10B(1):49-56, March, 1979.

Subject: Afton's Copper Smelter Proves Economic at 27,000 Tons

Yearly

The Afton copper smelter at Kamloops, British Columbia is the first commercial use of the Top Blown Rotary Converter (TBRC) Process for the combined smelting and converting of concentrates. (See the "In the News Section" of the September, 1977 issue of the Technical Awareness Bulletin). In 1979, 27,000 tons of 99.6 percent copper product is expected to be shipped from this \$25 million smelter.

The ore, native copper, bornite (Cu_5FeS_4), chalcocite (Cu_2S), and chalcopyrite (CuFeS_2) is crushed and processed to produce two distinct types of copper concentrate: a high grade metallic concentrate containing 80-90 percent copper, and a conventional flotation concentrate of 55-60 percent copper with 35 percent sulfur. The conventional concentrate is mixed with iron and limestone, then fed into the converter where it is melted by an oxy-fuel flame. This is later replaced by oxygen blowing into the molten bath; however, the major quantity of oxygen is supplied above the slag, in a highly exothermic reaction attaining 1350 C. Tilted at 15-20 degrees, the furnace rotates up to 40 rpm to protect the refractories from the intense heat. After pouring of the first slag, smelting resumes, then the slag is again removed. Then the native copper concentrate is charged into the converter and the procedure repeated, blowing the charge to blister copper (99.6 percent Cu).

All exhaust gas is collected and processed in a four-stage dual-alkali scrubbing system. The first stage reduces the temperature by waterspray to 340 C and removes particulates in an electrostatic precipitator; the second stage contacts the gas with sodium sulfite solution to absorb the sulfur dioxide; the third cools the gas to condense any mercury present; and the fourth stage includes the use of a proprietary filter. The sulfur control system also includes sulfur dioxide storage so that a continuous flow is available for a sulfuric acid plant.

(This news article may not be wholly consistent in a technical sense; several questions arise. However, it presents a useful description of the overall operation at Kamloops.)

Reference: Canadian Chemical Processing, 63(2):22-24, March 21, 1979.

Subject: Some Aspects of Ion Exchange in Copper Hydrometallurgy

Ion exchange is not at present a leading copper hydrometallurgy process but is an attractive alternate to the solvent extraction process, as solvent extraction is usually limited to liquors containing more than 1 g/l of copper. The purpose of this reported work was to compare the copper hydrometallurgical properties of (1) chelating resins, 8-hydroxyquinoline, salicylaldoxime, and XF-4196; (2) as-received impregnated foams, Kelex 100 (33 percent) and LIX 65N (23.5 percent); and (3) amberlite beads impregnated with Kelex 100 (33 percent) and LIX 65N (33 percent).

The ion exchange systems were examined for their high capacity for the metals of interest, high selectivity, and rapid equilibration. Total capacity was reported as copper uptake at pH 2 from a dilute sulfuric acid medium or at pH 4 from a sodium acetate/acetic acid buffered solution. Selectivity ratios were reported as the respective weights of copper and iron from solutions which were equimolar in copper (II) and iron (III). Kinetics or rates of equilibrium were time values ($t_{1/2}$) taken for the metal to occupy one-half of the available sites on the exchanges.

Results of the work show that only chelating resins are viable; impregnates have too low a capacity and high bulk volumes; impregnated beads are too high in cost. However, the impregnates were far superior to the chelating resins in selectivity. Solution to the problem appears to be an attempt to improve the selectivity of the XF-4196 (See this issue "Technology and Trends" 1.3.9, page 10).

It seems unlikely that ion exchange will replace solvent extraction in copper winning. However, ion exchange could complete the removal of copper from the solvent extraction raffinate and ensure complete copper removal from discharge liquors where excess iron must be bled from the system. A flow sheet of a process incorporating this idea is included in the paper.

Reference: Vernon, F., Hydrometallurgy, 4(2):147-157, 1979.

Subject: Copper-Selective Ion-Exchange Resin with Improved Iron
Rejection

A description of a new chelating ion-exchange resin with higher copper leaching efficiency and lower affinity for iron is given in this paper.

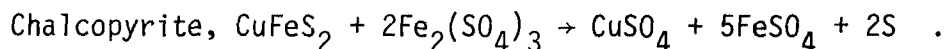
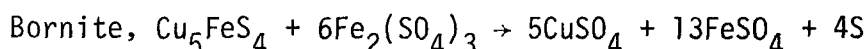
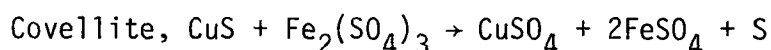
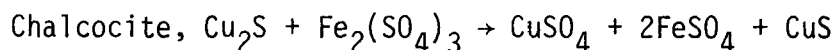
The resin, XFS 43084, (N-(2-Hydroxypropyl)-pecolyamine) has a several fold increase in copper/iron (III) selectivity and retains the copper absorption characteristics of the original XFS 4196 resin. Elution of copper can be done with 2N sulfuric acid at 15 times the rate of absorption. Aqueous ammonia can also be used to elute the copper. Equilibrium data for metal-ion absorption were determined, as well as the kinetics of metal exchange. These data are presented in eight graphs, together with a table comparing the properties of XFS 43084 with those of XFS 4196, the original ion-exchange resin.

Reference: Grinstead, R. R., Journal of Metals, 31(3):13-16, March, 1979.

Subject: A New Hydrometallurgical Method for the Processing of
Copper Concentrates Using Ferric Sulfate

This paper describes laboratory tests and a 3-week pilot-plant operation of a process for electrolysis of copper from leaching solutions in diaphragm cells, with double-stage anodic recovery of the spent leaching agent.

Leaching is carried out at 90-95 C with copper sulfides decomposed as follows:



The extraction of copper reaches 96 percent in 100-120 minutes except for chalcopyrite. After 4-6 hours, the yield of copper into solution may reach 97-98 percent. About 70-74 percent of the product is treated to crystallize $\text{FeSO}_4 \cdot \text{H}_2\text{O}$ by heating the solution to 170-175 C. The pregnant liquor is then delivered to the cathode compartment of the electrolyser where copper is won in one or two stages. The spent catholyte is combined with a portion of previously crystallized $\text{FeSO}_4 \cdot \text{H}_2\text{O}$ and delivered to the anode compartments where the initial anodic oxidation of FeSO_4 to $\text{Fe}_2(\text{SO}_4)_3$ takes place. This ferric sulfate is utilized in the second leaching stage. The spent leaching agent in the second leaching stage is reoxidized anodically before the first leaching stage.

The essential feature of the process is the double anodic recovery of the leaching agent, decreasing the quantity of ferric ions necessary to leach a given quantity of copper to about 50 percent of the required stoichiometric ferric ions to leach the same amount of copper.

Electrolysis of the copper solutions, containing a high concentration of ferrous sulfate after leaching the concentrate, is made possible by the application of diaphragms. These diaphragms (0.5 mm thick PVC) restrict mixing between iron-depleted catholyte and recharged anolyte.

Silver and lead occurring with the copper are not leached by the ferric sulfate solutions, but require leaching with concentrated brine solution in an additional leaching process.

Reference: Letowski, F., B. Kolodziej, M. Czernecki, A. Jedrczak, and Z. Adamski, Hydrometallurgy, 4(2):169-184, 1979.

Subject: A Hydro-Saline (Chloride-Ion) Cycle for Copper-Bearing
Waste Leaching

This paper describes the laboratory experiments conducted to determine the characteristics of KCl and NaCl solutions in the leaching of low-grade, copper-bearing wastes and the subsequent recovery of the copper by cementation onto aluminum and aluminum-alloy scrap and beverage cans.

KCl and NaCl were chosen for the tests because of their natural abundance. Sea water, geothermal brines, brackish-water in underground saline reservoirs, and in some ground-water in the U.S. southwest, all could be utilized in leaching the low-grade copper-bearing waste.

After leaching, the copper solution is recovered by cementation on a more electropositive metal, which typically has been iron. Recently, aluminum has been suggested for this use as the copper recovery is more efficient with aluminum (although a higher temperature is needed). At least 75 ppm Cl^- in solution is necessary to break down the protective aluminum surface oxide film. The efficiency of cementation of copper on aluminum doubles between 30 and 40 C. The efficiency of copper cementation on iron is less than half that of aluminum. Cementation on aluminum is ideally suited for dilute solutions, the cement copper is purer than that cemented by iron, and the cement is physically easier to remove from aluminum.

The proposed hydro-saline leach cycle and cementation on aluminum may not be generally economical at the present time; however, in the next decade with abundant aluminum scrap and the price difference between iron and aluminum decreasing, this process will become more attractive and profitable, in the authors' opinion. (This is not this reviewer's opinion.)

Reference: Murr, L. E., V. Annamalai, and P-C. Hsu, Journal of Metals, 31(2):26-32, February, 1979.

Subject: Noranda Reprocesses Slags to Enhance Recovery

In 1973, Noranda Mines Limited, at its Quebec copper smelter, began slag-milling.

The slag at Noranda is cooled slowly to promote agglomeration of metal values to promote agglomeration of metal values to permit easier recovery. The slag is then crushed to -325 mesh and floated with sodium amyl xanthate and Dow 1263. A concentrate containing 35-40 percent copper is produced. Only 0.28 percent copper reports to the tailings.

At Inco, a similar procedure of flotation of slags produces concentrates containing 15-50 percent copper, with only about 0.25 percent copper lost in the tails.

Reference: Canadian Chemical Processing, 63(2):24, March 21, 1979.

Subject: Recent Advances in Copper Electrowinning

Interest in hydrometallurgical processes is increasing, with nearly 15 percent of copper production at the present time by this method. This may increase to about 40 percent by 1985. This paper reviews the advances in copper electrowinning in detail. The first section is devoted to methods used to permit copper to be electrowon at high current densities, with the following section detailing the consequences of this operation on the various components of the system. The design and development of electrochemical cells is considered, with effects of impurities, the role of additives, and the morphology and growth of copper deposits discussed. A bibliography of 125 articles is included.

The depletion of copper sulfide ores is necessitating the use of lower grade or mixed oxide-sulfide ores, and material from old tailings dumps. Solvent extractants, such as LIX and Kelex, which have been recently developed, are a significant aid in processing these ores by hydrometallurgy. Impurity of product, buildup of iron in the dump, acid consumption, and increased cost of scrap iron all have led to a preference for the solvent extraction/electrowinning route for copper recovery. However, all processes have problems associated with them and the solvent extraction/electrowinning process is no exception. These problems are (1) organic burn of the cathodes, (2) acid misting in the tankhouse, and (3) alternative anode materials which can resist high acid concentration and methods to overcome the inclusion of lead in the cathode copper.

Further study is needed in the interaction of the solvent extraction stage with the electrowinning operation, the effects of organics and flotation agents in the electrowinning stage, and the feasibility of the use of diaphragm cells, cyclic voltammetry, and scanning electron microscopy.

Reference: MacKinnon, D. J., and V. I. Lakshmanan, Canada Center for Mineral and Energy Technology, Canmet Report 76-10, Jan. 1976.

Subject: Hydromet May Free Small Lead Districts From The Smelter

The chloride-electrolysis process of the U.S. Bureau of Mines (see Technology & Trends Index 2.3.2, page 1) could permit small mining districts to recover the lead from their own concentrates, saving millions of dollars each year.

At the present time, small mining districts must ship their concentrates to smelters, often at long distances, thus requiring shipping charges in addition to the smelter's fees. In this article, the author analyzes the economic feasibility of the installation of the new Bureau of Mines' hydromet process in two lead districts of Mexico.

The U.S. Bureau of Mines' Chloride-Electrolysis Process leaches the sulfide concentrates with a solution of ferric chloride, sodium chloride, and hydrochloric acid which, in 30 minutes, can dissolve 98 percent of the lead and 96 percent of the silver. The residue (50 percent of which is sulfur) is removed by filtration and pure lead chloride crystals are formed from the filtrate. These crystals are fused and electrolyzed at 320 C to produce lead. The leaching solution is regenerated with chlorine gas which is produced in the electrolytic cells. No sulfur dioxide air pollution results from this process, nor are workers exposed to lead fumes.

The two Mexican lead concentrators considered produce about 12,000 and 18,500 stpy, respectively. These would require \$3.4 and \$4.3 million capitalization for the USBM process, of which an assumed 60 percent would be financed (at 10 percent per year!). Over a 14-year assumed life, the estimated operating costs would be \$13.2 and \$18.1 million, respectively. Including ore depletion charges, gross profits accruing to the hydromet operation installation would be \$10.3 and \$12.5 million for the 14-year period. It is concluded that supplanting freight and custom smelting fees with the USBM hydromet process would be a winning proposition for small lead concentrators.

Reference: Rovirosa, N. Y., adapted from "The Hydrothermal Processing of Lead Concentrates in Small Scale Mining", Mining Engineering, 31(2):147-149, February, 1979.

Subject: Cominco Building World's First Zinc Pressure Leaching
Plant of 70,000 TPY Capacity

As part of an 8-year \$425+ million modernization and expansion program, Cominco announced the beginning of construction of a \$23-million, 70,000-ton per year pressure leaching zinc plant at its Trail, B.C., operations. The plant represents the first commercialization of the S-C (Sherritt-Cominco) process for zinc.

In this process, zinc sulfide concentrates are directly leached (without roasting) in oxygenated H_2SO_4 at high temperatures and pressures in a closed system. This removes the sulfur as free sulfur which is extracted and easily stored for later use or marketed. In Cominco's case, the added flexibility in fertilizer production will be a major feature. The new operation will greatly improve industrial hygiene and environmental issues. About one-fourth of the Trail zinc concentrate will be processed in the new facility after the second reactor comes on-stream in 1983. The balance of concentrates will continue to be processed by conventional roast-leach processing. (The modern fluidized bed roasters were installed in 1971. The leaching plant is being modernized as part of the current program.)

The pressure-leached product will join the main flow in the conventional leaching plant for purification and refining.

Reference: The Northern Miner, 65(6):A14, April 19, 1979.
American Metal Market, 87(78):9, April 20, 1979.

Subject: Sulfation Roast Boosts Recovery from Lean Concentrates

Ore reserves in the New Brunswick area of northeastern Canada total 400 million tons, grading 13 percent lead, zinc, and copper, yet only two operations are producing zinc concentrates. The reason more mining development has not taken place in this area is the startling losses which occur during the processing of the ore; as much as 35-45 percent of the copper, 30 percent of the lead, 13-20 percent of the zinc, and 35-50 percent of the silver.

Three years' work was recently completed to find and assess processes to better treat these low-grade zinc concentrates. Two processes were developed, one by Sherritt Gordon (a zinc pressure leaching technique) and a sulfation roasting process by the Research and Productivity Council (RPC) of New Brunswick.

The process sulfation roasts ore at a relatively low 685 C then leaches the roasted ore in two stages with weak, then strong sulfuric acid. The strong acid filtrate is recycled to the roaster to eliminate the iron sulfate which is the key to the improvement. The RPC process accepts bulk concentrates, eliminates the problem of iron in solution common to zinc smelters. It can be used in existing facilities without extensive modifications and is a low-energy process.

Pilot-plant operation at about a 10-ton per day rate is the next step in testing the RPC process.

(This news article presented no bottom-line data; improved recoveries from the complex, lean ores are merely inferred.)

Reference: Branch, S., Canadian Mining Journal, 100(3):51, 53, March, 1979.

Subject: Jersey Miniere Zinc Opens A New Refinery Capable of
Producing 90,000 tpy of Slabs

The Clarksville, Tennessee, zinc refinery, a joint venture of New Jersey Zinc Company and Union Miniere, was dedicated in mid-March. The \$200-million facility (refinery and mines), largest private industrial investment ever made in Tennessee, will produce 90,000 tpy of zinc slabs and can include 140,000 tpy of acid, 300 tpy cadmium, and 200 tpy copper. The zinc capacity can be doubled to 180,000 tpy if needed. (See "In the News", Technical Awareness Bulletins, 1978.)

The zinc ore (4-4.5 percent zinc) is mined and concentrated at the Elmwood and Gordonsville, Tennessee, mines to 60 percent zinc and 30-35 percent sulfur and shipped to Clarksville for refining. Here it is roasted in a Lurgi fluid-bed roaster at 1650 F to remove the sulfur as sulfur dioxide. This gas, after cleaning in an electrostatic precipitator, moves to the double-contact acid plant where it is converted to 400 tpd of sulfuric acid.

The calcined ore, after grinding, is leached with sulfuric acid to form a zinc sulfate solution. Zinc dust is added in first a cold, and then a hot stage to precipitate contaminants (cadmium, copper, and cobalt). The zinc sulfate solution is then electrolyzed, depositing metallic zinc on the aluminum cathodes. Sheets of zinc, 1/4 in. in thickness, are removed from the cathodes and melted in induction furnaces into 2,400 lb ingots or 50 lb slabs.

The air pollution control system cost \$12.2 million at Clarksville. Particulate matter is trapped in baghouses in the concentrate handling area, the zinc dust production unit, the dress grinding area, and in the cadmium smelting and casting areas. Total efficiency is 99.9 percent. In the acid plant, tailgas sulfur dioxide is less than half the 650 ppm allowed by the EPA. Water, just under 1 million gpd, is mostly cleaned and recycled, but some water is treated and directed to three settling ponds before it is returned to the Cumberland River.

Subject: Flotation Reagent Practice in Primary
and Byproduct Molybdenum Recovery

This article is a review of the reagents used as collectors, frothers, and depressors in the flotation of porphyry copper for molybdenum recovery. A large table details the primary flotation circuit, concentrate retreatment, and metallurgical results of plants in the U.S. (16), Canada (5), Chile (4), Peru (1), and the Communist World (4).

Examination of the table shows no consistent pattern in copper-molybdenum recovery except that better molybdenum recovery generally comes from ore with the lowest copper content. This is probably due to the greater attention paid to optimize molybdenum recovery because of low copper values in the ore.

This review revealed that about 60 percent (19 of the 30 plants) use xanthate as a collector and about the same percent use MIBC as the primary frother; however, the author's analysis of all the available published data showed no statistically significant correlation between the collector or frother used and molybdenite recovery. The reason is probably because copper is the most valuable mineral in the porphyries and the collector and frother are chosen primarily to increase recovery of the copper rather than molybdenum.

Another factor in reagent selection is the depression of copper in the molybdenum circuit. Preconditioning can be thermal or chemical. The thermal treatment is used by a third of the mines and includes roasting or steaming to decompose the collector on the surface of the sulfide mineral and vaporize the frother. The majority use sodium sulfide or hydrosulfide depression of copper sulfides. An oxidation process utilizes hypochlorites, peroxides, permanganates, or dichromates to oxidize copper and pyrite surfaces and to possibly destroy adsorbed collectors. The oxidizers are always combined with some form of cyanide (sodium or potassium ferro or ferri) to provide the desired HCN level with H_2O_2 in the conditioning step and to remove the collector coating on the copper mineral.

(In primary molybdenite mines--e.g., Climax and Henderson--the reagent selection is completely different. In early days, only pine oil was used for collector and frother. Today, a neutral oil called "Vapor Oil" is used for the collector and Artic Syntex L, a sulfated coconut oil, for some froth control. Sodium cyanide and Nokes reagent (P_2S_5 plus caustic) is used to depress gangue minerals.)

Reference: Crozier, R. D., Mining Magazine, 140(2):174-178, February, 1979.

Subject: How to Select a Byproduct Moly Recovery Process

(This article is the second in a series on byproduct molybdenum recovery. The first was abstracted in The Technical Awareness Bulletin Index 5.2.9, page 2.)

Recovery of molybdenite from copper operations, once considered uneconomical, today, with the very strong molybdenum market, it is a different story. With the many byproduct recovery processes available, it is difficult to choose the proper procedure to use for a particular ore. The author reviews the most attractive processes for treating different types of ores during initial startup.

Only established processes should be considered for a new mine, and operations should be kept as simple as possible. After the plant is in operation, more effective procedures can be developed. Because several of the reagents used are poisonous and known pollutants, environmental aspects must be considered. Despite the fact that potential pollutants from moly circuits are nearly always contained, local policy and the fear of public reaction are strong forces. These could be the most important factors in deciding which process to use.

The metallurgical effectiveness of a process is directly influenced by (1) copper mineralization, (2) reagents used in the copper recovery circuit, (3) the mode of occurrence of molybdenite in the ore body, and (4) the presence of natural floaters in the ore. Natural floaters are talc and pyrophyllite, sulfur, graphite, coal and native asphalt, and covellite. These cause unique design and process problems for their suppression.

The author's recommendations for flotation reagents and procedures can be summed up as follows:

Chalcopyrite ores: 1. Sodium hydrosulfide; 2. Arsenic Nokes

Chalcocite ores: 1. Sodium ferrocyanide with (a) acid-peroxide conditioning or (b) steaming or cooking; 2. Phosphate Nokes with (a) acid-peroxide conditioning or (b) steaming or cooking.

Mixed ores: 1. Phosphate Nokes with steaming or cooking pretreatment; 2. arsenic Nokes.

Reference: Shirley, J. F., World Mining, 32(4):44-47, April, 1979.

Subject: Fuming of Stannous Oxide From Slags

An accepted process for the separation of tin from slags is the fuming of stannous sulfide. The recovery of tin is high, 90-95 percent, but the process results in the production of sulfur dioxide. To avoid pollution of the environment by sulfur dioxide, there is interest in fuming of SnO to recover tin from slags. This paper describes the laboratory experimental work on the fuming of tin from slags as stannous oxide (SnO).

Four types of experiments were run: (1) fuming of liquid slags under an atmospheric pressure of CO-CO₂ or CO-Ar mixtures, (2) sparging with CO, (3) sparging with hydrogen, and (4) under reduced pressure. The results showed that, at atmospheric pressure, both with and without gas injection, the fuming of SnO was greater under reducing conditions. The reason for this is not clear unless a volatile hydroxide was being formed.

Whether the operation is commercially attractive depends on the control of oxygen potential and the rate at which SnO is fumed in the absence of reduction or oxidation of the slag. A range of oxygen potential exists within which virtually all tin removal from slag is by the vapor phase; therefore, volatilization of SnO is nearly as important a mechanism of tin removal from slag as tin reduction. Fuming of SnO from slags by purging with a nonreducing gas is not practical. Treatment of slags under a vacuum is concluded to offer an alternative to sulfide fuming, however. Much more study is needed to adequately assess the economic feasibility of such a process if sulfide fuming becomes unacceptable because of SO₂ emissions.

Reference: Roy, T. D., S. R. Chandrashekar, and D. G. C. Robertson,
Institution of Mining and Metallurgy, Section C, 87:C225-C230,
December, 1978.

Subject: "Thar's Gold in Them Thar Wells!"

The greatest potential of the Salton Sea geothermal field in California's Imperial Valley is not the 5000 MW electrical generation capacity, but the 2 ppm of gold the waters contain, according to this article. If 80 percent of the gold were recovered from these waters, the Salton Sea field alone could produce 2340 short tons of gold annually, double the present world's output. Because geothermal resources are exploited by power companies, most are not authorized to extract minerals; consequently, the mineral potential is neglected.

Minerals may be recovered from the sinters and precipitates where they have been concentrated by evaporation, and from the gases, as well as from the waters. The waters offer the lowest cost recovery; however, the mineral content is generally low. Geothermal waters carry arsenic, boron, sulfur, gold, antimony, silver, and base metals, and often high quantities of salt, potash, and manganese. Sintors and precipitates carry high concentrates of gold (85 ppm), antimony, and silver, with some values of mercury, zinc, and copper. Gases generally contain large quantities of carbon dioxide.

There are few data as to the life of the geothermal fields. However, one in Italy is still operating after 70 years, and some in the U.K. were once utilized by the Romans. The author claims, if the minerals are obtained from hot water leaching from host rock, concentration will change little with time. If, on the other hand, the minerals are derived from magma, mineral content depends on length of time those sources are operative.

The potential is great for geothermal areas to become a source of mineral supply, especially gold.

Reference: Barnea, J., Adapted from "Geothermal Minerals - The Neglected Minerals", Mining Engineering, 31(2):146-147, February, 1979.

Subject: Cyanide and Arsenic Removal from Gold Mine Effluent

This brief article previews a forthcoming report relative to a joint industry-government (Canadian) demonstration project to lower cyanide and arsenic levels in gold milling effluents from the Giant Yellowknife Mines, Ltd., operations.

Details of the project were not given; however, at a chlorine-to-cyanide ratio of 5:1 and, with a (pond) retention period of 5 hours' total, cyanide was less than 1 mg/l in the barren solutions. (Cyanate, rather than CO₂ and N₂, is the oxidation product.) A Cl:CN⁻ ratio of 9:1 was required for pond overflow. Iron precipitation reduced arsenic to the 0.2 to 0.5 mg/l level when iron: arsenic ratios of 7:1 to 4:1 were used.

"The 96-hour toxicity tests on treated effluents indicated that non-acutely lethal (??) effluents were possible after dechlorination". The added cost for treatment at Giant Yellowknife would range from \$0.94 to \$6.32/oz of gold, depending on treatment.

(This brief is not particularly rational in itself, but the forthcoming report should be of interest.)

Reference: Water and Pollution Control, 117(2):20, February, 1979.

Subject: Smoky Valley Operations at its Round Mountain Mine
in Nevada

This paper presents a history and description of gold mining at Smoky Valley's Round Mountain Mine near Tonopah, Nevada.

Operating intermittently since 1906, a joint venture of Copper Range, Case Pomeroy & Company, and Felmont Oil Corporation began full-time production mining in January, 1976. In 1979, production is expected to reach 55,000 oz of gold from 1,800,000 tons of ore. The cutoff for ore is 0.02 oz of gold per ton with material below that considered waste. Reserves total 12,000,000 tons of ore averaging 0.062 oz of gold per ton with 0.053 oz recoverable and 0.07 oz of silver per ton. Six years of production is planned at this rate.

The ore, after crushing to minus 2 inches, is piled in five, 50,000-ton heaps on an impervious membrane of ground rubber and asphalt to begin the heap leaching process. A dilute solution of 1 pound of sodium cyanide and 0.9 pound of lime per ton of solution is sprayed at the rate of 400 gallons per minute on each heap for 27 days. This is followed by a 2-day water wash when the residual gold content falls below the economic level. The leaching effluent flows through five activated-charcoal-loaded adsorption tanks at 1,600 gallons per minute where the cyanide complexes are adsorbed on the carbon at the rate of about 250 troy ounces of gold and silver per ton of charcoal.

The gold-and silver-laden carbon is stripped by hot sodium hydroxide and sodium cyanide solution at 190 F, then reactivated in a kiln and returned to the system. The pregnant solution is plated out on steel wool bats used as cathodes. The recovered gold and silver (in a 2 to 1 ratio) is then combined with fluxes and heated in crucible reduction furnaces to produce about 2,200 ounces of dore buttons weekly. These buttons are remelted and cast into two, 1,100 ounces bars per week for shipment to a refinery.

Water source is from Jett Canyon (8.5 miles west and 760 feet above the plant) within the Toiyabe Mountain range, and two wells. A deep well situated down slope from the waste dump is tested periodically for possible seepage of the cyanide leach solution.

Reference: Skillings, D. N., Jr., Skillings' Mining Review, 68(9):8-9, 16-17, March 3, 1979.

Subject: Eberle Mine: A Heap Leaching Case History

This article examines in detail (including a flow sheet) the heap leaching process operated at the Eberle mine by Challenge Mining Company in western New Mexico's Cooney district. Here, because the low-grade ore could not sustain a mining and milling operation, the district lay virtually dormant for 30 years until the advent of the cyanide heap leaching process.

Total investment at the Eberle operation is \$600,000, with about 34,000 short tons of ore capacity per year. A 60-day leaching period is expected to recover 70 percent of the gold and silver. Washing requires another 20 days at the Eberle operation. Eighty-five percent of the silver is precipitated with sodium sulfide before the pregnant leach solution is pumped through the charcoal stripping columns, and the balance is precipitated after stripping and before electrowinning to recover gold.

Preliminary analysis indicates operating costs (based on surface vein mining and processing of old mine dumps) of \$10.41 per ton. Projected profits are such that the investment should be recovered in about 2 years. Future operations will be supported by underground mining at the old Eberle mine.

Reference: Eveleth, R. W., Adapted from "New Methods of Working an Old Mine-Case History of the Eberle Group, Mogollon, N. M.," Mining Engineering, 31(2):138-140, February, 1979.

Subject: Heap Leaching is Small Miner's Golden Opportunity

If the ore is relatively free of carbonaceous cyanide-consuming and acid forming agents, and also free of excessive fines, heap leaching is ideal for the small scale gold and silver miner. This article explains the process, including costs, in some detail with considerable emphasis on stacking of the ore. (An actual gold heap leaching operation is described in this issue of the Technical Awareness Bulletin, "Smoky Valley Operations At Its Round Mountain Mine in Nevada".)

The author estimates that an entire heap leaching/recovery operation sized to leach 10,000-ton heaps in about 2 months can be put into production for \$200,000 (exclusive of mining costs) with minimal labor requirements, make-up sodium cyanide cost of about \$0.15 per ton per month and a power requirement of 0.0003 kw per ton. The low capital cost and relative ease of operation is attributed to two recent developments: (1) plastic membranes for the leach pads and (2) an activated carbon process for recovery of the gold and silver.

The most important factor in the leaching operation is the permeability of the rocks and of the heap as a unit. Permeability of the heap as a unit cannot be adequately determined in the laboratory. Therefore, heap construction is a critical operation and should be piloted in the field. Compaction of the heap should be avoided. The building of successive layers of cones as high as 16 feet, by the use of a mobile stacker/conveyor or a clamshell bucket on a small crane, is usually preferred.

Cyanide leaching of the heaps is best accomplished by a sprinkler system which guarantees good distribution, although it is more complicated and requires more maintenance than the ponding technique.

Single-use plastic leach pads, where the heap is leached as long as it is returning values, allows better flexibility than periodic removing and rebuilding of heaps on multiple use pads. Expended and washed heaps can be sealed with clay or contoured and vegetated for environmental compliance.

Reference: Kappes, D. W., Adapted from "Leaching of Small Gold and Silver Deposits", Mining Engineering, 31(2):136-138, February, 1979.

Subject: New Swedish Scheelite Flotation Process Results

In a prior issue of the Technical Awareness Bulletin, Index 10.2.8, page 2, the article "How Swedish Metallurgists Developed Selective Scheelite Flotation Process" was presented. After almost a year of operation, the present article describes the process in more detail with flowsheets of the grinding circuit, the copper flotation circuit, and the scheelite flotation circuit.

After grinding in a rod mill and in a ball mill in circuit with a cyclone, the 80 percent passing size is about 160 microns. Chalcopyrite is recovered as a valuable byproduct by a long scavenger series and four cleaner stages. Rejects from the copper flotation are conditioned with fatty acid which flocculates the scheelite into very stable aggregates which require special efforts to remove entrapped particles of calcite and fluorite. Tailings are flocculated with caustic lime before being pumped to the tailings pond. Effluent from a subsequent clarification pond has a turbidity of <10 FTU and contains <6 mg per liter of suspended matter, with only about 50 ppm of copper. Fluorine tests show 4-5 mg per liter. Attempts to lower this are in progress.

Selectivity in the 26 to 125 micron range is very good with a 97.6 percent recovery in a concentrate containing 73.4 percent WO_3 or 91.2 percent $CaWO_4$.

Future development includes: (1) improved stability of operations by using on-line analyses and reporting, (2) additional future development in the grinding stages where losses occur in the ultra fines; perhaps pebble grinding will replace ball milling, and (3) recoveries of 70 to 80 percent of the fluorite at concentrate grades of 98 and 95 percent, respectively. The latter will be achievable only at considerable investment, and will depend on recovery of the currently depressed fluorite market.

Reference: Grasberg, M., World Mining, 32(3):54-58, March, 1979.

Subject: Carbothermic Reduction of Domestic Chromites

All of the chromite used to produce ferrochrome in the U.S. must be imported. Because of the increasing demand for chromium and our dependence on foreign sources, the USBM has undertaken laboratory research to maximize iron and chromium recovery while decreasing energy requirements, from low-grade ore found in the U.S. This paper describes the work to determine the optimum types of carbonaceous reductants, temperatures, and duration of reduction to prereduce the domestic chromites in preparation for smelting. (Advantages of prereduction as opposed to direct smelting of concentrates include lower electrical energy requirements, improved smelting control and productivity, improved slag quality, and better overall recovery.)

Two chromite materials were used, representing the two major types of deposits; Mouat concentrate from Montana, a stratiiform, peridotite-gabbro complex containing 34 percent chromium oxide and 21 percent iron oxides, and the High Plateau ore from northern California, a podiform deposit, containing 54 percent chromium oxide and 17 percent iron oxides. Five reductants were investigated: coal char, coke breeze, metallurgical coke, petroleum coke, and shell carbon.

The reduction mechanism of both chromites cannot be described by simple kinetic equations; however, it is believed to be nucleation controlled. Based on the degree of reduction and metallization, coal char is the preferred reductant for both chromites up to 1300 C. At 1400 and 1500 C metallurgical coke is the choice for the Mouat chromite while coal char remains the choice of the High Plateau chromite. Reduction for both of the chromites is greatest during the first 15 minutes, and increasing temperatures tend to increase the degree of reduction and metallization.

The maximum reduction and metallization for 1 hour of both chromites are shown in the following tabulation:

<u>Material</u>	<u>Temp C</u>	<u>Reduction %</u>	<u>Metallization %</u>
Mouat chromite (High iron)	1200	32	27
	1300	57	44
High Plateau chromite (Low iron, metallurgical grade)	1200	25	12
	1300	49	33

Reference: Nafziger, R. H., J. E. Tress, and J. I. Paige, Metallurgical Transactions (B), 10B(1):5-14, March, 1979.

Subject: Cobalt and Nickel Recovery from Missouri Lead Belt
Chalcopyrite Concentrates

The Missouri Lead Belt reserves are estimated at 325 million tons, grading (in addition to lead and zinc) 0.3 percent copper, 0.02 percent nickel, and 0.015 cobalt. Copper recovered accounts for about 1 percent of the national primary copper production, and the copper concentrate fraction carries up to 30 percent of the nickel and cobalt content of the ore. Prior USBM studies on the recovery of nickel and cobalt were ineffective on the copper concentrates, and this investigation concentrated on this subject. The payoff would be the recovery of several tens of millions of pounds each of nickel and cobalt.

The Bureau found that leaching of the chalcopyrite concentrate from this source with ferric chloride concentrated nickel and cobalt in the residues at concentrations from 10 to 16 percent Ni and 8 to 11 percent cobalt, with recoveries estimated at about 80 percent. Froth flotation of reground and pretreated copper concentrates resulted in 67 percent of the nickel and cobalt reporting in concentrated forms to the tailings. Although the chloride leach appears to be more effective, it is not so adaptable to Missouri Lead Belt mill practice.

A moderately scaled-up demonstration (60 kg/hr) conducted at a commercial mill confirmed the initial laboratory finding relative to the applicability of flotation.

Reference: Clifford, R. K., and L. W. Higley, Jr., USBM-RI 8321, U.S. Department of the Interior, 1978.

Subject: Selective Recovery of Copper from Copper-Nickel Sulfide
Concentrates by Applying Segregation Technology

Technology for the "segregation" process (chloride-activated reduction of copper on coke in a solid-state operation) is well-documented for treatment of oxide ores and for dead-roasted chalcopyrite calcine. This report by AMAX researchers gives the results of laboratory studies directed toward selective "segregation" of copper from copper-nickel-iron sulfide concentrates. The studies were conducted on low-grade (4.6 percent Cu, 3.1 percent Ni, 45 percent Fe, 28 percent S) concentrates from Botswana, and higher-grade concentrates from Minnesota.

Concentrates were dead-roasted to 0.2 percent S max., and the calcine was blended with 0.5 percent NaCl and about 5 percent coke. Also, the addition of up to 10 percent SiO₂ to the silica-lean Minnesota concentrate aided segregation. Small-scale (50 gm of calcine) tests were conducted under a flowing moist CO atmosphere for 4 hours at 675 C. In these tests, 85-92 percent of the copper was reduced to metal, and greater than 99 percent of the nickel remained with the unreacted calcine. Tests of 600 gram lots were conducted, with products separated by flotation. (Copper was activated with Cyanamid Aerofloat 208 in conjunction with Aerofroth 65.) Under optimum conditions, about 80 percent of the copper and 0.1 percent of the nickel were collected in the flotation concentrate. Middlings, with intermediate amounts of copper and nickel were suitable for recycle through the segregation treatment. Simple melting of concentrate samples yielded blister copper--98 percent Cu, 0.75 percent Fe, 0.05 percent Ni. Thus, energy-efficient segregation treatment has been shown capable of separating metallic copper from mixed copper-nickel sulfides, with nickel concentrated in the tailing. SO₂ emissions are restricted to the enriched roaster offgas, which is readily processed in an acid plant.

Reference: Opie, W. R., L. D. Coffin, D. L. Armant and O. F. Cimler,
Metallurgical Transactions, 10B(1):27-32, March, 1979.

Subject: Amax will Use Energy-Saving Nickel Extraction Process
in New Caledonia

A new nickel extraction process has been developed and piloted tested by Amax which will reduce the energy required to extract nickel by 40 percent when compared to a pyrometallurgical process. A unique feature of the sulfuric acid leaching process is that it can economically treat both the limonitic layer (low nickel and magnesium content) and the garnieritic fraction (high nickel and magnesium content) of nickel-bearing laterites. A combination of high-pressure and atmospheric-pressure leaching is used, with the residual acid in the solution neutralized by the readily soluble magnesium from the garnieritic fraction at atmospheric pressure. The key to the process is the consideration of elemental sulfur as a source of primary energy with credits for the sulfuric acid by-product.

Reference: Engineering and Mining Journal, 179(11):32, November, 1978.

Subject: The Kennecott Process for Nickel-Slag Cleaning

Since 1975, it has been confirmed that a stirred electric furnace for cleaning copper and molybdenum from oxidized slags, produced by converters or the Noranda Process, was feasible. Because nickel recovery from oxidized slag is thermodynamically similar to the case for molybdenum, the application to nickel was considered.

This paper describes the thermodynamics and laboratory experiments on nickel-slag cleaning considering the Kennecott Process from two points of view: (1) modifying existing smelters by treating nickel converter slag directly in a stirred electric furnace and (2) developing a direct-smelting process including a pyrometallurgical slag cleaning step. Experimental data were obtained on the rates of magnetite reduction and metal recovery as a function of slag temperature and composition, matte composition, and furnace and stirrer design.

By adding the stirred electric furnace to existing equipment, an increase of about 15 percent in plant capacity can be expected. A commercial operation built to incorporate the stirred electric furnace has been designed for a nickel smelter along the lines of Kennecott's Utah Copper Division furnace that is in use.

The advantages of the Kennecott slag-cleaning process are (1) fast magnetite reduction, (2) fast digestion rate of concentrate or flux, and (3) a higher recovery of metal values.

Reference: Ammann, P. R., J. J. Kim, and T. A. Loose, Journal of Metals, 31(2):20-25, February, 1979.

Subject: Galena-Sphalerite-Chalcopyrite Flotation
at St. Joe Minerals Corporation

This paper discusses the significant variables in the differential flotation of galena, sphalerite, and chalcopyrite at St. Joe's Bushy Creek mill in Missouri.

The ore is normally 3-10 percent lead, 0.4-2.0 percent zinc, and 0.05-0.40 percent copper in a dolomitic limestone gangue. Isopropyl xanthate is the primary collector for galena and chalcopyrite flotation. Zinc sulfate depresses sphalerite. Copper recovery is increased by the addition of Z-200; however, excess is avoided to prevent floating sphalerite.

High zinc recovery requires 5 minutes' conditioning of the tailings with copper sulfate to activate the sphalerite. Insufficient copper sulfate produces a high zinc tailing assay. Residual xanthate and Z-200 from the lead copper circuit is sufficient collector for good recovery and grade for zinc-lean tailings, but added collectors are necessary at >0.4 percent zinc. Sodium cyanide is added to the zinc recleaner to depress iron minerals. Cyanide is necessary for zinc recovery, although the mechanism is not understood. Water added at this point increases the zinc concentrate grade to 58-60 percent zinc.

Lead-copper separation is made by a 3-5 minute conditioning period with causticized starch and SO_2 in which the chalcopyrite is floated away from the depressed galena. Preconditioning with potassium dichromate improves the grade by lowering the lead assay 1-2 percent. The SO_2 is added to keep the pH at 4.5-5.0, which also helps to depress the lead. Starch is a general depressant and will depress the copper if insufficient SO_2 is used.

One of the most significant variables is the xanthate level; if the optimum is used, good separation is not difficult. However, if excess is used, lead is difficult to depress.

Reference: Clifford, K. L., E. J. Haug, and K. L. Purdy, Mining Engineering, 31(2):180-182, February, 1979.

Subject: The Characterization of Mercury
in Some Sulfide Concentrates

The presence of mercury in many sulfide ores, although in small quantities, could pose an environmental problem in both smelting and hydrometallurgical processing because of the toxicity of its vapor and compounds. This paper examines in detail mercury distribution in the copper-gold ore of the Timmons, Ontario, Pamour Porcupine operations and in the pyritic zinc-lead-copper-silver ore from the mill of the Brunswick Mining and Smelting Corporation, Bathurst, New Brunswick. Flowsheets of both operations and several tables of analytical data are presented.

In both milling operations, most of the mercury reports to the various concentrates, which may contain in the range of 5-15 ppm mercury. Mercury concentrations in plant tailings typically range from about 0.1-0.7 ppm.

A number of analyses were made of the minerals present in the concentrates of the two operations to determine which of the minerals present were responsible for the mercury content. The copper concentrate of the Pamour Porcupine Mines contained 60 percent of the mercury in less than 2 percent of the input weight, with the three mercury-bearing minerals identified as native silver-electrum, tennantite-tetrahedrite, and sphalerite. In the concentrate from the Brunswick operation, the zinc concentrate contained 67 percent of the mercury in 11 percent of the input weight. More than 90 percent of the mercury was contained in the sphalerites with tetrahedrite and pyrite as possible minor carriers.

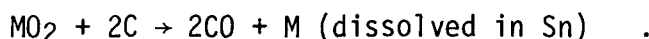
During pyrometallurgical processing of either copper or zinc concentrates, most of the roaster-volatilized mercury is trapped by the weak acid scrubber as a $(\text{Hg,Cu})_x (\text{Se,S})_y$ solid, but a small amount passes into the sulfuric acid. Only about 1 percent of the mercury input remains with the calcine.

Reference: Dutrizac, J. E., and T. T. Chen, CIM Bulletin, 72(803):201-208, March, 1979.

Subject: A New Carbothermic Process

Carbothermic reduction of oxides of highly reactive metals (U, TiO₂, UO₂, Al₂O₃, SiO₂, ZrO₂, and MgO, etc.) usually results in the formation of carbide or mixtures of the metal and carbide, neither of which is particularly useful or saleable.

The authors have recently developed a new method for the production of commercially pure, highly reactive metals which eliminates the formation of the carbide and the use of very high temperatures. The method uses a liquid metallic solvent (tin) to lower the chemical activity of the reduced, dissolved reactive metal, eliminating formation of the carbide. The reaction is as follows:



The choice of the liquid metal solvent is very important. Sn has been found a suitable solvent for uranium and several other metals; however, additional research is needed to find the best solvent for each case.

The paper describes the laboratory work done using uranium to establish the overall reaction of the process and the effects of temperature and pressure. The oxide and carbon were pelletized before insertion in a quartz tube and heated by induction to between 1550 and 1630 C under a pressure of 1-10 torr of carbon monoxide. The U-Sn was then heated by induction in a yttria crucible to 1300 C in a vacuum of 10⁻⁵ torr to distill the tin, thus allowing recovery of the uranium.

Reference: Bakshani, N., N. A. D. Parlee, and R. N. Anderson, Industrial Research/Development, 21(2):122-126, February, 1979.

Subject: Future Effects on the Mining Industry
of 1977 Clean Water Act

The author, in this article, examines the Clean Water Act of 1977 and its predecessor, The Federal Water Pollution Control Act of 1972, to determine the implications they will have on the mining industry. The Clean Water Act of 1977 amended parts of the 1972 Act. Important factors include extensions of deadlines for compliance.

A proposal by the EPA in August of 1978 was made to revise the existing National Pollutant Discharge Elimination System regulations. In these revisions, the EPA has shown that its intention is to propose and work for the most stringent controls possible on each and every pollutant from each industrial plant. The industry will have to assume "worst case" interpretation by the EPA of every proposed regulation. In short, mining companies must expect to be required to achieve zero discharge of pollutants to surface waters and to the ground.

The cost of developing a new mine will often not be worth the economic risk, with the long and costly environmental studies that will be required before construction can begin, according to this analysis. Existing mines could easily be closed with job losses and adverse impact on the balance of payment position of the U.S.

Reference: Pickering, I. G., Mining Congress Journal, 65(2):75-80,
February, 1979.

Subject: Plasma Process is Ready for Metals Recovery

Plasma technology, which seems to have something going for it in numerous potential pyrometallurgical applications, appears to be on the verge of another breakthrough. In Farinqdon, Oxfordshire, England, a unit will make trial runs this spring to recover platinum-group metals from chromite ore.

The technique, called the expanded precessive plasma process (EPP), centers on a water-cooled, metal sheathed, plasma gun with a doped tungsten electrode and a small flow of argon. Mounted at 15 degrees from the vertical, and revolving at 1500 or more rpm, the gun produces a precessing cone of plasma between the gun and a counter electrode or the hearth of the furnace. Smelting, or any other chemical action, occurs during the 300 millisecond time that the falling curtain of fine material takes to pass through the high-temperature zone.

The trial runs this spring will take place in a 1,400-kVA demonstration unit (a 300-kVA unit has already completed successful testing). Tetronics, a British firm, and the U.K. affiliate of Foster Wheeler, developed this technique and is believed to be ahead of competitors in application of EPP. Success in this test, sponsored by Texas Gulf, could trigger a big breakthrough for the EPP process.

A future use of EPP could be in ferrochrome or steel manufacturing which would call for a 6,000-kVA unit costing \$5 million. It is projected that the cost of ferrochrome production would be lowered by \$0.04/lb.

Three distinct advantages of EPP should be noted: (1) the use of fine particulate ore (<0.5-mm diameter) which is the size many ores are reduced to during beneficiation, (2) the high temperature which permits strongly endothermic reactions in very short periods of time, and (3) precise control of furnace atmosphere.

Reference: Chemical Engineering, 86(5):75-77, February 26, 1979.

Subject: Baghouses: Separating and Collecting Industrial Dusts

This two-part treatise reviews in comprehensive fashion baghouse technology. The subjects discussed in the first part include:

Fabric properties, material, weaves (including staple construction), comparative performance, cleaning methods, design requirements, housing construction, construction, testing for fire hazard.

The second part tells how one goes about buying a baghouse installation--the analysis of bids, guarantees, flow-rate estimating, performance assessment, etc. Monitoring of installation, startup, and testing for emissions and leaks are also covered.

(This review is not specific to nonferrous metals operations, many of which require baghouses, but the ABC's of buying, owning, and operating these facilities covered in the pair of articles is a worthwhile addition to the personal library of staff with baghouse responsibilities.)

Reference: Kraus, M. N., Chemical Engineering, 86(8):94-106, April 9,
and 86(9):133-142, April 23, 1979.

NONFERROUS METALS RESEARCH REPORTS AVAILABLE THROUGH
NATIONAL TECHNICAL INFORMATION SERVICE

Assessment of Technology for Possible Utilization of Bayer Process Muds-
REF. NO. EPA-600/2-76-301

Control of Sulfur Dioxide Emissions from Copper Smelters: Volume I-Steam
Oxidation of Pyritic Copper Concentrates-REF. NO. EPA-650/2-74-085a

Control of Sulfur Dioxide Emissions from Copper Smelters: Volume II-Hydrogen
Sulfide Production from Copper Concentrates-REF. NO. EPA-650/2-74-085b

Copper-REF. NO. EPA-600/1-77-003

Determination of Hazardous Elements in Smelter-Produced Sulfuric Acid-
REF. NO. EPA-650/2-74-131

Energy Consumption: The Primary Metals and Petroleum Industries-REF. NO.
EPA-650/2-75-032b

Environmental Considerations of Selected Energy Conserving Manufacturing
Process Options. Vol. VIII. Alumina/Aluminum Industry Report-
REF. NO. EPA-600/7-56-034h

Environmental Considerations of Selected Energy Conserving Manufacturing
Process Options. Vol. XIV. Primary Copper Industry Report-
REF. NO. EPA-600/7-76-034n

Industrial Process Profiles for Environmental Use: Chapter 25. Primary
Aluminum Industry-REF. NO. EPA-600/2-77-23y

Industrial Process Profiles for Environmental Use: Chapter 26. Titanium
Industry-REF. NO. EPA-600/2-77-023z

Measurement of Sulfur Dioxide Particulate, and Trace Elements in Copper
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Metallic Recovery from Waste Waters Utilizing Cementation-REF. NO.
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Methodology for Assessing Environmental Implications and Technologies:
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Operation of a Sulfuric Acid Plant Using Blended Copper Smelter Gases-
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Process Modifications for Control of Particulate Emissions from Stationary
Combustion, Incineration, and Metals-REF. NO. EPA-650/2-74-100

Reclamation of Sulfuric Acid from Waste Streams-REF. NO. EPA-670/2-75-016

Regeneration of Chromated Aluminum Deoxidizers-REF. NO. EPA-660/2-73-023

Systems Study of Conventional Combustion Sources in the Primary Aluminum
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Water Pollution Control in the Primary Nonferrous Metals Industry, Vol. 1-
Copper, Zinc, and Lead Industries-REF. NO. EPA-R2-73-247a

Water Pollution Control in the Primary Nonferrous Metals Industry, Vol. II-
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