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information circular **8225**

Pdf. 1 of 3

COPPER

A Materials Survey

By A. D. McMahon

UNITED STATES DEPARTMENT OF THE INTERIOR

BUREAU OF MINES

1965

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UNITED STATES
DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
WASHINGTON, D.C. 20240

January 11, 1965

Hon. Edward A. McDermott
Director
Office of Emergency Planning
Washington, D. C.

Dear Mr. McDermott:

In accordance with the agreement of April 15, 1955, between

the Department of the Interior and the Office of Civil and

Defense Mobilization, which assigned responsibility to Interior

for preparation and revision of Surveys covering 45 mineral

commodities, the Bureau of Mines with the cooperation of the

Geological Survey has prepared and herewith transmits to you

the revision of Copper, A Materials Survey.

Sincerely yours,

Acting Director

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Tables—Continued

COPPER

A Materials Survey

By

A. D. McMahon [1]

Introduction

This survey is one of a series of Bureau of Mines publications designed to serve the needs of Government and industry for comprehensive information on mineral commodities and activities.

Copper, a versatile metal, has been vital to all civilizations from prehistoric times to the present. Its essential value in antiquity and until the nineteenth century arose from its malleability and ease of working, its corrosion resistance and durability, its attractive colors in alloyed and unalloyed forms, and, of course, its availability. The principal uses were for tools, utensils, vessels, weapons, pipe, statuary and other objects of art, and for building and architectural purposes where formability, permanence, and beauty were necessary qualities. Utilization of copper based on these physical and mechanical properties has grown tremendously for commercial, structural, mechanical, architectural, and art items; and its capacity for forming numerous alloys has led to a myriad of applications from miniature copper and brass eyelets to huge bronze battleship propellers. The early nineteenth century marks the epoch of the greatest use of copper, transmitting electrical energy. This particular property of copper is fundamental to the spectacular growth of the electrical industry and all associated industries relying on electricity for power, light, and heat. More than half of all copper produced is used for transmission of electricity.

The copper industry has been able to expand along with the mounting demand because of the ever increasing discovery of resources in the world and advances in mining, metallurgical, and fabricating technologies. World production and consumption rose progressively from about 18,000 tons in 1800 to more than 4.5 million tons in 1960. The estimated world reserve of primary copper has more than doubled since 1935, being 212 million tons in 1960. Ninety percent of this reserve is credited to eight countries—the United States, Canada, Chile, Peru, the Soviet Union, Poland, Northern Rhodesia, and the Republic of the Congo. The United States share is 32.5 million tons of copper in ore, averaging approximately 0.9 percent.

Another reserve being accumulated in the industrial countries of the world is copper products now in use that will eventually be discarded as scrap and become available for recovering secondary copper. In the United States this resource of more than 30 million tons, provides about one-fourth of the annual supply.

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[1] Physical scientist, Branch of Nonferrous Metals, Bureau of Mines, Washington, D.C.

Work on manuscript completed July 1963.

The ores of copper are classified as sulfide or oxidized ores. They are further distinguished by copper content, type of copper-bearing mineral, mode of occurrence, and associated minerals; and they may be designated as high grade or low grade, primary or secondary, or disseminated, replacement, vein, massive sulfide, native copper, and complex. Although many copper minerals are known, relatively few are commercially significant. The predominant copper sulfide minerals found in the sulfide ore deposits of the world are chalcopyrite, chalcocite, bornite, enargite, and covellite; the principal oxidized minerals are chrysocolla, malachite, and azurite. Antlerite, atacamite, and brochantite are important oxidized ore minerals of some deposits in Chile. The copper-bearing ores of many deposits contain significant values in minerals of other metals that are recovered as byproducts or coproducts in concentrating, smelting, and refining processes. Metals recovered from copper ores as byproducts include iron, lead, zinc, molybdenum, nickel, cobalt, gold, silver, the platinum group metals, selenium, tellurium, and arsenic.

Most copper ores are mined from large low-grade deposits, which are the source of about 80 percent of the annual world production of primary copper. The economical recovery of copper from such ores requires mass extraction and treatment processes. Deposits near the surface are worked by open-pit mining; caving or other large-scale mining methods (cut and fill, shrinkage stoping, room and pillar) are used for underground mining of deep ore bodies. The metallurgical processes for recovering copper metal from its ores are concentrating, smelting, and refining for sulfide ores and leaching followed by chemical precipitation or electrowinning for oxidized ores. The final refinery product is almost pure copper (99.9+ percent) in refinery shapes (wirebars, billets, cakes, slabs, ingots, and ingot bars), which are used for producing copper wire and copper and copper-base alloy, semifabricated products such as strip, sheet, plate, tube, rod, and shapes. Figure 1 illustrates the sequence of operations followed to obtain copper products from copper ore.

The largest copper producer and consumer in the world throughout the 1925–62 period was the United States. Before World War II mine production and the recovery of secondary copper were adequate for domestic requirements and provided substantial quantities for export. When the United States entered World War II, however, large supplies of copper were required from abroad and the United States became a net importing nation. This situation continued through 1960.

Approximately one-half of the total United States supply of copper is derived from domestic mine production; a little more than one-fourth is obtained from imports; and about a fourth is secondary copper recovered from old scrap. Mine productive capacity in the United States has not changed substantially since World War I, according to peak production years associated with the wars and other times of increased industrial activity. Beginning with 1916 the mines produced about 1 million tons of copper annually in 1916, 1929, 1942–43, 1955–57, and 1960–63. However, because of the development of more efficient mass mining techniques and equipment, mine productivity, in output of copper per man hour, has increased almost fivefold.

Secondary copper is produced from new and old scrap, the two general classes of scrap copper. Old scrap consists of metal articles that have been discarded because of wear, damage, or obsolescence, usually after serving a useful purpose. New scrap is generated in manufacturing items for consumption and reenters the processing and fabrication cycle as run-around copper which does not contribute to new supply. Only copper from old scrap, whether in unalloyed or alloyed form, is a real addition to supply. There has been no sustained increase in the supply of secondary copper for many years. Since 1941 the recovery of copper from old scrap has fluctuated mostly between 400,000 and 500,000 tons a year, reaching a maximum of 515,000 tons in 1955—a period of high prices for copper.

Imports have come largely from properties owned by U.S. investors in Mexico, Chile, and Peru and from Canada. Chile has been the principal source, followed in order by Canada, Peru, and Mexico. In recent years significant quantities were furnished by Northern Rhodesia, the Republic

of South Africa, and the Philippines. The United States exports refined copper to many countries—most of it destined for the United Kingdom, West Germany, Italy, France, and Japan.

Consumption of copper in the United States was at its highest during World War II. Since then, except for a few years, it has followed a declining trend caused to a marked degree by the substitution of aluminum, steel, plastics, and copper-clad materials for large-use items. The predominant users of copper are the electrical industry, the building construction and automotive industries, and manufacturers of industrial equipment and supplies. Wire mills and brass mills usually account for more than 95 percent of total consumption, wire mills having 50 percent or more in most years.

World consumption of copper increased almost 50 percent from 1954 through 1962, principally because of the flourishing industrial activity in Europe and Japan. The resultant new demand was met by a similar growth in world production, largely from expanded output in Canada, Chile, Peru, Northern Rhodesia, the Republic of the Congo, U.S.S.R., China, Australia, and the Philippines.

In the United States approximately 200 firms are engaged in producing and selling primary copper. Principal segments of the industry are mining, smelting, refining, fabricating, and marketing. The million tons of primary copper produced annually comes from large operations. Of 196 copper mines reporting production in 1962, the 25 largest mines accounted for 97 percent of the total output; the first 10 produced 74 percent, and the first 5, 48 percent.

A total smelting capacity of 8,800,000 tons is provided by 20 smelters and the total refining capacity consists of 1,963,500 tons at 12 electrolytic refineries and 371,000 tons at 6 fire refineries. About 35 companies in the United States are recognized as the important fabricators and users of primary copper; this they process into copper and copper alloy forms of sheet, strip, rod, tube, wire, and extruded and rolled shapes. Some of the larger fabricators are affiliated with the major copper producers who thus have facilities for processing ores from the mines to the finished copper and brass products.

Recovery of scrap copper constitutes an important branch of the copper industry in the United States. Old scrap collected by hundreds of scrap dealers is processed into secondary copper by primary smelters and refineries, secondary smelters (brass and bronze ingot makers), and copper mills.

Government legislation and programs involving segments of the copper industry usually comprise regulation, taxation, or aid to promote economic stability or to maintain national security. Some examples are the mining laws that regulate title, exploration, development, and extraction of ores from deposits on Government or Indian lands; import taxes levied for protection from foreign competition and reduction or suspension of such taxes for trade considerations; tax concessions granted mining enterprises for depletion in the Internal Revenue Code; laws regulating malpractices in securities and financial markets and prohibiting activities restraining trade, monopolies, and unfair trade practices. In emergency periods, such as World War II and the Korean War, price, export, import, distribution, and use controls were established for copper. Public Law 520, July 23, 1946, authorized acquisition of strategic and critical materials (including copper) for the Strategic Stockpile, and the Defense Production Act of 1950 provided for Government assistance in exploration of copper deposits, development and production loans, and guaranteed price contracts.

The copper supply outlook for the near future in the United States is encouraging. This view is based on an estimated ore reserve containing 32½ million tons of copper, an annual availability from secondary sources (old scrap only) of 450,000 tons, and a reserve of 1,142,000 tons of copper in the national stockpile. Also, the increased ore reserves and expanded copper production in Canada, Chile, and Peru provide ample sources of imports in the Western Hemisphere.

MINING
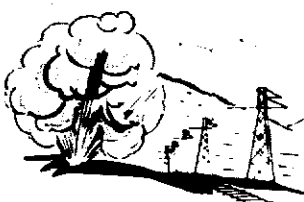
Blasting
The ore body is broken up by blasting.
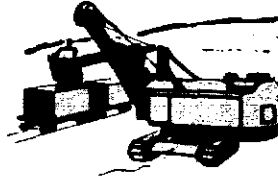
Loading
The ore, averaging about 1 percent copper, is loaded into ore cars by electric shovels.
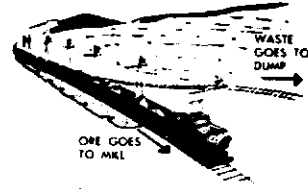
Hauling
The cars of ore are hauled to the mill.

MILLING
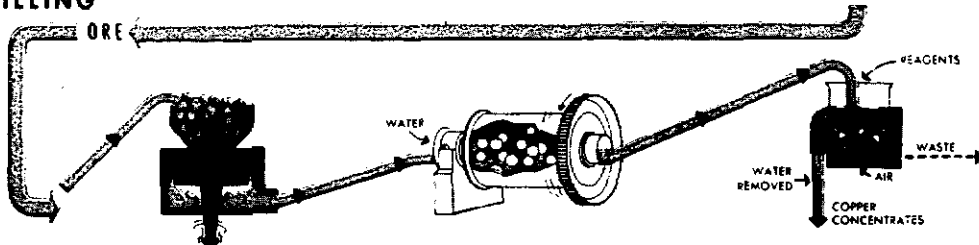
Crushing
The ore is crushed to pieces the size of walnuts.

Grinding
The crushed ore is ground to a powder.

Concentrating
The mineral-bearing particles in the powdered ore are concentrated.

SMELTING
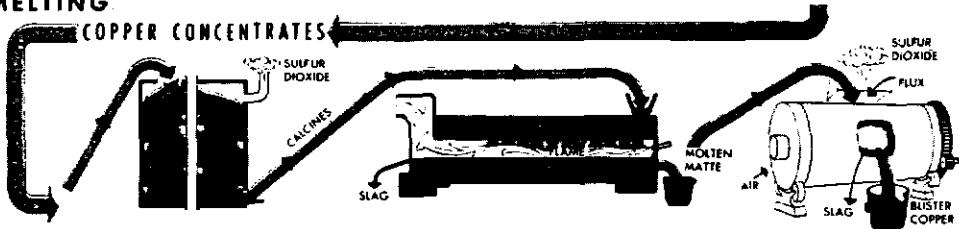
Roasting
The copper concentrates (averaging about 30 percent copper) are roasted to remove sulfur.

Reverberatory Furnace
The roasted concentrate is smelted and a matte, containing 32-42 percent copper, is produced.

Converter
The matte is converted into blister copper with a purity of about 99 percent.

REFINING
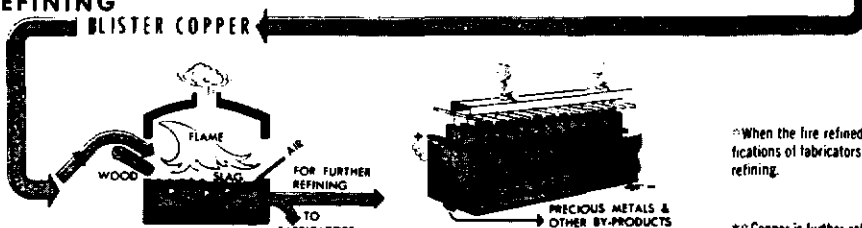
Refining Furnace
Blister copper is treated in a refining furnace.*

Electrolytic Refining
Copper requiring further treatment is sent to the electrolytic refinery.**

*When the fire refined copper meets the specifications of fabricators, it is used without further refining.

**Copper is further refined electrolytically when the special properties of electrolytic copper are required, e.g., when the copper is to be used for electrical conductors, and/or when precious metals are present in sufficient quantities to make recovery desirable.

FABRICATING
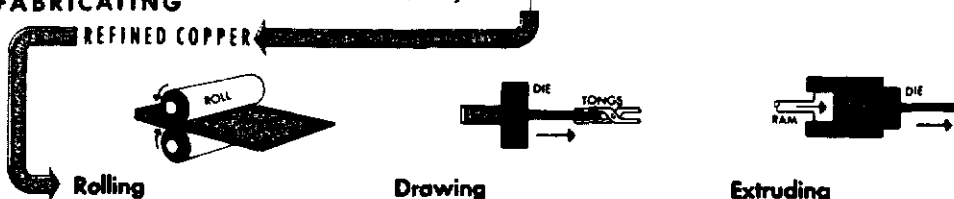
Rolling

Drawing

Extruding

Fire refined or electrolytic copper and/or brass (a mixture of copper and zinc) is made into sheets, tubes, rods and wire.

Sheets, tubes, rods and wire are further fabricated into the copper articles you see in everyday use.

FIGURE 1.—Basic Steps—Copper Ore to Finished Product.

(Courtesy, Kennecot Copper Corp.)

ACKNOWLEDGMENTS

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CHAPTER 1.—PROPERTIES, COMMERCIAL CLASSIFICATIONS, AND USES OF COPPER AND COPPER BASE ALLOYS

NATURE OF THE METAL

Copper is one of the few common metals that finds its greatest application in the commercially pure rather than the alloyed form, principally because of its superior performance as a conductor of electricity. However, metals such as zinc, tin, and others readily alloy with copper to form the widely used brasses and bronzes. Copper is an important alloying element in a large number of alloys having metals other than copper as the principal component and copper and copper-base alloy powders are formed into structural parts. This versatility and the exceptional service qualities inherent in the metal are responsible for its important role in the industrial economy.

PROPERTIES OF COPPER

The properties of copper having major significance are its high-electrical and thermal conductivities, corrosion resistance, good ductility and malleability, and high strength. In addition, copper has a pleasing color, is nonmagnetic, and is easily finished by plating or lacquering; it can be welded, brazed, and soldered satisfactorily. Certain of these basic properties can be improved for many applications by alloying with other metals. As a consequence the popular commercial brasses, bronzes, copper-nickel alloys, and nickel silvers have been developed.

Physical constants of copper:

Chemical symbol_____	Cu
Atomic number_____	29
Atomic weight_____	63.54
Valence_____	1 & 2
Crystal structure_____	Face-centered cubic.
Density, annealed, 20° C_____	8.89
Hardness, Moh_____	2.5–3
Melting point, ° C_____	1083
Boiling point, ° C_____	2595
Electrical resistivity, 20° C, microhm-cm., annealed_____	1.71

Physical.—Copper is the only metallic element besides gold having a rich natural color other than some shade of gray. The distinctive red color of metallic copper is best seen on a fresh or polished surface. Copper precipitated from solution is brownish red and the colloidal metal ranges from crimson or scarlet to various shades of blue and brown. Thin films of copper appear greenish-blue by transmitted light; molten copper has a golden color; and copper vapor is green. The compact metal has a bright metallic luster and takes a bright polish, which soon tarnishes when exposed to air. Old surfaces often have an orange tinge due to a film of cuprous oxide (Cu_2O). Full oxidation of a surface causes formation of a film of the black cupric oxide (CuO). Extended exposure to weathering produces a green (patina) surface deposit which varies depending on the atmosphere but the principal constituent is basic copper sulfate with traces of basic copper carbonate (malachite) for most atmospheres. Near salt water the amount of sulfate may be sharply reduced, with the carbonate increased substantially, and large amounts of basic copper chloride (atacamite) are included. In rare instances, basic copper nitrate has been reported where copper is exposed near power stations.

Grain size, fabricating method, and mechanical surface treatments (polishing, buffing, and vapor blasting) also affect the finished color. Combinations of alloy and mechanical treatments provide an unusual range of natural color finishes for copper; chemical treatments provide additional colors and finishes.

The density of commercial coppers ranges from 8.4 to 8.94 grams per cubic centimeter. Cast tough-pitch copper contains about 3 to 5 percent voids or gas holes, by volume, and such copper will have an apparent density of 8.4 to 8.7. The voids close during rolling; after working and annealing the density of tough-pitch copper is 8.89 to 8.94, depending on the residual oxygen content. Copper containing 0.03 percent oxygen has a maximum density of 8.92, and values as high as 8.9592 have been obtained for pure copper. The density of liquid copper is 8.22 near the freezing point. The shrinkage of copper on solidification is 4.96 percent.

Electrical.—The high electrical conductivity of copper, more than anything else, accounts for its widespread use in the electrical field. Only one metal, silver, is a better conductor in cross-section. Since aluminum has higher conductivity by weight, it is the principal rival of

copper for certain types of electrical installations.

The conductivity of copper is expressed in percentages, thus providing a convenient measure of quality for electrical purposes and for comparison with other metals. The basis for rating is a mass conductivity standard adopted in 1913 by the International Electrotechnical Commission and subsequently by the American Standards Association, American Society for Testing Materials, and other bodies. The standard mass resistivity is 0.15328 ohm (meter, gram) at 20° C; that is, a copper wire 1 meter long, weighing 1 gram, and having a resistance of 0.15328 ohm at 20° C. The value of 100 is assigned to the reciprocal of this standard resistivity, and conductivities of other specimens are stated in percentages of this conductivity. This standard conductivity is designated the International Annealed Copper Standard (IACS). A standard density of 8.89 grams per cubic centimeter was also adopted to permit expression of the standard in terms of both mass and volume units as in the following equivalents:

0.15328 ohm (meter, gram), 875.20 ohms (mile, pound); 0.017241 ohm (meter, sq. mm), 1.8241 microhm (cm, cm²); 0.67879 microhm (in, in²), 10.371 ohms (ft, mil)—all at 20° C.

ASTM specifications require all copper intended for electrical use to meet a 100-percent conductivity minimum. Commercial coppers designated for electrical purposes have conductivities between 100.5 and 101.8 percent. Very high purity copper prepared for research use has a conductivity of 102.3 percent.

The electrical resistivity of metals increases with temperature, the average change for copper between 0° and 100° C. being 0.42 percent for each degree. At extremely low temperature the electrical resistance almost vanishes for copper (and other metals). Some resistivity values at temperatures from almost absolute zero (−273° C) to that of molten copper are:

Temperatures, °C:	Resistivity, microhm-cm
−258.6	0.014
−206.6	.163
−150	.567
−100	.904
+20	1.7241
+100	2.28
+200	2.96
+500	5.08
+1,000	9.42
+1,500 (liquid)	24.62

Source: Handbook of Chemistry and Physics, 1960–61, The Chemical Rubber Publishing Co., Cleveand, Ohio, 42 ed., 1960, p. 2590.

Copper may be cold worked to great extremes with but a slight loss of electrical conductivity, compared to losses in brasses and other alloys. The most severe cold working does not reduce the conductivity more than 3 percent.

The conductivity of copper by weight is surpassed by several light metals, notably aluminum. Relative conductivity values for silver, copper, and aluminum by volume and weight are:

Metal:	Conductivity percent by—	
	Volume	Weight
Silver	100	44
Copper	94	50
Aluminum	57	100
Iron	16	---

Thermal.—Copper is the second best heat conductor of all metals, again being surpassed only by silver. Comparative values for thermal conductivity for different metals at room temperature (18° C) are as follows:

Heat Conductivity, cal/cm/cm²/sec/° C:

Silver	1.006
Copper	.934
Gold	.705
Aluminum	.480
Iron	.161

Important thermal constants of copper are:

Melting point	1083.0° C (1981.4° F)
Boiling point	2595° C (4703° F)
Latent heat of fusion	48.9 cal/gram (89.8 Btu/lb)
Heat capacity	.0919 cal/gram/° C
Linear coefficient of expansion.	16.42 x 10⁻⁶/° C
Thermal conductivity, 18° C.	.934 cal/cm/cm²/sec/° C

Mechanical.—One or more of the qualities of strength, hardness, ductility, malleability, and ease of joining usually are desired with the other physical and chemical properties of copper. The properties denoting plastic deformation are rated highest, and those referring to strength are rated lowest in fully annealed metal.

Pure copper is not particularly strong or hard in the annealed condition, but both strength and hardness are increased considerably by cold working. Commercial coppers in the annealed state have tensile strengths from 32,000 to 35,000 pounds per square inch. Values to 75,000 psi have been obtained for severely cold worked copper. Annealed copper has a Brinell hardness of about 42, which can be increased to more than 100 by cold working. Copper is little hardened by quenching or other heat treatment. The tensile strength and elongation values obtained by cold working the three basic commercial types (tough pitch, deoxidized, oxygen free) are identical from a practical standpoint. This equivalence in mechanical properties is responsible largely for continuance of the historical tendency to consider both oxygen- and

phosphorus-bearing coppers as commercial forms of the metal, and to reserve alloy designation for products having more pronounced alteration of physical characteristics.

Annealed or hot-rolled electrolytic tough-pitch copper is very ductile. High purity copper is so ductile that it is difficult to determine the existence of a finite limit beyond which it cannot be cold worked. (No evidence of such a limit has been observed in wires drawn 99.4 percent, which had developed a tensile strength of 75,000 psi.) Copper is very malleable which allows the rolling or hammering of the fully annealed metal into very thin sheets without cracking. Although there are apparently no quantitative criteria for measuring malleability, copper compares favorably with gold and silver in acceptance of plastic deformation without rupture.

Soft copper has very little elasticity or resilience, but cold working increases values for both properties. Soft copper has an elastic limit or yield point of 3,000 to 17,000 psi, depending on the convention used in defining these terms; there is little true elasticity, and even small loads may produce some permanent deformation. Cold-rolled copper has a yield point of 40,000 to 55,000 psi, and the Young modulus for hard copper is about 16 million psi.

The endurance limit, which is the maximum stress a metal will withstand without failure during numerous cycles of stress, is 11,000 psi for annealed copper. This can be increased by work hardening to 15,000 psi for tough-pitch copper and to 19,000 psi for deoxidized copper.

Copper and many of its alloys may be joined by welding, brazing, and soldering. The welding of oxygen-bearing copper requires special techniques because the high temperature of the heated zone may cause rejection of the cuprous oxide to grain boundaries, weakening the area. Success has been reported with fast welding processes when the welding time is so short that virtually no oxide segregates, or when the segregation is so minor that the strength of the joint is not seriously impaired. Such processes are helium- or argon-shielded arc welding, a resistance in air method, and use of a gas torch with a slightly oxidizing flame. Deoxidized and oxygen-free coppers are readily joined by arc- and gas-welding processes and brazing. Properly welded joints in oxygen-free copper will have strength and ductility about 90 percent of the annealed strength of the base metal. The properties of the weld may be greatly improved by cold working.

The oxygen-bearing, deoxidized, and oxygen-free coppers are readily joined by torch, furnace, dip, and twin-carbon-arc brazing

processes, using either copper-phosphorus or silver-alloy types of brazing filler metals. (ASTM B260-56T and AWS A5.8-56T specifications for brazing filler metal.) By proper selection of the brazing process and filler metal it is possible to braze oxygen-bearing copper without loss of strength due to copper oxide segregation.

Soldering with the common types of tin-lead solders is an effective and simple way of joining sections of copper. Selection of a particular solder will usually depend upon the operating temperature and other working conditions of the required service as well as economic considerations.

Copper may be clad on other metals or may be clad with other metals, in each instance to achieve some special purpose. Copper cladding on steel is used to reduce the overall cost or to increase the strength while maintaining the high conductivity of copper. Copper cladding of aluminum combines the lightness of that metal and the conductivity of copper. Copper is clad with certain silver brazing alloys to facilitate the brazing of special types of tools and improve their performance.

Chemical.—Copper has an atomic number of 29 and an atomic weight of 63.54. It is the first member of subgroup IB of the periodic table, being followed by silver and gold. Each of these three metals form a univalent series of salts, but copper also forms a bivalent series and gold a trivalent series that are more stable than the corresponding univalent compounds. The electronic configuration of copper is 2:8:18:1. Loss of the outermost electron produces the cuprous ion Cu^+, and a second electron may be lost in the formation of the cupric ion Cu^{++}. The formation of these two ions is an important factor in considering the corrosion behavior of copper. There are two stable isotopes of copper, Cu^{63}, consisting of 29 protons and 34 neutrons, and Cu^{65}, with 29 protons and 36 neutrons. There are also known to be at least eight unstable (radioactive) isotopes having the mass numbers 58, 59, 60, 61, 62, 64, 66, and 67.

Some of the characteristic features of the metals in this group are:

1. They occur as native metal and may be separated from compounds with relative ease.

2. They are not very active chemically. As the atomic weight increases, the chemical activity decreases.

3. Copper and silver oxidize very slowly in air at ordinary temperatures.

4. Their oxides and hydroxides are slightly basic.

5. They form insoluble chlorides, for example, $CuCl$.

6. Complex ions are readily formed, for example, $[Cu(CN)_2]^-$.

7. Complex cations are formed with ammonia, for example, $[Cu(NH_3)_4]^{++}$.

One of the main reasons for the wide use of copper and copper-base alloys is the excellent resistance to corrosion displayed in a wide range of environments. Copper possesses very high resistance to the atmosphere; to naturally occurring waters, both fresh and salt; and to alkaline solutions, except those which are distinctly ammoniacal. Behavior of copper in acids is greatly dependent upon oxidizing conditions which affect it adversely. It has good resistance to many saline solutions. However, copper offers low resistance to sulfur and sulfur compounds, producing a corrosion product of copper sulfide. It may be seriously damaged by solutions such as sea water and brine when these move against it at high velocities.

Copper rarely corrodes through galvanic action as a result of contact with other metals. In the electromotive series of elements, copper is near the noble end and normally will not displace hydrogen from acid solutions. Copper is cathodic to the commonly encountered metals—tin, lead, nickel, iron, zinc, aluminum, and magnesium—that can be used to displace copper from solutions of its salts. Mercury, silver, palladium, platinum, and gold—being below copper—can be displaced from their salt solutions by metallic copper. There are many conditions, however, in which displacement does not proceed in accordance with the common electromotive series. The potential differences between dissimilar metals vary, depending upon the nature of the corrosive environment.

The crystal structure of copper is face-centered cubic, the cube side being 3.6078 A at 18° C; the closest distance of approach of atoms, 2.551 A. This is a close-packed structure, being one of two possible structures formed by the closest possible packing of uniform spheres. Each atom has 12 equidistant nearest neighbors. The structure has the highest degree of atomic concentration and symmetry to be found in any crystal structure.

Common gases such as O_2, CO, SO_2, and H_2 dissolve in molten copper. Some of these gases undoubtedly react with the copper or with other compounds; thus dissolved oxygen is probably in equilibrium with Cu_2O:

$$4Cu + O_2 \rightleftharpoons 2Cu_2O,$$
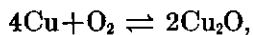
and reducing gases such as CO and H_2 react with Cu_2O to form copper, CO_2, and H_2O. The nature and amount of gas remaining in copper just before it solidifies markedly affects the properties of solid copper.

Copper will dissolve in most acids when aided by oxidizing action to form soluble copper salts. Copper also forms complex salts with ammonium compounds and with cyanides. In the complex copper-ammonium salts the copper is in the form of a cuprammonium ion which has a deep-blue color. Cupric ions impart a greenish-blue color to the solution; cuprous ions are colorless.

Examples of complex cyanides are the soluble sodium and potassium salts $NaCu(CN)_2$ and $KCu(CN)_2$. In these salts the copper is present in the negative ion $[Cu(CN)_2]$; these complex cyanides are used in electrolytes for electroplating copper.

Cupric chloride forms a soluble double chloride with ammonium chloride—$CuCl_2$. $2NH_4Cl.2H_2O$. The insoluble cuprous chloride forms a similar compound with ferrous chloride, and this makes possible the dissolving of cuprous chloride in brines of ferrous chloride.

Copper is fungitoxic and is used more in preparing agricultural fungicides than any other metal. Compounds of other metals (silver, mercury, cadmium, and others) may be more effective fungicides but the popularity of copper compounds is largely due to a fairly high degree of fungitoxicity, a low toxicity for most higher plants and animals, and a lower price than most other metals of comparable effectiveness in disease control.

Effects of Impurities.—The conductivity, tensile strength, and other properties of copper are affected to varying degrees by the presence of impurities which may either (1) be dissolved in the copper in solid solution or (2) be insoluble in solid copper. The most important of those in the first class are nickel, iron, arsenic, antimony, and phosphorus. The second group includes bismuth, lead, selenium, tellurium, sulfur, oxygen, and oxides.

Oxygen is present in all commercial copper, except in the deoxidized and oxygen-free grades. Its effect on the mechanical properties is not great, slightly increasing the tensile strength and reducing the ductility as the oxygen content increases. In small amounts oxygen increases the electrical conductivity of commercial copper, very likely by oxidizing other impurities and thus removing them from solid solution. Large amounts of oxygen, however, reduce the conductivity by forming copper oxide, reducing the effective cross section of the metal.

Sulfur, selenium, and tellurium, in general, affect the mechanical properties of copper adversely. In amounts to 1 percent, selenium and tellurium may be used for increasing machinability.

Bismuth also has a harmful effect on the mechanical properties of copper; it interferes seriously with hot rolling and slightly with

cold rolling. Its effect may be partly neutralized by additions of oxygen, arsenic, and antimony. It is almost completely insoluble in copper.

On the whole, antimony is another harmful metal, although it is sometimes added to copper when high recrystallization temperatures are desired. In amounts of 0.5 percent and higher it hardens copper, decreases its ductility, and lowers its electrical conductivity. Antimony is similar in effect to phosphorus and arsenic and is intentionally added in small amounts (0.02–0.10 percent) to some high-brasses for increasing resistance to dezincification.

Arsenic is added intentionally to 0.6 percent because of its slight hardening and strengthening effect, especially in the cold-worked condition. It raises the recrystallization temperature but has little effect on the ductility and malleability. It decreases the electrical conductivity considerably. Arsenic is an excellent inhibitor of corrosion by dezincification in leaded muntz metal, admiralty brass, naval brass, and aluminum brass; 0.02 to 0.10 percent is added to these alloys for that purpose. It is not useful in the alpha-beta brasses because it does not inhibit dezincification of the beta phase.

Next to oxygen, silver is the most common element in commercial copper. It has negligible effect on the mechanical properties and electrical conductivity but significantly increases the recrystallization temperature and prevents softening of cold-worked material by short time exposure to heat.

The small amount of iron normally present in commercial copper has very little effect on its mechanical properties. In larger amounts, to 2 percent, it hardens and strengthens copper slightly without destroying ductility, but it reduces the electrical conductivity, especially in the absence of oxygen.

Lead is sometimes added in small amounts to copper to increase its machinability, but it should not exceed 0.005 percent if the copper is to be hot rolled; otherwise it will cause "hot shortness." If operations are conducted at room temperature, still larger amounts have little effect on the ductility of copper. It can be rendered somewhat less harmful by introducing oxygen.

Gases have a great effect on the physical properties of copper, especially in castings. Hydrogen is very soluble in liquid copper but will not cause unsoundness in copper castings, in the absence of oxygen, unless the solubility in the solid state, which is high, is exceeded. Carbon monoxide is probably soluble in solid and liquid copper to about the same extent and in the absence of oxygen is not harmful. Both

carbon dioxide and nitrogen behave as if they were insoluble in copper. The influence of gases in copper is a complex problem of great importance to producers of copper castings. On the whole, the amount of gas evolved in casting should exactly neutralize the natural shrinkage of the metal on passing from the liquid to the solid state.

PROPERTIES OF COPPER POWDERS

The quality of metal-powder parts depends upon properties of the powders used. The most important of these properties are:

Purity.—Clean particle surfaces insure good contact between particles (needed for optimum mechanical properties). The presence of foreign particles must be held to a minimum, usually specified.

Apparent Density.—Apparent density (sometimes bulk density) is the weight of the unit volume of powder and is of particular importance in the pressing operation. The lower the value, the greater the volume needed for a part of given size.

Compressibility.—Compressibility (ratio of volume of loose powder to volume of compact) is important for both fabrication and end properties and is affected by physical characteristics of powder particles and distribution of particle size. Particle-size distribution affects press feed, dimensional changes during sintering, porosity of the compact, and final attainable density and strength.

Flow Rate.—The ease with which a metal powder can be fed into the die is determined by its flow rate. Low flow rates retard automatic pressing and may require the use of vibrating equipment.

Sintering Properties.—Sintering is usually done either in an inert, reducing, or neutral atmosphere or in a vacuum. The sintering temperature of metal powder is critical; when metal-powder compacts are sintered, either in the solid state or in the presence of a minor portion of a liquid phase, suitable physical and mechanical properties must be obtained within predictable and reproducible dimensions.

Green Strength.—This is the property of the powder to be handled without breakage after compacting and before sintering.

Mechanical and physical properties of sintered parts are closely related to the final density that can be achieved. As the density of a metal powder part increases, strength also increases, and at a density of 100 percent the properties theoretically will be at least equal to those of solid stock.

COMMERCIAL CLASSIFICATION

By Method of Refining

The broadest commercial classification of copper is related to the method by which it is refined. Thus, copper is described as electrolytic or fire refined:

Electrolytic Copper.—Copper which has been refined by electrolytic deposition, including cathodes which are the direct product of the refining operation; refinery shapes cast from metal cathodes; and products of fabricators made therefrom. Usually when this term is used alone, it refers to electrolytic tough-pitch copper without elements other than oxygen being present in significant amounts.

Fire-Refined Copper.—Copper that has been refined by a furnace process only, including refinery shapes and products of fabricators made therefrom. Usually when this term is used alone, it refers to fire-refined, tough-pitch copper without elements other than oxygen being present in significant amounts.

By Method of Casting or Processing

There are three basic classes of commercial copper. They are known as tough-pitch copper, oxygen-free copper, and deoxidized copper; there are several types of each available in refinery shapes and wrought and cast products. Seventeen types are recognized by the American Society for Testing Materials in its "Classification of Coppers" which bears the designation B 224–58. This classification is shown in table 1.

Tough-Pitch Copper.—Copper, that is, either electrolytically refined or fire refined, cast in the form of refinery shapes, and that contains a controlled amount of oxygen for obtaining a level set in the casting. The term is also applicable to the products of fabricators' products made therefrom.

The types of tough-pitch copper designated by the American Society for Testing Materials are: Electrolytic tough-pitch (ETP); fire-refined, high-conductivity tough-pitch (FRHC); fire-refined, tough-pitch (FRTP); arsenical tough-pitch (ATP); silver-bearing, tough-pitch (STP); silver-bearing, arsenical tough-pitch (SATP); and casting (CAST). ETP, FRHC, and STP are high conductivity coppers and the only types used for casting wire bars (table 1).

Tough-pitch copper can be cast with a level, set by adjusting the oxygen content between normal limits of 0.02 to 0.05 percent. Because of its adaptability to high-tonnage melting and casting techniques it is the least expensive kind to produce and has long been the standard variety for producing wire, rod, plate, sheet, and strip. The presence of oxygen affects the

mechanical properties to some degree, while rendering certain impurities insoluble by converting them to oxides. The scavenging effect is beneficial both to high conductivity and to low-annealing temperature.

Oxygen-Free Copper.—Electrolytic copper that is free from cuprous oxide, produced without using metallic or metalloidal deoxidizers. By extension, the term is also applicable to fabricated products. The types of oxygen-free copper are as follows: Oxygen-free without residual deoxidants (OF), oxygen-free phosphorus bearing (OFP), oxygen-free phosphorus and tellurium bearing (OFPTE), oxygen-free silver bearing (OFS), oxygen-free tellurium bearing (OFTE). The OF and OFS types are high-conductivity coppers.

Oxygen-free copper is produced either by melting selected cathodes in the presence of carbon or carbonaceous gases and then casting in a reducing atmosphere or by coalesence of specially prepared and treated cathodes under heat and pressure. In the former method, the impurities are alloyed by the melting operation, resulting in higher annealing temperatures than are encountered with the tough-pitch variety. The net change in conductivity is negligible, however, since the tendency for loss caused by increased solubility of impurities can be offset by gains from elimination of cuprous oxide and regulation to allow partial oxidation of some of the impurities, notably iron. It can also be modified during production by additions of silver, phosphorus, and so forth.

Coalesced copper (PDCP) is made by compressing specially prepared granular cathodes in a reducing atmosphere while hot and then without previous melting by extruding through a die into commercial shapes. The impurities mechanically trapped in the original cathodes are physically dispersed in unalloyed form in the extruded products. As a result, oxygen-free copper produced by this method has a conductivity advantage of several tenths of a percent, and a low-annealing temperature. It is extruded to sizes for mill finishing and thus is not available in many of the standard dimensions associated with refinery shapes produced by casting. Alloying additions are also precluded by the nature of operations. The extrusion operation provides an excellent surface, free from laps and slivers which may act as points of weakness in insulated coatings. Consequently, a large proportion of coalesced copper is used for manufacturing conductors into wire, flatwire, bar, and strip. The products are also characterized by high-residual ductility, resistance to hydrogen embrittlement, and suitability for metal to glass seals.

Deoxidized Copper.—Copper cast into refinery shapes, freed from cuprous oxide by

TABLE 1.—*Classification of copper*

Class and type	ASTM designation	Forms in which available							
		From refiners				From fabricators			
		Wire bars	Billets	Cakes	Ingots and ingot bars	Flat products	Pipe and tube	Rod and wire	Shapes
Tough pitch:									
Electrolytic	ETP	X	X	X	X	X	X	X	X
Fire refined, high conductivity	FRHC	X	X	X	X	X	X	X	X
Fire refined	FRTP	------	X	X	X	X	X	------	X
Arsenical	ATP	------	X	X	------	X	X	X	------
Silver bearing	STP	X	X	X	X	X	X	X	X
Silver-bearing arsenical	SATP	------	X	X	------	X	X	------	------
Casting	CAST	------	------	------	X	------	------	------	------
Oxygen free:									
Without residual deoxidants	OF	X	X	X	------	X	X	X	X
Phosphorus bearing	OFP	X	X	X	------	X	X	X	------
Phosphorus and tellurium bearing	OFPTE	------	X	------	------	------	------	X	X
Silver bearing	OFS	X	X	X	------	X	------	X	X
Tellurium bearing	OFTE	------	X	------	------	------	------	X	X
Deoxidized, phosphorized:									
High-residual phosphorus	DHP	X	X	X	------	X	X	X	X
Low-residual phosphorus	DLP	------	X	X	------	X	X	X	X
Silver bearing	DPS	------	X	X	------	X	X	------	X
Arsenical	DPA	------	X	------	------	X	X	------	------
Tellurium bearing	DPTE	------	X	------	------	------	------	X	X

Source: ASTM Standards 1961, pt. 2, Non-Ferrous Metals, p. 261.

using metallic or metalloidal deoxidizers. The term is also applicable to fabricated products. The types of deoxidized copper are: Phosphorized, high-residual phosphorus (DHP), phosphorized, low-residual phosphorus (DLP). phosphorized, silver bearing (DPS), phosphorized arsenical (DPA), and phosphorized, tellurium bearing (DPTE).

Deoxidized coppers (DLP and DHP) are produced by deoxidizing with sufficient phosphorus to insure combination with all of the available oxygen. The high-conductivity type, DLP, must contain very little residual phosphorus, and the total content rarely exceeds 0.011 percent of phosphorus. Greater amounts are intentionally added to produce type DHP which may contain 0.015 to 0.040 percent phosphorus, resulting in a conductivity range of 80 to 90 percent I.A.C.S. A significant increase in annealing temperature is also obtained which permits soldering of hard-drawn products with minimum softening. The residual phosphorus is most beneficial in preventing absorption of oxygen by the copper during hot-working and annealing operations, thus avoiding embrittlement during those subsequent processes which require heating in a reducing atmosphere. It is especially important to provide a residual deoxidizer in

material to be welded by gas, carbon-arc, or metal-arc methods where both embrittlement and porosity would result from oxidation. Consequently, the quantity of phosphorized copper used for tube manufacture alone exceeds that of all other oxygen-free or deoxidized types combined.

TERMS RELATING TO SPECIFIC KINDS OF COPPERS

The American Society for Testing Materials (ASTM Standards 1961) has adopted descriptive terms for identifying the principal types of cast and processed coppers:

High-Conductivity Copper.—Copper which, in the annealed condition, has a minimum electrical conductivity of 100 per cent I.A.C.S. as determined in accordance with ASTM methods of test.

Casting Copper.—Fire-refined, tough-pitch copper usually cast from melted secondary metal into ingot and ingot bars only, and used for making foundry castings, but not wrought products.

Phosphorized Copper.—General term applied to copper deoxidized with phosphorus. The most commonly used deoxidized copper.

High Residual Phosphorus Copper.—Deoxidized copper with residual phosphorus present in amounts (usually 0.013 to 0.040 percent) generally sufficient to decrease appreciably the conductivity of the copper.

Low Residual Phosphorus Copper.—Deoxidized copper with residual phosphorus present in amounts (usually 0.004 to 0.012 percent) generally too small to decrease appreciably the conductivity of the copper.

Arsenical Copper, Phosphorous Bearing Copper, Silver Bearing Copper, Tellurium Bearing Copper.—Copper containing the designated element in amounts as agreed upon between the supplier and the consumer.

Any of these alloyed coppers can be produced as tough-pitch, oxygen-free, or deoxidized varieties.

The most important classification for wrought forms of commercial copper and copper alloys is based on composition, the forms being designated by long established names. Commonly accepted trade terms for the different kinds of copper, and their nominal chemical compositions, are as follows:

Accepted trade term:	Co position, Percent
Electrolytic tough pitch (ETP).	99.90 Cu–0.04 O
Phosphorized, high-residual phosphorus (DHP).	99.90 Cu–0.02 P
Phosphorized, low-residual phosphorus (DLP).	99.90 Cu–0.005 P
Lake_____	Cu–8 oz./ton Ag
Silver bearing (10–15)____	Cu–10 to 15 oz./ton Ag
Silver bearing (25–30)____	Cu–25 to 30 oz./ton Ag
Oxygen free (OF), no residual deoxidants.	99.92 Cu (min.)
Free cutting, lead bearing_	99.0 Cu–1 Pb
Free cutting, tellurium bearing.	99.5 Cu–0.5 Te
Free cutting, selenium bearing.	99.4 Cu–0.6 Se
Cadmium copper_____	99.0 Cu–1 Cd
Chromium copper_____	Cu–0.4 to 1.2 Cr
Beryllium copper_____	Cu–1.8 to 2.05 Be

FORMS

ASTM defines refinery shapes and fabricators' products as follows:

Refinery Shapes

Wire Bar.—Refinery shape for rolling into rod (and subsequent drawing into wire), strip or shape.

Approximately 3½ to 5 in. square in cross-section, usually from 38 to 54 in. in length and weighing from 135 to 420 lb. Tapered at both ends when used for rolling into rod for subsequent wire drawing and may be unpointed when used for rolling into strip. Cast either horizontally or vertically.

Cake.—Refinery shape for rolling into plate, sheet, strip, or shape. Rectangular in cross section of various sizes. Cast either horizontally or vertically, with range of weights from 140 to 4000 lb or more.

Billet.—Refinery shape primarily for tube manufacture. Circular in cross-section, usually 3 to 10 in. in diameter and in lengths up to 52 in.; weight from 100 to 1500 lb.

Ingot and Ingot Bar.—Refinery shapes employed for alloy production (not fabrication).

Both used for remelting. Ingots usually weigh from 20 to 35 lb and ingot bars from 50 to 70 lb. Both usually notched to facilitate breaking into smaller pieces.

Cathode.—Unmelted flat plate produced by electrolytic refining. The customary size is about 3 ft. square and about ½ to ⅝ in. thick, weighing up to 280 lbs.

Copper Powder.—Finely divided copper particles produced by electrodeposition.

Fabricator Products

Wire.—A solid section, including rectangular flat wire but excluding other flat products, furnished in coils or on spools, reels, or bucks. Flat wire may also be furnished in straight lengths.

Tube.—A hollow product of round or any other cross-section, having a continuous periphery.

Pipe.—Seamless tube conforming to the particular dimensions, commercially known as "Standard Pipe Sizes."

Shape.—A solid section, other than rectangular, square or standard rod and wire sections, furnished in straight lengths.

Shapes are usually made by extrusion but may also be fabricated by drawing.

Flat Products.—A rectangular or square solid section of relatively great length in proportion to thickness.

Included in the designation "flat product" depending on the width and thickness, are plate, sheet, strip, and bar. Also included is the product known as "flat wire."

Rod.—A round, hexagonal, or octagonal solid section. Round rod for further processing into wire (known as "hot-rolled rod," "wire-rod," or "redraw wire") is furnished coiled. Rod for other uses is furnished in straight lengths.

Copper Powder.—Finely divided copper particles produced by (1) high velocity atomization of molten copper with stream of compressed gas, steam, or water; (2) gaseous reduction of finely divided oxides; and (3) precipitation from solutions.

Grades and Specifications

The various types of copper are graded by the American Society for Testing Materials principally according to their chemical composition. Specifications adopted for these coppers are as follows:

ASTM Designation:

B4–42—Lake copper wire bars, cakes, slabs, billets, ingots, and ingot bars (must originate on the northern peninsula of Michigan).

(a) *Low Resistance Lake Copper.*—minimum purity of 99.900 percent, silver being counted as copper.

	Resistivity, max., international ohm (meter, gram)
Wire bars_____	0. 15328
Cakes, slabs, and billets:	
for electrical use_____	. 15328
for other uses_____	. 15694
Ingots and ingot bars_____	. 15694

(b) *High Resistance Lake Copper.*—minimum purity of 99.900 percent, silver and arsenic being counted as copper; resistivity greater than 0.15694 international ohm (meter, gram) at 20° C.

B5–43—Electrolytic-copper wire bars, cakes, slabs, billets, ingots, and ingot bars—minimum purity 99.900 percent, silver being counted as copper.

	Resistivity, max., international ohm (meter, gram)
Wire bars_____	0. 15328
Cakes, slabs, and billets:	
for electrical use_____	. 15328
for other uses_____	. 15694
Ingots and ingot bars_____	. 15694

B115–43—Electrolytic cathode copper—minimum purity of 99.90 percent, silver being counted as copper. The copper shall have a resistivity not to exceed 0.15328 ohms (meter, gram) at 20° C (annealed).

B170–59—Oxygen-free electrolytic copper wire-bars, billets, and cakes—minimum purity of 99.95 percent, silver being counted as copper. Resistivity not to exceed 0.15328 ohms (meter, gram) at 20° C (annealed). Test samples shall be free from cuprous oxide as determined by microscopic examination at 75 X magnification. Prepared wire samples, annealed at 800° to 875° C for 20 min in hydrogen, shall withstand four 90° bends (2 each in opposite directions) without fracture.

B72–60—Fire-refined casting copper—must conform to the following requirements as to chemical composition:

	Grade A, percent	Grade B, percent
Copper plus silver minimum	99. 75	99. 50
Arsenic _____maximum__	0. 0075	0. 10
Antimony_____do____	. 012	. 012
Bismuth_____do____	. 003	. 003
Iron_____do____	. 010	. 010
Lead_____do____	. 100	. 30
Nickel_____do____	. 10	. 10
Oxygen_____do____	. 10	. 10
Selenium_____do____	. 040	. 040
Sulfur_____do____	. 01	. 01
Tellurium_____do____	. 014	. 014
Tin_____do____	. 025	. 05

B216–49—Fire-refined copper for wrought products and alloys—must conform to the following requirements as to chemical composition:

	Percent
Copper plus silver_____minimum__	99. 88
Arsenic_____maximum__	. 012
Antimony_____do____	. 003
Selenium plus tellurium_____do____	. 025
Nickel_____do____	. 05
Bismuth_____do____	. 003
Lead_____do____	. 004

COPPER BASE ALLOYS

Nature

Copper-base alloys comprise a series of alloys having at least 40 percent copper with the amount of copper not less than that of any other constituent. Theoretically, adding another element to copper makes the mixture an alloy, however small the addition. But, in practice, copper is considered commercially pure if it is not less than 99.88 percent, silver being counted as copper. When other elements are included with copper in small enough amounts so that they produce certain desired qualities without changing the basic characteristics of the copper, the result is a modified copper with a copper content of neither less than 99.3 percent nor more than 99.88 percent, silver again being counted as copper. All

other combinations in which copper is the chief constituent are copper-base alloys.

There are four traditional copper-base alloys, namely, brass, bronze, nickel-silver, and cupro-nickel. Nomenclature for many compositions may be confusing under the original concept that brasses are copper-zinc alloys and bronzes are principally copper and tin. Modern metallurgical developments, however, have resulted in important new alloys and necessitated revision and elaboration of many of the original simple definitions.

Properties

WROUGHT ALLOYS

Brass.—The most widely used and best known copper-base alloys are the brasses. Copper and zinc together form a complete series of solid solutions. As zinc is added to copper, tensile properties increase, electrical and thermal conductivities decrease, and there is some diminishing of corrosion resistance. Brasses are commonly used in applications where it is desired to improve some specific characteristic of copper while sacrificing only those qualities that are unimportant to the particular application. In general, the brasses offer mechanical properties superior to those of copper with some loss in the electrical and thermal conductivities. For certain mill products and manufactures, brasses are selected instead of copper because of the lower cost.

There are two broad classifications of wrought copper-zinc alloys, one containing 64 to 99 percent copper, consisting of a single phase and known as alpha brasses; the other containing 55 to 64 percent copper, containing two phases and known as alpha-beta brasses. The alloys of copper and zinc containing less than 55 percent copper, owing to predominance of the beta phase, are brittle and of no commercial significance.

Alpha brasses are exceptionally ductile and malleable at room temperature and can be cold worked by any of the commercial methods such as deep drawing, spinning, stamping, forming, cold rolling, cold heading, flaring, and upsetting. Their hardness is increased by cold work, the degree of hardness being dependent on the amount of cold work and the copper content of the alloy. Alpha brasses containing more than 85 percent copper have work-hardening properties similar to those of copper, and because of this property these brasses are used extensively for applications requiring successive drawing operations without intermediate anneals. After cold-working operations, the brasses can be rendered mallea-

ble or ductile again by heat treatment at temperatures ranging from 700° to 1,400° F, depending upon the properties desired. Alpha brasses are single-phase alloys; they are not susceptible of hardening by heat treatment.

Those alpha brasses containing between 64 and 80 percent copper possess relatively poor hot-working properties. In order to hot roll or hot forge these alloys successfully the utmost care must be taken to keep lead, a natural impurity of most zinc, to a trace. Brasses of this group are best hot worked at temperatures in excess of 1,350° F; the best results are obtained if all hot working is done within the range of 1,350° to 1,550° F. The alpha brasses containing 80 percent or more of copper have hot-working properties comparable to those of copper, which is extremely plastic through a wide temperature range. As in the case of copper, however, care must be exercised to control the lead within very close limits. As a general rule, if these brasses do not contain in excess of 0.01 percent lead, they can be hot forged, hot rolled, or otherwise hot worked without any difficulty.

In the alpha range, tensile properties increase with increasing zinc content, which is also accompanied by a change of color from red through gold. to the green yellows and a progressive decline of electrical and thermal conductivities.

The corrosion resistance of alpha brasses is adversely affected by the same substances and conditions as those affecting the resistance of copper. In some instances they may be corroded by substances, particularly those which might be called active chemical reagents, that do not affect copper to any appreciable extent. An exception to this generality is that in resisting corrosive attack of sulfides the brasses, on the whole, are better than copper; and their superiority in that respect becomes more marked as the zinc content increases. It is of particular interest that in combating the corrosion of sea water, certain of the brasses—for example, 85–15 brass (known as rich low or red brass)—withstand corrosion better than copper.

Brasses containing less than 85 percent copper when exposed to certain media frequently fail in a characteristic manner known as dezincification. Failures of this kind are identified by the spongy areas of copper in the form of layers or so-called plugs on the affected surface. This spongy copper results from the solution of fractions of the alloy in the media and a redeposition of copper by chemical displacement. Arsenic, antimony, and phosphorus are effective dezincification inhibitors in alpha brasses.

Brass that contains less than 85 percent copper may under certain conditions fail by stress-corrosion cracking or, as it is more commonly called, season cracking. Conditions promoting this form of failure are the presence of internal stress, produced by cold-working operations, and exposure to mild atmospheric corrosion. Traces of ammonia in the atmosphere accelerate this type of corrosion. Season cracking can be prevented effectively by relief annealing below the recrystallization temperature.

The following are the more important alpha brasses:

Name:	Copper, percent	Zinc, percent
Gilding metal	95	5
Commercial bronze	90	10
Red brass	85	15
Low brass	80	20
Cartridge brass	70	30

Gilding metal and commercial bronze are extremely easy to cold work, have hot-working properties comparable to copper and can be readily spun, drawn, forged, and upset. These alloys have slightly higher tensile properties and about the same ductility as copper but lower thermal and electrical conductivities.

Red brass and low brass have excellent corrosion-resisting and cold-working properties, frequently being superior to those of copper. Cartridge brass, yellow brass, and intermediates are known generally as the high brasses. These alloys possess the optimum combination of strength and ductility. High brasses all have excellent cold-working properties and can be readily spun, drawn, forged, and upset.

The alpha-beta brasses—that is, those containing from 64 to 55 percent copper—are much easier to hot work than those in the alpha range, the ease of hot working increasing as the copper content decreases. These brasses become increasingly difficult to cold work as the copper content decreases, and those containing less than 58 percent copper are unsuited for any cold-working operations. Like the alpha brasses, the alpha-beta brasses can be rendered soft after cold-working operations by annealing within the temperature range from 700° to 1,400° F, depending upon the properties required. The alpha-beta brasses, however, can be hardened slightly by quenching from the annealing temperature. The hardening is produced by formation of a greater amount of beta in the alloy than would be produced by slower cooling.

The alpha-beta brasses possess the highest tensile properties and the lowest ductility of any of the copper-zinc alloys. Both of these properties are affected by the ratio of the beta phase to the alpha phase. Alloys of the lowest

copper content, because of the greater percentage of beta phase, are the strongest and least ductile. The alpha-beta brasses with the higher copper content approach the alpha brasses containing 64 percent copper in ductility and strength.

Lead is added to brass primarily to improve its machineability; it also definitely improves shearing, blanking, and piercing operations. Lead is virtually insoluble in alloys of copper and zinc and, when present, occurs as finely divided and distributed metallic particles. This uniform dispersion throughout a brass alloy causes the chips to break off, and very little heat is transmitted to the edge of the cutting tool during machining. With leaded brasses, friction between the chip and the tool is reduced to a minimum while, with copper-zinc alloys, the chips are long and tough, tool wear is high, and lubrication problems are difficult.

The presence of lead in brasses does not appreciably influence the mechanical strength or corrosion resistance of the parent alloy, but it drastically reduces the ease of flaring, upsetting, cold heading, and bending operations. Lead in alpha brasses renders them hot-short and unsuitable for fabricating by hot-working methods. The alpha-beta brasses containing between 55 and 60 percent copper can be hot rolled if the lead does not exceed 1 percent and can be successfully hot forged with up to 2 percent lead. If greater amounts of lead are present, cracking in forging may occur. As the copper content increases beyond 60 percent, less lead can be tolerated. When the limit of the beta phase is reached, between 63 and 64 percent copper, lead must be kept to a trace if hot-working properties are to be retained.

Bronze.—Tin is the essential alloying element in the phosphor bronze group of alloys. The maximum limit for tin is about 10 percent in alloys that are to be cold worked, and up to this amount the copper-tin alloys are of alpha grain structure. Addition of tin to copper increases the strength and hardness at a rapid rate and reduces the melting point, density, and electrical and thermal conductivities. Tin is added to some of the brasses in amounts of about 1 percent for improving resistance to corrosion by dezincification in sea water, as in admiralty brass, naval brass, and manganese bronze. Tin is particularly beneficial in providing the high-elastic limit, resilience, and endurance strength characteristics of springs made from phosphor bronze.

The copper-silicon alloys or silicon bronzes are essentially alloys of copper and silicon containing a third constituent usually from 0.25 to 1.25 percent of one of four elements—tin, manganese, zinc, or iron. These alloys are divided into two general types, grades A and B. Grade A silicon bronzes are those containing the maximum of silicon (2.25–3.25 percent) and of the third constituent and are used in those applications requiring the highest tensile properties in combination with a resistance to corrosion equal to or better than that of copper. Grade A alloys possess welding characteristics similar to those of the mild steels. Grade B silicon bronzes contain smaller amounts of silicon (0.90–2.25 percent) and of the third constituent and are characterized by unusually good cold-working properties in combination with tensile properties comparable to 70–30 brass, corrosion resistance similar to that of copper, and weldability almost equal to that of grade A alloys.

Aluminum bronzes are high-copper alloys containing between 4 and 10 percent aluminum. Small amounts of iron, nickel, silicon, and manganese are frequently added to the alloys of higher aluminum content to increase strength and hardness. Industrial aluminum bronzes are of two general types: The alpha or single phase alloys—often referred to as homogeneous alloys; and the alpha-beta or two-phase alloys—known commercially as duplex bronzes. Under equilibrium conditions, 9.8 percent of aluminum is soluble in copper before the beta phase appears, but in commercial practice equilibrium conditions are practically never reached, and alloys containing more than 7.5 percent aluminum usually exhibit two phases.

The alpha aluminum bronzes possess excellent cold-working properties. They also have good hot-working properties and can be readily hot forged, rolled, and extruded. They are most plastic within a temperature range of 1,450° to 1,650° F. Their hot plasticity increases as the aluminum content increases and, conversely, their cold-working properties decrease with a corresponding increase in sensitivity to work hardening. The annealing characteristics are similar to those of the alpha brasses, and softening of work-hardened alloys can be accomplished by annealing between 800° and 1,400° F, depending upon the properties required.

The duplex (two-phase, alpha-beta) aluminum bronzes have excellent hot-working properties through a much wider range than the alpha bronzes. They can be extruded and hot forged into very intricate shapes. Their hot-working properties compare favorably with those of the alpha-beta brasses but they can be cold worked only lightly.

All aluminum bronzes possess good resistance to scaling or oxidation at elevated temperatures, being better in this respect than any of the other copper-base alloys. The resistance to scaling or oxidation increases with the alumi-

num content. The corrosion resistance of these alloys is due to formation of aluminum oxide (Al_2O_3) on exposed surfaces. This film is very resistant to attack by mineral acids, but it is soluble in alkalies. Thus, aluminum bronzes have been widely used in acid environments, but offer only moderate resistance to the attack of strong alkalies. In general the wrought aluminum bronzes are selected for those applications requiring high-tensile properties, good corrosion resistance, strength, and wear-resisting qualities.

Cupronickels.—Nickel is the major alloying element in the cupronickels. The most evident effect is the change in color of the alloy as the nickel content increases. The color of cupronickel is almost nickel white. The addition of nickel to copper increases the strength, hardness, and modulus of elasticity but lowers the electrical and thermal conductivities substantially. Although nickel and copper are mutually soluble in all proportions in the solid state, 30 percent nickel is usually the maximum content permissable for alloys to be manufactured with brassmill equipment. At more than 30 percent nickel the melting points and annealing temperatures reach values that are too high for the furnaces, and the resistance to cold work becomes too great for customary brassmill reduction schedules. Cupronickels are particularly suitable for service at elevated temperatures.

Minor Additions.—Manganese is added to some of the wrought copper alloys, principally as a minor or secondary alloying element. In manganese, aluminum, and silicon bronzes, it improves the grain structure. In cupronickels and nickel-silvers, small amounts are added to improve the metal quality of mill shapes for working operations where its presence is permitted in the finished products.

Phosphorus is confined almost entirely to the phosphor-bronzes, where it is present in amounts to 0.50 percent. It serves as a deoxidizer, and the residual phosphorus content increases the mechanical properties. In admiralty and aluminum brass, phosphorus is used as one of the inhibitors of dezincification. It reduces electrical conductivity substantially, but when used in combination with a small amount of nickel to form nickel phosphide, the phosphide can be precipitated by heat treatment to increase strength, and because it is not in solid solution, the phosphorus ceases to affect conductivity appreciably.

Iron used as a minor alloying element in manganese, aluminum, and silicon bronzes adds strength and makes a finer-grained alloy. Small amounts added to cupronickels improve resistance to corrosion by sea water.

Arsenic is added to the extent of a few hundredths of a percent as an inhibitor of dezincification corrosion in admiralty and aluminum brass.

Cadmium in amounts to 1 percent is added to copper to increase strength and wearing qualities with less sacrifice of electrical conductivity than when other alloying elements are used to obtain the same strength. This combination of properties makes copper-cadmium alloys especially suitable for trolley wires, long-span power lines, and similar applications.

Beryllium is added to copper in amounts to about 2 percent to obtain precipitation-hardening alloys of strength and hardness far beyond any of the other copper alloys. The commercial copper-beryllium alloys after a 2- or 3-hour heat-treatment at 1,450° to 1,500° F, followed by a water quench to return the alpha structure, are malleable and can be drawn, stamped, or cold worked. After such working or cold working, and in order to establish phase equilibrium with attendant high mechanical properties, parts are subjected to an aging anneal at 525° to 575° F for varying periods of time, dependent upon the physical dimensions of the part and the properties required. It is common practice to add a third constituent; cobalt is frequently added to obtain high-temperature stability, and nickel is introduced as a minor constituent to retard grain growth.

CASTING ALLOYS

The properties of brass and bronze castings are dependent on the composition of the alloy, the cleanliness of the metal, and the methods of casting. The properties may vary, as the percentages of the major constituents are at or near the minimum or maximum limits permitted in the specifications for the alloy. They may be affected because of the tolerance specified for other elements. The techniques of sand casting, permanent-mold casting, die casting, investment casting, plaster-mold casting, centrifugal castings, and other methods may account for variations found in castings of the same alloy. Test bars cut from a casting produced by one method can have different properties from those obtained from a casting produced by another method.

Strength, ductility, hardness, etc., are subject to variation due to grain size. In general, the coarser the grain the lower will be the tensile strength, elongation, reduction in area, hardness, and impact strength. Grain size is dependent chiefly upon the rate of cooling of the casting, which is influenced by many variables.

Commercial Classification

Copper-base alloys are grouped in two classes according to specifications required in the processes of fabrication. These two classes are wrought alloys and casting or foundry alloys.

WROUGHT ALLOYS

These are made by alloying the various components and casting the alloy into slabs, cakes, or billets for hot or cold working by rolling, drawing, extrusion, or forging. Depending on the alloy, the wrought products comprise such diverse shapes as plate, sheet, strip, bar, rod, wire, and seamless tube. The chief elements alloyed with copper are zinc, tin, lead, nickel, silicon, and aluminum and to a lesser extent, manganese, cadmium, iron, phosphorus, arsenic, chromium, beryllium, selenium, and tellurium. The range of chemical compositions for the wrought alloys of copper is narrower than for casting alloys. Many compositions that can be poured successfully into either sand or permanent molds are entirely unfit for working operations. In rolling, drawing, forging, extrusion, or tube-billet piercing, a high degree of ductility or malleability, either hot or cold, is a prime requisite. Compositions of wrought alloys referred to as standard in the copper- and brass-mill products industry are shown in table 2.

Brass.—This is any copper-base alloy containing zinc as the principal alloying component, with or without smaller quantities of other elements. There are alloys that are definitely brasses by definition, yet they have names that include the word bronze; for example, commercial bronze (90 percent Cu–10 percent Zn). Although it is recognized that these alloys technically are not bronzes, they do resemble bronze in appearance and, because of long usage, terms such as "commercial bronze" have widespread acceptance.

Bronze.—This is a copper-base alloy having tin as the principal alloying constituent. However, the term "bronze" is seldom used alone and the copper-tin alloys have come to be known as phosphor bronze because of the small residual phosphorus content. With a suitable modifier use of the term "bronze" also extends to a variety of copper-base alloy systems, having the principal alloying element other than tin or zinc. These systems include the aluminum bronzes, the beryllium bronzes, and the silicon bronzes.

Nickel-Silver.—Formerly known as German silver, this is a copper alloy containing both nickel and zinc. Its name derives from its predominantly silvery color. Depending on the composition, the color may vary from yellowish white to silvery white.

Cupronickels.—Essentially, these are copper and nickel, the nickel content usually being 10, 20, or 30 percent. Small amounts of iron and manganese may be and usually are added.

Special Alloys.—There are many special and proprietary alloys that are essentially defined by their names; included are aluminum-tin bronze, aluminum-silicon bronze, cadmium bronze, chromium copper, beryllium copper, leaded copper, selenium copper, and others.

CASTING ALLOYS

Mostly brasses and bronzes, these are made essentially from a complex assortment of industrial nonferrous scrap by scientific selection, smelting, and refining processes for sand and permanent-mold castings. The quantities of alloying metals, such as lead, tin, iron, and aluminum are usually greater than in wrought alloys and are a necessary part of the composition. Elements present as impurities are greater in amount than they are in wrought metals but are not considered harmful within the limits of the specifications. Casting alloys are produced by firms known as ingot makers or manufacturers, and most all casting alloys are marketed and consumed in ingot form. In 1945, the American Society for Testing Materials adopted a standard classification of different types of cast copper-base alloys. There are many hundreds of these alloys, and the classification simplifies the terminology of the various types; each class shown generally covers many alloys. This classification bearing the ASTM designation B119–45 and endorsed by the American Foundrymen's Association is reproduced here as table 3.

USES OF COPPER AND COPPER-BASE ALLOYS

The use of copper and its alloys is universal. They are practically synonymous with all things electrical and have a myriad of uses in construction, autos, aircraft, missiles, railroad equipment, marine equipment, appliances, machinery and equipment, scientific instruments, utensils, jewelry, cladding, and many other applications.

Approximately half of all copper consumed is for electrical applications, including power transmission, electronics, and electrical equipment, transportation, and complex weapons systems. The greatest use is for construction (homes, plants, office buildings) and power generation and transmission. Virtually all building wiring is copper. The transmission of power continues to be a major market for copper, even though aluminum is now used for most high-voltage overhead lines; copper still

TABLE 2.—*Composition and forms of wrought copper-base alloys*

Name	Copper	Zinc	Lead	Tin	Other Element	Other Percent	Typical forms [1]
Brasses-nonleaded:							
Guilding, 95 percent	95. 00	5. 00					S,W.
Commercial bronze, 90 percent	90. 00	10. 00					S,R,W,T.
Red brass, 85 percent	85. 00	15. 00					S,R,W,T.
Low brass, 80 percent	80. 00	20. 00					S,R,W.
Cartridge brass, 70 percent	70. 00	30. 00					S,R,W,T.
Yellow brass, 66 percent	66. 00	34. 00					S,W.
Yellow brass, 63 percent	63. 00	37. 00					R,W.
Muntz metal	60. 00	40. 00					S,T.
Leaded brasses:							
Leaded commercial bronze	89. 00	9. 25	1. 75				R.
Leaded tube brass	66. 50	33. 00	. 50				T.
High-leaded tube brass	66. 50	31. 90	1. 60				T.
Low-leaded brass	64. 50	35. 00	. 50				S.
Medium-leaded brass	64. 50	34. 50	1. 00				S,R,W.
High-leaded brass	64. 00	34. 00	2. 00				S,R.
Free-cutting brass	61. 50	35. 25	3. 25				R.
Leaded muntz metal	60. 00	39. 40	. 60				P.
Free-cutting muntz metal	60. 00	39. 00	1. 00				T.
Forging brass	60. 00	38. 00	2. 00				R,F.
Architectural bronze	56. 00	41. 50	2. 50				ES.
Tin and aluminum brasses:							
Inhibited admiralty [2]	71. 00	28. 00		1. 00			P,T.
Naval brass	60. 00	39. 25		. 75			P,S,R,W,T,F.
Leaded naval brass	60. 00	37. 50	1. 75	. 75			R,F.
Manganese bronze	58. 50	39. 20		1. 00	{Mn / Fe	0. 30 / 1. 00	P,R,F.
Aluminum brass [2]	77. 00	21. 00			Al	2. 00	T.
Phosphor bronzes:							
Phosphor bronze	94. 75			5. 00	P	. 25	S,R,W,T.
Do	91. 75			8. 00	P	. 25	S,R,W.
Do	89. 75			10. 00	P	. 25	S,R,W.
Do	98. 70			1. 25	P	. 05	S.
Leaded	93. 70		1. 00	5. 00	P	. 10	S,R.
Free-cutting	87. 90	4. 00	4. 00	4. 00	P	. 10	R.
Cupronickels:							
Cupronickel	88. 35				{Ni / Fe / Mn	10. 00 / 1. 25 / . 40	P,T.
Do	68. 90				{Ni / Mn / Fe	30. 00 / . 60 / . 50	P,R,T.
Nickel-silvers:							
Nickel silver	65. 00	17. 00			Ni	18. 00	S,R,W.
Do	55. 00	27. 00			Ni	18. 00	S,R,W.
Do	65. 00	23. 00			Ni	12. 00	S,W.
Do	65. 00	25. 00			Ni	10. 00	S,W.
Silicon bronzes (Copper-silicon alloys):							
High-silicon bronze	95. 80				{Si / [3] Mn	3. 10 / 1. 10	S,R,W,T,F.
Low-silicon bronze	98. 25				{Si / [3] Mn	1. 50 / . 25	S,R,W,T.
Aluminum bronzes:							
Aluminum bronze	95. 00				Al	5. 00	S,R,T.
Do	92. 00				Al	8. 00	S,R.
Do	90. 25				{Al / Fe	7. 00 / 2. 75	P,S.
Do	82. 00				{Al / Ni / Fe / Mn	9. 50 / 5. 00 / 2. 50 / 1. 00	P,R.
Aluminum-silicon bronze	90. 75				{Al / Si	7. 25 / 2. 00	R,F.

See footnotes at end of table.

TABLE 2.—*Composition and forms of wrought copper-base alloys*—Continued

Name	Nominal composition, percent						Typical forms [1]
					Other		
	Copper	Zinc	Lead	Tin	Element	Percent	
Other copper alloys:							
Cadmium bronze	99.00				Cd	1.00	W.
Chromium copper	99.14				{Cr	.85	R.
					{Si	.01	
Beryllium copper	[4] 97.70				Be	1.90	S,R,W.
Leaded copper	99.00				Pb	1.00	R.
Selenium copper	99.40				Se	.60	R.

[1] S=sheet and strip; R=rod; W=wire; T=tube; P=plate; F=forgings; and ES=extruded shapes.
[2] Arsenic, antimony, or phosphorus: 0.02 to 0.10.
[3] In place of manganese, iron or zinc are also used in comparable amount.
[4] Ni+Co+Fe=0.40.

Source: Butts, Allison B. Copper, The Metal, Its Alloys, and Compounds. Reinhold Pub. Corp., 1954, p. 537.

TABLE 3.—*Cast copper-base alloys*

Class	Addition elements	Remarks
	Copper	
Copper	Not more than 2 percent total of arsenic, zinc, cadmium, silicon, chromium, silver, or other elements.	Conductivity copper castings, pure copper, deoxidized copper, and slightly alloyed copper.
	Brasses	
Red	2 to 8 percent zinc. Tin less than zinc. Lead less than 0.5 percent.	Alloys in this class without lead seldom used in foundry work.
Leaded red	2 to 8 percent zinc. Tin less than 6 percent, usually less than zinc. Lead more than 0.5 percent.	Commonly used foundry alloys. May be further modified by addition of nickel. See ASTM Specifications B62 and B145.
Semired	8 to 17 percent zinc. Tin less than 6 percent. Lead less than 0.5 percent.	Alloys in this class without lead seldom used in foundry work.
Leaded semired	8 to 17 percent zinc. Tin less than 6 percent. Lead over 0.5 percent.	Commonly used foundry alloys. May be further modified by addition of nickel. See ASTM Specification B145.
Yellow	More than 17 percent zinc. Tin less than 6 percent. Less than 2 percent total aluminum, manganese, nickel, iron, or silicon. Lead less than 0.5 percent.	Commonly used foundry alloy.
Leaded yellow	More than 17 percent zinc. Tin less than 6 percent. Less than 2 percent total aluminum, manganese, nickel, or iron. Lead more than 0.5 percent.	Commonly used foundry alloy. See ASTM Specification B146.
High-strength yellow, manganese bronze.	More than 17 percent zinc. More than 2 percent total of aluminum, manganese, tin, nickel, and iron. Silicon less than 0.5 percent. Lead less than 0.5 percent. Tin less than 6 percent.	Commonly used foundry alloys under name of "manganese bronze", and various trade names. See ASTM Specification B147.
Leaded high-strength yellow, leaded manganese bronze.	More than 17 percent zinc. More than 2 percent total of aluminum, manganese, tin, nickel, and iron. Lead more than 0.5 percent. Tin less than 6 percent.	Commonly used foundry alloys. See ASTM Specifications B132 and B147.
Silicon	More than 0.5 percent silicon. More than 5 percent zinc.	Commonly used foundry alloys. See ASTM Specification B198.

TABLE 3.—*Cast copper-base alloys*—Continued

Class	Addition elements	Remarks
	Brasses—Continued	
Tin_____	More than 6 percent tin. Zinc more than tin.	Alloys in this class seldom used in foundry work.
Tin-nickel_____	More than 6 percent tin. More than 4 percent nickel. Zinc more than tin.	Alloys in this class seldom used in foundry work.
Nickel, nickel silver_____	More than 10 percent zinc. Nickel in amounts sufficient to give white color. Lead less than 0.5 percent.	Commonly used foundry alloys, sometimes called German silver.
Leaded nickel, leaded nickel-silver.	More than 10 percent zinc. Nickel in amounts sufficient to give white color. Lead more than 0.5 percent.	Commonly used foundry alloys, sometimes called "German silver." See ASTM Specification B149.
	Bronzes	
Tin_____	2 to 20 percent tin. Zinc less than tin. Lead less than 0.5 percent.	Commonly used foundry alloys. May be further modified by addition of some nickel or phosphorus, or both. See ASTM Specifications B22 and B143.
Leaded tin_____	To 20 percent tin. Zinc less than tin. Lead more than 0.5 percent, less than 6 percent.	Commonly used foundry alloys. May be further modified by addition of some nickel or phosphorus, or both. See ASTM Specifications B61 and B143.
High-leaded tin_____	To 20 percent tin. Zinc less than tin. Lead more than 6 percent.	Commonly used foundry alloys. May be further modified by addition of some nickel or phosphorus, or both. See ASTM Specifications B22, B66, B67, and B144.
Lead_____	Lead more than 30 percent. Zinc less than tin. Tin more than 10 percent.	Used for special bearing applications.
Nickel_____	More than 10 percent nickel. Zinc less than nickel. Less than 10 percent tin. Less than 0.5 percent lead.	Commonly used foundry alloys. Sometimes called German silver or nickel-silver.
Leaded nickel_____	More than 10 percent nickel. Zinc less than nickel. Less than 10 percent tin. More than 0.5 percent lead.	Commonly used foundry alloys. Sometimes called German silver or nickel-silver. See ASTM Specification B149.
Aluminum_____	5 to 15 percent aluminum. To 10 percent iron, with or without manganese or nickel. Less than 0.5 percent silicon.	Commonly used foundry alloys. Some may be heat-treated. May be further modified by addition of some nickel or tin, or both. See ASTM Specification B148.
Silicon_____	More than 0.5 percent silicon. Not over 5 percent zinc. Not over 98 percent copper.	Commonly used foundry alloys. Some are readily heat-treated. See ASTM Specification B198.
Beryllium_____	More than 2 percent beryllium or beryllium plus metals other than copper.	Most of these alloys are heat treatable.

Source: American Society for Testing Materials. ASTM Standards. Pt 2 Non-Ferrous Metals (Specifications) Electronic Materials. Philadelphia Pa., 1958, pp. 186-188.

dominates in high-voltage underground lines. Generation and utilization of electrical energy requires extremely large quantities of copper. Examples are heat exchangers, windings in motors, generators and transformers, magnet wire in control apparatus, and bus bars.

For communications, telephone and telegraph wire and cable are the two greatest uses. Numerous components in radio and television sets and telegraph receiving and sending equipment are made of copper or its alloys, and coaxial cables employ a substantial amount of copper. The excellent electrical conductivity of copper has made it the major metal used in expanding applications of printed circuits.

The noncorrosive quality of copper and its alloys accounts for many uses in construction for roofing products, plumbing goods, builders hardware, and functional decorative applications. Sheet roofing, gutters, downspouts and heads, flashing, fittings, and accessories consume a large amount of copper. Copper and red brass are found frequently in plumbing items such as tank-lever lift arms, toilet tank trim, lavatory fittings, tub fittings, faucets, traps and drains, flush valves, piping and tubing, and other fixtures. Brass and bronze are popular for builders hardware because they are decorative, noncorrosive, and easy to fabricate for such items as locks, door knobs

and knockers, door stops, hinges, bolts, latches, and safety hasps.

The automotive industry accounts for about 9 percent of the copper consumed in the United States, using about 30 to 40 pounds per vehicle. Volume use is in radiators, heaters and defrosters, air conditioning units, bearings, bushings, carburetors, oil lines, strainers, wiring, switches, and plating. Major uses take strip, tube, plates, and wire; secondary uses take rods, bars, and foundry products.

More copper is required for the maze of wiring in electrical systems of airplanes than for any other use in the aircraft industry. Some of the larger planes use as much as 3,000 pounds of copper in electrical systems and mechanical components.

Use of copper is minor, though essential, in missiles. As in aircraft, the major use is in wiring systems.

Railroads use a large quantity of copper in diesel locomotives, passenger cars, and switching and signal devices. The average Pullman car uses almost 2,000 pounds of copper tube in the water supply system, air brakes, heating system, air conditioning unit, condensers, evaporator, and heating coils.

Tankers, cargo ships, pleasure craft, passenger liners, and naval vessels use copper and its alloys in a great variety of marine equipment. Naval brass is used for propeller shafting (because of its strength, torque resistance, machinability, and corrosion resistance); manganese bronze for struts and rudders (high strength and corrosion resistance); silicon bronze for fastenings and water tanks; soft and hard drawn copper for exhaust yokes; fuel lines, water lines, and piping (resistance to corrosion and leaks, workability, long life). The USN nuclear submarine *Skipjack* uses brass in capstans, torpedo tubes, diving gear, and valves; more than 48,000 feet of copper tube in hydraulic, air, and lubricating oil lines; and more than 6,000 feet of cupronickel tube in auxiliary cooling lines, heat exchangers, lavatories, bilge pumps, and salt water service lines.

Manufacturers of appliances such as washing machines, air conditioning units, refrigerators, radio and television sets, and other units specify copper and its alloys for many components requiring materials that are electrically or thermally conductive, corrosion resistant, and durable. Radio and television sets provide the largest market for printed circuits, but there is great potential for their use in appliances, automobile instrument panels, and electronic computers.

Large machinery and equipment, turbines, and heat exchangers use copper or its alloys in many ways. Specific uses are copper for motors, other electrical equipment and tubing; bronze for worm wheels, oil baffles, check valves, and levers; cupronickel for tubing; naval brass, muntz metal, and aluminum bronze for plates, sheets, bars, and weld wire.

Copper is used extensively in watches, clocks, microscopes, projectors, and many types of gages.

Solid copper, brass, and bronze are popular materials in utensils, jewelry, furnishings, and decorative items. They are used in manufacturing such items as: Brass and bronze medallions; solid brass furnishings—towel bars, curtain rods, wall hooks, and soap dishes; copper, brass, and bronze jewelry; cookware; tableware; and bronze and brass plaques and signs.

From data contained in the 1958 Census of Manufactures by the Bureau of the Census, a table compiled by the Copper Division of BDSA, U.S. Department of Commerce reproduced here in table 4 shows end uses of copper and copperbase alloys in the form of bare wire, insulated wire and cable, and other mill shapes and forms. The percentage of total metal consumed by various fabricating industries is of particular interest. The importance of the electrical industry is apparent, as almost 35 percent of total consumption was accounted for by electrical machinery. Consumption by the military, the automotive industry, and for refrigeration equipment is also significant.

Products of the powder-metallurgy process have found wide usage in machine components. Compacted structural parts are used in automotive, railroad, farm, aircraft, marine, and communications equipment and in household appliances, such as washers, dryers, ironers, electric fans, and refrigerators. Metal powder of single elements, or in combination with other metals or nonmetals are processed to make oil-impregnated porous bearings; cams, gears, motor brushes, clutch facings; contact points, filters, filaments, lock assemblies; and many other various shapes.

The number of automotive applications for powder-metal parts has doubled since 1950. Chrysler Corp. and General Motors Corp. now use about 100 different powder-metal parts in their cars; the Ford Co. uses slightly less. Iron powder mixed with copper, or copper and carbon, is used for transmission and engine parts that receive high impact and fatigue loads. Many new · applications are believed feasible as powders and processing methods are designed for strength properties.

TABLE 4.—*End-uses of copper, 1958*

(In thousands of pounds)

Consuming industry	Bare wire (electrical conductor)	Insulated wire and cable	Other mill products	Castings [1]	Total	Percent of total consumption
Metal fabricating industries, etc.[2]	197,280	257,563	1,132,013	214,656	1,801,512	
Ordnance and accessories	(3)	[3] 4,885	40,438	63	45,386	2.5
Fabricated metal products	6,876	2,838	473,975	90,684	574,368	31.88
Cutlery			3,279	109	3,388	0.18
Hand and edge tools, except machine tools, and hand saws	118	177	1,809	364	2,468	.13
Hardware, n.e.c.	1,480	325	56,802	9,246	67,853	3.7
Plumbing fixtures			3,545	[4] 126	3,671	.2
Plumbing fixture fittings and trim (brass goods)			55,765	19,906	75,671	4.2
Heating equipment, except electric	708	1,077	13,222	[4] 1,232	16,239	.9
Fabricated structural iron and steel	(3)	(4)	[5] 1,275	272	1,547	.086
Metal doors, sash and trim		17	1,165	153	1,335	.074
Boiler shop products	123	95	22,920	1,574	24,712	1.31
Sheet-metal work	545	(7)	9,350	207	10,102	.55
Screw-machine products		176	70,437	647	71,260	3.9
Bolts, nuts, screws, rivets and washers			43,792	(7)	43,792	2.4
Metal stampings	2,565	188	70,303	509	73,565	4.08
Miscellaneous fabricated wire products	1,050	201	11,363		12,614	.67
Valves and pipe fittings (except plumbers brass goods)	287	582	108,948	56,334	166,151	9.2
Machinery, except electrical	7,469	16,902	142,711	71,681	238,763	13.25
Steam engines and turbines	(6)	(6)	(6)	7,397	7,397	.41
Internal combustion engines	(6)	[4] 572	4,432	2,516	7,520	.42
Farm machinery and equipment	40	283	3,979	2,003	6,305	.35
Construction machinery	342	1,959	7,040	[4] 4,368	13,709	.76
Mining machinery and equipment	459	445	810	1,429	3,143	.17
Oil field machines and equipment	(6)	[4] 89	1,393	1,377	2,859	.16
Elevators and moving stairways	(6)	[4] 2,109	585	214	2,908	.16
Hoists, cranes, and monorails	76	711	102	716	1,605	.09
Metal-cutting machine tools		403	576	1,558	2,537	.14
Metal-forming machine tools		47	151	1,589	1,787	.10
Special dies and tools	155	(6)	2,041	743	2,939	.16
Metalworking machinery, n.e.c.	1,258	1,858	4,565	2,723	10,404	.58
Food products machinery	(3)	[4] 1,995	1,314	2,900	6,209	.35
Woodworking machinery	94	2,118	95	101	2,408	.13
Paper-industries machinery			3,436	978	4,414	.24
Printing-trades machinery			1,511	1,005	2,516	.14
Special-industry machinery, n.e.c.	82	276	4,022	2,152	6,532	.36
Pumps and compressors	595	1,801	8,126	19,550	30,072	1.67
Ball and roller bearings	(6)	(6)	3,075	1,475	4,550	.25
Blowers and fans	(6)	206	1,899	319	2,424	.13
Power transmission equipment	161	101	1,141	5,538	6,941	.39
General industrial machinery, n.e.c.	373	113	5,098	1,554	7,138	.40
Refrigeration machinery	3,253	946	55,058	2,446	64,703	3.59
Measuring and dispensing pumps		27	846	558	1,431	.08
Service industry machines, n.e.c.	109	111	620	1,218	2,058	.11
Machine shops	472	732	27,796	5,254	34,254	1.90
Electrical equipment	176,597	205,395	224,134	22,712	628,838	34.91
Electrical measuring instruments	1,880	3,890	4,378	572	10,720	.60
Transformers	45,002	32,111	21,478	544	99,135	5.50
Switchgears and switchboards	5,431	5,961	41,740	8,250	61,382	3.41
Motors and generators	59,817	62,746	29,972	1,184	151,719	8.42
Industrial controls	2,133	3,102	7,780	377	13,392	.74
Welding apparatus	1,764	3,306	3,853	2,779	11,702	.65
Electrical industrial goods, n.e.c.	6,652	6,575	3,370		16,597	.92
Household refrigerators	(6)	1,318	[4] 8,801		10,119	.56
Household laundry equipment	(6)	3,004	[4] 610		3,614	.21
Electrical housewares and fans	982	4,014	3,495		8,491	.47
Household vacuum cleaners	1,432	145			1,577	.09
Household appliances, n.e.c.	286	815	4,387	1,204	6,692	.37
Lighting fixtures	566	4,409	9,161	699	14,835	.82
Current-carrying devices	3,490	3,938	28,289	3,050	38,767	2.15
Noncurrent-carrying devices	(6)	(6)	4,450	994	5,444	.30
Radio and TV receiving sets	4,438	4,802	758		9,998	.55
Telephone and telegraph apparatus	5,267	17,461	25,164	31	47,923	2.66
Radio and TV communications equipment	951	9,224	2,415	2,453	15,043	.84
Electronic components, n.e.c.	14,585	10,047	10,024	321	34,977	1.94
Engine electrical equipment	21,921	28,527	14,009	254	64,711	3.59
Transportation equipment	4,350	20,769	189,306	13,184	227,609	12.63
Motor vehicles and parts	1,560	4,628	167,955	4,037	178,180	9.89
Aircraft	374	4,008	659	126	5,167	.29
Aircraft engines and parts	(6)	341	[4] 730	1,049	2,120	.12
Aircraft equipment, n.e.c.	329	558	4,187	718	5,792	.32
Shipbuilding and repairing	(3)	[4] 3,186	7,268	5,980	16,434	.91
Boatbuilding and repairing			1,487	681	2,168	.12
Locomotives and parts	2,087	4,491	6,014	593	13,185	.73
Trailer coaches	(6)	3,557	[4] 1,006		4,563	.25
Instruments and related products	1,988	6,774	27,279	15,542	51,583	2.86
Scientific instruments	(6)	3,810	1,279	596	5,685	.32
Mechanical measuring devices	974	1,473	11,974	12,411	26,832	1.49
Automatic temperature controls	1,014	1,491	14,026	2,535	19,066	1.06
Miscellaneous manufactures			36,170	795	36,965	2.05
Silverware and plated ware			8,341	795	9,136	.51
Needles, pins, and fasteners			27,829		27,829	1.54

[1] Excludes castings produced and consumed in same establishment.
[2] Metal fabricating industries include establishments whose primary activity is manufacturing products fabricated beyond metal mill or foundry level.
[3] Bare wire included with insulated wire and cable.
[4] Data not available for castings produced and consumed in same establishment.
[5] Bare wire included with other mill products.
[6] Not available.
[7] Not applicable.
[4] Between 1 and 1.5 million pounds of insulated wire and cable were consumed.

Source: Copper Division, BDSA, Department of Commerce.

Copper As an Alloying Element

In addition to its role in copper-base alloys, copper is an important constituent of a large number of alloys having a metal other than copper as the principal component.

Aluminum Alloys.—In 1911, the German metallurgist, Wilm, announced development of Duralumin, a series of alloys containing about 4 percent copper, 0.5 percent magnesium, and 0.5 percent manganese. This development marked the real beginning of wrought aluminum alloys. Pure (99.9[+] percent) aluminum, as cast, has tensile and yield strengths of only 9,000 psi and 3,000 psi, respectively. Addition of copper to and beyond the limit of solid solubility strengthens aluminum rapidly. Aluminum will dissolve 5.65 percent of copper at 548° C (1,018° F), the eutectic temperature, but will hold only 0.45 percent at equilibrium at 300° C (572° F). The hardening effect of copper added to aluminum is due to age hardening or precipitation hardening caused by precipitation of very fine particles of $CuAl_2$. The precipitation of this compound to form a two-phase structure was found to produce a stronger alloy than the solid solution, while retaining high ductility.

Although the newest wrought alloys contain copper, it is no longer the principal hardening element. The new alloys are widely used in aircraft construction, but the aluminum-copper-base alloys still dominate the light alloy field.

Ferrous Alloys.—Pure copper and pure iron are miscible in the liquid phase in all proportions. Introducing carbon affects the miscibility, however, so that a molten eutectic iron containing 4.3 percent carbon can dissolve only about 3 percent copper. Ductile alloys can be produced throughout the entire composition range, but copper in commercially important iron-base alloys is limited to approximately 2.5 percent, and is usually lower. About 0.25 percent copper in copper-bearing steel will roughly double the corrosion resistance of the steel in air. Copper increases the depth of hardening after quenching any steel, but its relative effect is much greater in the high-carbon than the low-carbon steels. In cast iron, copper reduced the tendency toward chilling in light sections and at the same time promoted soundness in heavy sections. In addition, copper in cast iron tends to increase strength, improve machinability, and improve resistance to wear, galling, impact, and shock.

Gold-Copper-Silver Alloys.—Copper hardens gold principally by forming solid solutions. Other elements are added to impart certain properties or characteristics. The gold-copper-silver alloys are more commonly used than gold-copper alloys. An 18-carat alloy of 75-percent gold with about equal parts of silver and copper has a yellow color; replacing more of the silver with copper produces a red gold. The 14-carat alloy of 58.35-percent gold, 2.5 to 5 percent copper, and balance of silver is green. Increasing the copper to 20 to 30 percent makes yellow golds, and higher copper gives a reddish color. The 10-carat golds are largely high-copper alloys with various additions to control color. White gold is a series of gold-nickel-copper-zinc alloys containing the requisite amount of gold to make 10-, 12-, and 14-carat alloys. The copper varies from 51.5 to 31.0 percent.

Nickel-Copper Alloys.—Copper is soluble in nickel in both the liquid and solid states in all proportions, and all the alloys have homogenous structures—regardless of the heat treatment. Although all of the alloys are ductile, only Monel and constantan have much commercial utilization. A third alloy, 70 Ni–30 Cu, has a special use in the form of wire or ribbon for magnetic temperature compensation in electrical meters which depend on the flux produced by a constant voltage or by a certain load current for their actuation.

Monel is the most important nickel-copper alloy. It is available in both cast and wrought form. This alloy—67 percent nickel, 30 percent copper, and the balance iron and manganese—is a strong, ductile alloy that can be formed into rod, wire, plate, sheet, strip, and tubing. Ingots are cast of this alloy that weigh as much as 14,000 pounds. The tensile strength varies from 70,000 psi in the annealed state to 120,000 to 140,000 psi in the full-hard condition. The mechanical properties are quite stable with temperatures to 400° C (750° F). Monel is widely used in chemical, pharmaceutical, and marine equipment, laundries, and kitchens because of its pleasing appearance, strength, and resistance to sea water, dilute sulfuric acid, and strong caustics.

Constantan is an alloy of about 45 percent nickel and 55 percent copper [2] that is distinguished by its high electrical resistance and unusually constant temperature coefficient of electrical resistivity. Extensive use is made of this property in electrical instruments.

Silver-Copper Alloys.—The silver-copper system is relatively simple, with a eutectic at 28.1 percent copper which solidifies at 779.4° C (1,435° F). The maximum solid solubility of copper in silver at this temperature is 8.8 percent, decreasing to about 0.1 percent in equilibrium at room temperature. The high-silver, silver-copper alloys can be age hardened. They have been considered to be the classical

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[2] Although more than 50 percent copper, constantan is listed by the *National Bureau of Standards* and by the *American Society of Metals* as a nickel-base alloy.

examples of age hardening by precipitation of finely divided particles with simultaneous changes in physical properties. All of the alloys of silver and copper are ductile. Of the high-silver alloys, the most important is the 92.5 silver-7.5 copper alloy known as sterling silver. The alloy is universally prized for making flatware and objects of art. It can be hardened with a considerable gain in yield strength. Another important alloy is 90 silver-10 copper, or coin silver. It is used for silver coinage in the United States and for electrical contacts when a harder material than pure silver is desired. The eutectic silver-copper alloy (28 percent Cu) is used to some extent as a brazing metal.

Platinum-Copper and Palladium-Copper Alloys.—Both platinum and palladium form continuous series of solid solutions with copper. Although the alloys are potentially useful, little actual demand exists for the binary alloys. The 60 palladium-40 percent copper alloy has some use as electrical contacts. The ternary palladium-silver copper alloys with more than 5 percent copper and not more than 65 percent palladium are age hardenable and are used in dentistry.

Zinc-Copper Alloys.—The binary high-zinc, Zn–Cu alloys are of no great importance commercially. More important are the ternary zinc-copper-aluminum diecasting alloys that are so widely used in the automotive, electrical and other industries, The tendency toward expansion is an important property of these alloys, causing them to fill molds completely.

Other Alloys.—The high-tin, ternary tin-antimony-copper alloys include all of the high-tin babbitt metals. The fatigue strength and the indentation hardness of these alloys increase with increasing antimony and copper. However, the maximum resistance to cracking in service is obtained with about 3 to 7 percent antimony and 2 to 4 percent copper. The bond strength of babbitt metals cast on steel decreases as the copper content is increased from 3½ to 7 percent. It has been shown that the decreased bond strength is associated with precipitation of Cu_6Sn_5 at the interface of the babbitt and the steel, where the compound concentrates during freezing by normal segregation. Tin-lead-antimony-copper alloys are also used for bearings, but are considered inferior to the high-tin alloys. Lead-base bearing alloys often contain copper ranging from 0.2 to 7.4 percent. Pewter may also contain a little copper as a hardener. Copper in amounts up to 4.7 percent may be used in type metals. The zinc-tin bearing metals, britannia metal, and various white casting alloys usually contain from 2 to 11 percent copper.

Copper is added to manganese alloys in amounts from 3.5 to 35 percent to increase elongation. The silver-copper-zinc alloys in which the copper ranges from 0 to 40 percent are widely used as silver solders.

The properties of several binary and more complex alloy systems based on copper have been investigated. Studies were made of the precipitation hardening of copper containing 2.2 and 4.5 percent cobalt, with and without adding 6 to 12 percent manganese. Precipitation hardening is greater in the alloys containing manganese.

Copper-titanium and copper-zirconium alloys were investigated by Dies, who reported that an alloy containing 0.25 percent zirconium retained work-hardness at moderately elevated temperatures better than the conventional silver-bearing copper and had high-electrical conductivity. Copper alloys containing as much as 2 percent titanium with 0.3 to 1.2 percent aluminum were found by Gruhl and Cordier to give considerable response to precipitation hardening, together with comparatively high ductility and good workability, both hot and cold.

A Japanese patent covers a dental alloy of golden color containing 51 to 60 percent copper, 35 to 43 percent zinc, 0.3 to 2 percent silicon, and 0.5 to 5 percent of either iron or cobalt. Another patented material for special applications—namely, for control rods in nuclear reactors—consists of copper containing 5 to 20 percent boron and 25 to 79 percent nickel or chromium.

Copper Compounds

Of the many known copper compounds only a few are manufactured industrially and used on a large scale. In point of tonnage, copper sulfate ($CuSO_4$) is the most important to industry. Anhydrous copper sulfate is a white crystalline substance, but the usual commercial form is the pentahydrate ($CuSO_4.5H_2O$), called blue vitriol, which contains about 26 percent copper. It is highly valued as a fungicide and as a source of minerals for plant and animal life. Copper sulfate is used extensively to activate sphalerite and other sulfide minerals in metallurgical flotation, as a print toner in photography, in dyes, galvanic cells, and antiseptics, and as the raw material for producing the complex copper ammonium compound necessary for making crayon. Other copper compounds and their uses are given in table 5.

TABLE 5.—*Copper compounds and their uses*

Name	Formula	Crystal color	Soluble in—	Typical uses
Basic cupric acetate___	$Cu(C_2H_3O_2)\cdot CuO\cdot 6H_2O$.	Blue_____	Water, ammonium, hydroxide acid.	Catalyst for making organic compounds.
Basic cupric carbonate_	$CuCO_3\cdot Cu(OH)_2$_____	Dark green_	Slightly in water and acid.	Raw material for production of other copper compounds, pigments for paints and ceramics, fungicides, pyrotechnics.
Bordeaux mixture_____	$3Cu(OH)_2\cdot CuSO_4$____	Blue_____	Water, ammonium hydroxide, acid.	Fungicide.
Cuprous chloride_____	$CuCl$_____	White_____	Hydrochloric acid, ammonium hydroxide.	Catalyst in chemical manufacturing, condensing agent for soaps, fats, and oils.
Cuprous cyanide_____	$CuCN$_____	White_____	Hydrochloric acid, ammonium hydroxide.	Electroplating, catalyst, insecticide, antifouling paints.
Cupric chloride_____	$CuCl_2$_____	Yellow brown.	Water, alcohol, ammonium chloride.	Mordant, refining of gold and silver, recovery of mercury from ores, pyrotechnics, photography.
Cupric nitrate_____	$Cu(NO_3)_2$_____	Blue_____	Water, alcohol_____	Electroplating solutions, burnishing iron, preparation of metal catalysts.
Cuprous oxide_____	Cu_2O_____	Dark red___	Hydrochloric acid_____	Production of copper salts, ceramics, electroplating, fungicides, antifouling paints.
Cupric oxide_____	CuO_____	Brownish black.	Acids_____	Rayon and ceramic industries, electrical depolarizer, chemical analysis, catalyst.

BIBLIOGRAPHY

Anaconda American Brass Co. Cu—Copper Metals by Anaconda Pub. B-40, 1961, 199 pp.

Butts, Allison. Copper, The Metal, Its Alloys and Compounds. Am. Chem. Soc. Monograph Series, Reinhold Pub. Corp., New York, 1954, 936 pp.

Copper & Brass Bulletins. Copper & Brass Research Association quart. pub. 1960–1961.

Copper & Brass Research Association. Copper—The Cornerstone of Civilization. 1959, 33 pp.

Iron Age. Copper: Good on All Counts. V. 182 No. 24, Dec. 11, 1958, pp. 126–129.

Kennecott Copper Corp., All About Kennecott—The Story Of Kennecott Copper Corporation. 1961, 31 pp.

Newton, Joseph, and Curtis L. Wilson. Metallurgy of Copper. John Wiley & Sons Inc., New York, 1942, 518 pp.

The Organization For European Economics Co-Operation. Powder Metallurgy. Paris, France, 1955, 309 pp.

Steel. Copper and Its Alloys. V. 143, No. 17, Oct. 27, 1958, pp. 75–90.

U.S. Department of Commerce. Uses of Copper—1952. Pt. 1-Brass Mill Products, 17 pp.; pt. 2-Copper Wire Mill Products, 10 pp.; pt. 3-Brass and Bronze Foundry Products, 10 pp., Business and Defense Serv. Admin., 1955.

Wilkins, R. A., and E. S. Bunn. Copper and Copper Base Alloys. McGraw-Hill Book Co., New York and London, 1943, 355 pp.

CHAPTER 2—HISTORY

PREHISTORY AND EARLY HISTORY OF COPPER

The industrial history of mankind is divisible into two major epochs—a stone age and a metal age. Between these two epochs there was a transition period, when metals were found in their native state and fashioned by hammering into useful implements including weapons and objects of art. Gold and silver were of no industrial value to primitive man, and the use of copper became the link between the ages of stone and metal. Copper was the second metal used by man, gold being the first. The oldest known relics of copper are not necessarily the first that were made, but hammered copper is found among Chaldean remains that go back to 4500 B.C.; and in the Badarian graves of Fayoum in Egypt there is native copper that may be even more ancient. Some writers believe that two millenniums separated the first use of hammered copper from the beginning of true metal culture, about 3500 B.C., when copper was smelted from its ores and cast in a mold. This dates the first knowledge of copper at about 6000 B.C.

The prehistory and early history of copper is traced through archeological, philological, technical, historical, and classical writings concerning the role of copper in changing and advancing civilizations. The sequence of cultural development as divided into the well-defined neolithic, copper, bronze, and iron ages is not confirmed by the study of the early use of the metals. Such a progression was probably true over different spans of time in the ancient Near East, Egypt, and parts of prehistoric Europe, but a definite and uniform succession of metal cultures all over the world cannot be substantiated. The North and South American Indians had not entered the metal age until many centuries after the Egyptians, and as the mineral resources of North America yielded no tin, there was no bronze age there. Southern India and southern and central Africa also skipped the bronze period. Australia, New Zealand, Japan, and the islands of the Pacific passed directly from the use of stone to that of iron.

Mining

Little is known about ancient copper mining although it is certain that both open-pit and underground mining was practiced in antiquity.

The most important copper-ore deposits were in Sinai, Syria, Baluchistan and Afghanistan, Caucasia and Transcaucasia, the Taurus region, Cyprus, Macedonia, Iberia, and Central Europe. The copper in use among the Sumerians of southern Mesopotamia (3500–3000 B.C.) was obtained from Asia Minor which was also the source for supplying the Assyrians much later. The peaks of production in this region fell between 2400–2000 B.C. and 1500–1200 B.C.

Copper was worked in Egypt beginning from Dynasty III (2600 B.C.). It is estimated that about 10,000 tons was produced over 1,500 years. Some copper mines were in the Arabian Desert, but by far the greatest quantity of copper came from mines of the Sinai peninsula. Although these mines were first worked for turquoise, remains of crucibles, slag, and copper objects prove that smelting of ore was practiced in predynastic times. Production at the oldest mines in Wādi Maghara continued (with a break between Dynasties V and XI) to 1750 B.C. From 1600 B.C. onwards, mining was concentrated in the neighboring region of Jabal Serābīt al Khādim, until it ceased about 1200 B.C. when Egypt had come to depend upon imports from Cyprus and Armenia.

Another important copper-mining center was in the Wādī al 'Arab between the Dead Sea and the Gulf of Aqaba. Here mines were worked from the early to the middle bronze age, and again between 1800 and 1300 B.C. by the Edomites. Mining was by the room-and-pillar method; some of the pillars still show thin veins of ore.

In Cyprus copper mining began about 2500 B.C. Both oxide and sulfide ores found here were used in supplying Egypt during Dynasty XVIII (1580–1350 B.C.). The mines were operated successively under the dominion of the Egyptians, Assyrians, Phoenicians, Greeks, Persians and Romans. The English word, copper, was derived from the Greek name of the island of Cyprus, Kypros, through the Latin cuprum. The same deposit is still being exploited, supplying 39,000 tons of copper in 1960.

Copper was used in China as early as 2500 B.C., according to the Shu Ching epics, but no mention is made of the source or the mining of this copper.

In Europe, copper mines belonging to the bronze age are known in Austria, Germany, France, Spain, Portugal, Greece, and Tirol.

In Tirol the mines of Mitterberg seem to have been worked from about 1600 B.C. in the bronze age to the Hallstatt period of the iron age, about 800 B.C. Probably the most important copper deposits in Europe were those of Spain and Portugal, including the outstanding Rio Tinto ore body which is still being exploited. In Britain copper deposits found in Anglesey, Cornwall, Devon, and Shropshire are of historical interest, having been developed by the Romans after the great invasion of A.D. 43.

In North America evidence, discovered by archeologists in pits found on the Upper Peninsula of Michigan and on Isle Royale in Lake Superior, reveals that copper mining and the forming and use of copper implements and weapons was carried on during a prehistoric period extending from about 5000 to 1000 B.C. There are many of these pits in the area, estimates of those on Isle Royale alone are into the thousands. A section of charred log found in one pit and subjected to carbon-dating methods was determined to be 3,000 years old, plus or minus 350 years; another piece from the same pit was tested and found to be 3,800 years old, plus or minus 500 years.

When copper was first mined in Central Africa is not known. Early Portuguese explorers noted production of copper and iron from Katanga in their 18th-century reports. Several of the Rhodesian copper belt and Katanga ore bodies had been mined extensively and mining operations had ceased long before the Europeans arrived. These ancient copper workings and outcrops pointed out by the Africans undoubtedly led to extensive exploration campaigns which resulted in development of the presently known copper deposits.

Metallurgy

Copper metallurgy started when primitive man discovered that certain stones after being hammered looked like gold. This new natural material was worked by the same processes used when working bone, stone, or fibers—hammering, cutting, bending, grinding, and polishing. This is considered the introductory phase to copper metallurgy.

Annealing native copper was the initial phase of true metallurgy. When it was found that copper when hot was much more malleable and easy to shape, tempering or annealing followed by hammering became common practice. Brittle forms of native copper which had resisted shaping could be worked with good results after heating. This discovery must have been made around 5000 B.C., for the technique was common knowledge of the peasant culture that had spread over South-

Western Asia and North Africa by the fourth millennium before Christ.

Reducing oxide ores and melting copper is linked with development of the pottery kiln. It is believed that only a civilization that made well-baked pottery requiring a high-baking temperature would possess the technical equipment that made the reduction of ores possible. With the knowledge of melting copper came the new way of fashioning metal objects by casting. This art also required a knowledge and ability to manufacture crucibles, tongs, and means of developing blast air (blowpipe, bellows). Archeological studies find that both stages had been reached in Near Eastern prehistory. The date of discovery is roughly estimated about 4000–3500 B.C.

The last phase, reduction of sulfide ores, certainly falls in historic times, though its exact beginning is still obscure. The two stages of roasting the sulfide ores, then smelting with charcoal, in different types of furnaces, each suited to one of the stages, was known to the ancient metallurgists. Finds of such specialized furnaces and lumps of semirefined and pure copper both in the Near East and such European metallurgical centers as Mitterberg prove this. The working of pyrites is connected with the invention of bellows, which is dated around 1580 B.C.

Bronze

Use of bronze dates from remote antiquity. Its first use was probably for celts and dagger blades of the second city of Troy, about 2500 B.C. It was known in Crete around 2000 B.C. and by the late Aegean civilization between 1600 and 1000 B.C. Significant styles of utensils, weapons, implements, tools, and engraved ornaments which have been found reflect the levels of art development in particular areas in certain periods. With the invention of hollow casting about the middle of the 6th century before Christ bronze became the most important material of monumental sculpture. The advancement of the art of working bronze throughout the Hellenistic, Roman, Byzantine, and Medieval periods is reflected in bronze statuary, furniture, church doors, and other ornamental and useful articles made in those times and still in existence. Bell founding, an important industry in northern Europe, France, Germany, England, and the Netherlands, began in the early part of the Middle Ages.

Brass

The mention of brass in the Bible and by Homer, Hesiod, Aristotle, and others un-

doubtedly refers to copper or bronze, because brass was not known as such in ancient times. Some references claim that brass was known to the Romans and possibly to earlier cultures, indicating that brass may have been produced by the smelting of cupriferous zinc-ores which could have come from mines in Laurium, Sardinia, and Cyprus.

Brass was produced in the Low Countries beginning about A.D. 300 and it became an important article of commerce. This brass was made by the calamine process in which copper shots were heated with calamine and charcoal. In 1781 James Emerson patented the process for producing brass from copper and zinc metals. However, brass was produced by the calamine method as late as 1850.

GROWTH OF INDUSTRY IN UNITED STATES

The greatest use of copper developed from the phenomenon of transmitting electricity and the invention and progressive improvement of the electric generator. In 1831 Michael Faraday discovered that an electromotive force (voltage) is set up in a conducting wire when it is moved at right angles to or across a magnetic field. With this principle he developed the dish dynamo or electric generator. Development of the electrical industry was dependent upon parallel development of other industries, one of the major ones being the copper industry, which provided copper wire for transmitting electricity and for component parts of generators and motors.

Invention of the telegraph by Samuel F. B. Morse in 1840 was the beginning of the first public service of electricity. In 1850 the first international cable was laid between Dover, England, and Calais, France. Alexander Graham Bell demonstrated the principle of the electric speaking telephone in June 1875. Edison applied for the first incandescent lamp patent November 4, 1879. Many such inventions and discoveries were made—all involving electricity and requiring copper for its transmission.

The electrical industry began to receive public attention in 1885, and the many developments responsible for its progress until 1905 were made possible by expansion of copper production in the United States.

Copper was first produced in the Colonies in 1709 at Simsbury, Conn. In 1705 a vein of copper ore was discovered in Simsbury. In 1712 another vein was exposed in what is now Cheshire. Other undertakings were carried on for a short time in East Hartford, Bristol, and other points. At Cheshire the paying ore lay only in small pockets and the shafts were soon abandoned. More extensive operations were carried on at Simsbury. The oldest mining charter in the country was given in 1709 to the company that worked this mine. For 30 years after 1707 these operators were actively engaged. They reported that $15,000 had been expended by them. The legislature passed several bills designed to encourage the workers. But since the laws of England forbade smelting the ore here, the product of the mine was shipped to England, many consignments being made. Small amounts were smelted here, notwithstanding the prohibition. After 1737, the returns not being satisfactory, the energy of the operators flagged. The deposit has been worked but intermittently since. In 1760 a company was formed in England to procure copper from the Simsbury mine. Two ship loads of ore were sent from New York. One of these was captured by the French; the other sank in the English channel. This ended foreign interest in the deposit. Later, for a short time in 1837 and again in 1857 operations were carried on.

It was not until exploitation of rich ore deposits of the Northern Peninsula of Michigan in the early 1850's that production in the United States exceeded a few hundred tons a year. Thereafter, smelter production increased from 728 tons in 1850 to 8,064 in 1860, 14,112 in 1870, and 30,240 in 1880. In the early 1880's, U.S. industry was dominated by Calumet & Hecla Mining Co. in Michigan, which was producing half of the domestic output at a cost per pound that would be competitive today, because ore ran as high as 20 percent copper. All the Michigan copper, known as Lake copper, was marketed by Calumet & Hecla through a pooling agreement, and the preeminence of Michigan was considered unassailable, particularly by those who financed the Lake development.

The first discovery of copper in Montana was in the Parrot mine at Butte in August 1866, but throughout the seventies the Butte mines were exploited only for their silver output and their silver possibilities. The area became famous for copper production when mining of copper ore began at the Anaconda mine in the early 1880's. Later, the Anaconda Copper Mining Company gained control of most of the properties in the Butte area because of its affiliation with Amalgamated Copper Company.

Another great copper area was slowly being exploited in the Southwest at the same time. Development of the oxidized silver-copper ores of Arizona was financed by the New York metal traders, Phelps Dodge & Co. Thus the Bisbee operation was first acquired by Phelps Dodge and in 1885 was merged with the Copper Queen mine, forming the basis for great mining

operations in Arizona. In addition, the Bingham, Utah, property was brought into operation by 1907, by Utah Copper Co., which was later taken over by Kennecott Copper Corp.

The Lake pool, headed by Calumet & Hecla, made a determined fight against this western competition. Prices were slashed to discourage Montana producers, and by 1886 copper was selling at 10 cents a pound, compared to an average price of more than 20 cents in the previous decade. Western production, however, was scarcely checked, and the Lake pool was forced to admit defeat before the increasing production of Butte copper. In 1887, Butte production alone exceeded that of all Michigan.

The price decline, following increased United States production, was accompanied by rapidly increased consumption by the growing electrical industry. One of the most spectacular corners in the history of market gamblers—the Secretan corner—occurred at this time. By 1887, Secretan, a French financier, had contracted for three-fourths of the world copper and had begun to raise prices. Consumption fell while supplies of newly mined copper and scrap flooded the market. As stocks increased, prices tumbled. Efforts to secure a reduction in the mounting contract deliveries of copper failed, and in March 1889 the market collapsed.

While the industry was recovering from the Secretan debacle and the price still hovered around 11 cents, the ground was laid for a new effort to corner the market—the Amalgamated pool, involving Standard Oil, National City Bank (New York), and the Anaconda Copper Mining Co. Their scheme was much the same as that used by Secretan, and although the new combine did not attain its goal, for much the same reason that Secretan failed, the precarious position in which the industry found itself was saved in part by quiet liquidation of the Amalgamated surplus copper stock holdings during the next 3 years by a London banking firm.

The industry had hardly recovered from the failure of Amalgamated Copper Co. before it embarked on another price-raising attempt. The pool of 1906–07 was helped materially by the rapid increase in consumption and the high level of speculative prosperity. Elimination of the influence of F. Augustus Heinze in 1906 strengthened the position of Amalgamated at Butte. Under Federal mining law, the owner of the apex of a vein had the right to mine the whole vein. In the complicated Butte vein structure, Anaconda and the Amalgamated found themselves facing the possibility of losing most of their ore in a legal battle with Heinze, who owned a number of fractional claims on which his lawyers and engineers found the apex

of the chief Butte veins. In 1906 Heinze was bought out by Amalgamated for 10.5 million dollars; and Amalgamated, in an effort to recoup, curtailed production, forcing prices up. In 1907, with further restriction of production, the price reached 25 cents before the financial panic of that year. The price broke to about half the 1907 peak, and as production from the new porphyries expanded, the price remained near the 13-cent level until 1912.

Increased supplies from Arizona, New Mexico Utah, and Nevada, made possible by large-scale exploitation of the great porphyry deposits, would have broken the market if European consumption had not been at an abnormally high level. German demand in particular was almost insatiable and not only held the price around 13 cents but led to an increase to 18 cents in 1912–13.

The domestic copper industry received tremendous impetus during World War I. Domestic mines supplied 60 percent of the world primary copper during the war, and U.S.-controlled mines in other parts of the Western Hemisphere supplied almost 20 percent more. The United States was the clearing house for world copper during this period, importing nearly all the production of other Western Hemisphere sources and supplying most of the European demand.

The immediate postwar period brought many serious problems. To cope with oversupply, the industry organized the Copper Export Association to operate in the export market under the Webb-Pomerene Act. Huge war stocks were liquidated in an orderly fashion, and the industry was pulled through the acute depression of 1921, while the average price was maintained at slightly above the prewar average. Maintenance of the prewar price was far from satisfactory to an industry with inflated war costs; but, considering that primary consumption in 1919–23 was 8 percent below the 1909–13 period and that productive capacity in the Western Hemisphere had expanded 65 percent above the prewar figure, this was an accomplishment of note. This organization did not include foreign companies.

During the next 8 years, 1922–29, the swelling output of the mines of the Chile Copper Co., Braden Copper Co., and (after 1924) Union Minierè du Haut Katanga was more responsible than any other single factor for keeping copper prices below the prewar level during this period, bringing about the inactivation of the Copper Export Association in 1924. In 1926, Copper Exporters, Inc., was formed under provisions of the Webb-Pomerene Act. This organization included producers of more than 80 percent of the United States mine production, representing

all the outstanding units. With them were associated foreign producers, which, together with the domestic members, controlled more than 90 percent of the world copper output.

The new organization was a means of increasing the dominance of U.S. producers. By eliminating the influence of speculators, who operated chiefly through the London Metal Market, prices could be dictated by New York, since most of the world production and consumption centered in the United States. Whereas the Copper Exporters, Inc., did succeed in fulfilling their first aim—reducing the power of middlemen and eliminating the London influence—they were not as successful in their second stated aim—stabilization of prices—for prices showed a strong tendency to rise.

This increase in prices was not at once apparent, for in 1927, the first full year of the Exporters activity, prices declined. At the same time, mine production decreased, and stocks increased. By 1928, however, the price of copper began to rise, bringing with it increased production of primary and secondary copper. This increase carried through 1929 with the price for that year averaging 18.23 cents per pound, 24 percent more than in 1928, and mine production was at the second highest point in the history of the industry.

The copper industry had just begun to benefit from rapid expansion of this era when the stock-market crash came late in 1929. Production dropped sharply in 1930 and 1931, and by 1932 refinery output was 75 percent below the 1929 level. Copper Exporters, Inc., disbanded in 1933.

Of the 10 copper refineries operating at 94 percent of their total capacity in 1929, 9 were active in 1932 but were using only 25 percent of their capacity because of the depression.

Anaconda, Kennecott, and Phelps Dodge in 1929 controlled 66 percent of the mine output of the United States and 31 percent of the output outside the United States, 48 percent of the total world production. By 1932, however, the combined share of these three companies had declined from 48 to 36 percent of the world total. United States production also declined from 47 percent to 23 percent of the world total.

Another event that had a far-reaching effect on the United States copper industry was passage in 1932 of a 4-cent excise tax on all foreign copper imports. This was equivalent to a 72-percent ad valorem duty based on the average price in 1932 and was practically an embargo. Most imports of ores, concentrates, and other unfinished forms of copper entered under bond for treatment and re-export without payment of the tax. A small portion was imported,

the tax was paid, and the metal that was exported in manufactured form was eligible for a drawback of 99 percent of the tax. Only small quantities of tax-paid copper were retained in the United States for ultimate consumption.

Business activity revived a little by 1933, bringing with it relief for the copper producers. This trend of increased production continued for the next 5 years, except for a slight recession in 1938.

The copper industry broke all production records in the 1939–49 decade. Refinery output soared from 1,126,000 tons in 1940 to 1,502,000 tons in 1943, the peak for the period. Although output decreased quantitatively after 1943, the amount of capacity being utilized averaged approximately 75 percent, which when the excess capacity created by wartime requirements is considered, indicated that the metal-preparation segment of the industry was in a relatively sound economic condition. However, domestic mines were incapable of supplying the production demands of the economy, thereby forcing operators of smelters and refineries to rely more heavily on imports of unrefined copper.

In Arizona in the 1950's the Lavender Pit, San Manuel, Silver Bell, Copper Cities, Pima, and Esperanza mines started production; capacity at the Ray mine was substantially increased by conversion from block caving to open-pit mining; and the Mission and Christmas mine deposits were developed for production. The White Pine mine in Michigan, the Yerington mine in Nevada, and the Kelley and Berkeley mines in Montana were brought into production. United States mine production exceeded all previous records in 1956 (1,104,156 short tons) and almost reached this record in 1957 and 1960; 1958 and 1959 were low production years because of voluntary cutbacks in mine output and strikes.

Increasing demand in the United States and Europe beginning in the latter half of 1954 created a copper shortage, and the price of major producers in the United States rose continually, reaching 46 cents a pound on February 21, 1956, highest in 90 years. Prices quoted by custom smelters in the United States, the London Metal Exchange, and other European markets were even higher, attaining a peak of 55 cents per pound. (See chapter 6.)

A number of substantial copper properties in Canada began producing in the 1950's. Large reserves were developed at the Toquepala, Cuajone, and Quellaveco deposits in Peru and at the El Salvador, Rio Blanco, Mantos Blanco, La Africana, Chuquicamata, and Braden properties in Chile. Additional sources of ore were found in the copper belts of Northern Rhodesia

and the Republic of the Congo. In Europe, large deposits were developed in Yugoslavia, at Majdanpek; the Glogów area of Poland; Ireland; Finland; Bulgaria; and the U.S.S.R. Substantial ore bodies were found in Australia and the Philippines. World mine production increased progressively each year, except for 1958, and in 1960 was 66 percent higher than in 1950.

TECHNOLOGICAL DEVELOPMENTS

Principal technological developments in the United States pertinent to the copper mining and smelting industry during the past 100 years include the following:

1. Application of square-set mining, first developed by Philip Deidesheimer in the early 1860's, which solved the problem of mining the large and fabulously rich, but soft and crumbling, silver sulfide ore body at the Comstock silver mines of Nevada. The system was immediately successful and has since found worldwide application for ground support in mining ores that could not be mined by any other known method.

2. Introduction of a block-caving system at the iron mines in Michigan and the use of slushers for mucking, the forerunner of modern mechanical loading in underground metal mines, both near the close of the 19th century. These two developments were not generally accorded a prominent place in mining history, but their significance appears when viewed from the perspective of later years.

3. Adaption of the Bessemer steel converter to the conversion of copper matte into metal in France in 1880 and in the United States in the late 1880's. This development eliminated the need for dead roasting copper sulfide ore as a prelude to blast-furnace smelting to black copper. Previously, emphasis had been on the treatment of oxide ores, and the worldwide practice was to convert sulfides to oxides by cumbersome methods of roasting; in some cases this was followed by leaching, in others, by blast-furnace smelting. The converter, with concomitant development of electrolytic refining to purify the resultant blister copper and to recover its precious metals and other byproducts, started electrolytic copper in the long, hard, competition to become accepted as equal, if not superior, to Lake copper. The converter also was a major factor in lowering costs so that lower-grade ores could be utilized.

4. The beginning of open-pit mining in 1907 permitted profitable exploitation of huge bodies of low-grade disseminated copper ores in the Southwest that previously had been considered economically unworkable. Advancements in mining technology effecting development of better mining methods, mining and transportation equipment, explosives, and engineering control contributed to increasing output by open-pit mining, accounting for 80 percent of the copper ore and 75 percent of the copper produced in 1960.

5. Rapid development of the flotation process introduced in copper concentrators between 1913 and 1916. This process improved the recovery of sulfide minerals from fine-grained ore fractions. Previously, the practice was to use gravity methods to concentrate the ore, but ore that required fine grinding to liberate minerals from gangue was not recoverable if the average tenor was below the economic limit of direct smelting.

About 1921, the discovery that certain flotation reagents could be used to effect separation of metallic sulfides from each other made available additional tonnages of copper and other minerals in selective mixed ores. Selective flotation can now make satisfactory separations of these minerals if the minerals themselves are not so intimately intergrown as to require excessive comminution to liberate them as discrete particles. With introduction of flotation at the Utah Copper Co. mill after 1914, recovery was increased from 75 percent in that year to more than 90 percent by 1935, despite a general decline in the grade of ore treated. At the Calumet & Hecla Consolidated Copper Co., Inc., mill, flotation was applied only to slime, and part of the indicated improvement in recovery was due to leaching of sand tailings. Up-to-date figures have not been compiled; however, it is safe to assume that since 1935 the percentage of recovery had decreased, owing to the complexities and lower grade of ores being treated. By no means does this detract from the effectiveness of this innovation, but it does point up its limitations.

6. With flotation came a marked change in copper-smelting methods. No longer was the blast-furnace converter cycle adequate, because most of the fine flotation concentrates were blown into the stack, and expensive recycling techniques were required to recover copper from the dust. Again the steel industry made an important contribution to copper metallurgy—the open-hearth steel furnace. This furnace metamorphosed into the copper reverberatory furnace to smelt copper concentrate into matte. To control the sulfur content, partial roasting of fine concentrates to a granular calcine before reverberatory smelting rather than sintering for blast-furnace smelting gained in importance. Later improvements in flotation now make it possible, in many cases, to control the sulfur and iron in concentrates at the mill to eliminate

roasting. And in 1960, Phelps Dodge Corp. initiated the use of reformed natural gas in place of green poles for reduction in the anode furnaces of their smelters in Arizona.

The following chronology indicates the major events important to development of the copper industry from 1850 to 1960, inclusive.

CHRONOLOGY

1850 Michigan mines became principal U.S. copper producers.

1855 First Atlantic cable was laid.

1857 Duty of 5 percent ad valorem, laid on copper ingots in 1846, was removed.

1861 Civil War began.

1861 Duty was placed on copper ingots, etc., 2 cents a pound; on ores, 5 percent; and other copper, 25 to 35 percent ad valorem.

1864 Duty on copper ingots was placed at 2.5 cents a pound until 1869.

1865 Civil War ended. "Greenback" inflation occurred.

1869 Duty on raw copper was raised to 5 cents a pound, 1869 to 1883; "Black Friday" occurred on Wall Street; first transcontinental railway was completed.

1871 Calumet & Hecla consolidated the Michigan mines.

1873 Panic of 1873 occurred.

1875 Rich copper deposits were discovered in Arizona and Montana.

1878 Edison invented incandescent lamp, marking beginning of electrical industry.

1879 Anaconda Silver Mining Co. was organized; copper mining began in Globe-Miami, Ariz.

1880 First electric lamps were installed in New York, December 20.

1883 United States became the leading world copper producer.

1884 Duty on raw copper was reduced to 4.5 cents per pound.

1887 Secretan acquired corner on copper; 85 percent of United States output was controlled.

1888 Montana became leading United States producer.

1889 Secretan corner was broken by flood of new and scrap copper.

1891 Baring Bros. failed because of copper corner, precipitating depression of nineties.

1892 General Electric Co. was formed. New York-Chicago copper telephone line was installed.

1894 Raw copper was placed on "free list." Serious labor troubles occurred.

1896 Business recovered.

1898 Spanish-American War occurred.

1899 Amalgamated Copper Co., capital $75 million, acquired Anaconda.

1900– Was era of consolidations, mergers, and
1905 financial wars for control of rich Montana copper deposits. F. Augustus Heinze threatened Amalgamated-Anaconda mineral rights by securing control of key properties in Butte.

1906 Amalgamated bought out Heinze for $10.5 million, ending 5-year fight for control.

1907 "Rich man's panic" was experienced on Wall Street, October.

1909 Daniel C. Jackling proved economical his idea of mining and processing the low-grade copper ores of the western United States on a scale previously unknown, thereby revolutionizing copper mining in the United States and abroad and making available new and needed copper reserves to meet world requirements.

1912 Braden Copper Co. in Chile began production of low-cost copper.

1914 World War I began. Clayton Act was passed.

1915 Kennecott Copper Corp. was formed.

1916 Chile Copper Co. was in production.

1917 United States entered war; copper was placed under price control at 23.5 cents a pound.

1918 World War I ended. Webb-Pomerene Act was passed.

1919 Copper Export Association was formed.

1920 Postwar inflation affected industry.

1921 Deflation occurred; "buyers' strike" enforced; most copper mines were shut down. Copper Export Association pool removed 200,000 tons of copper from domestic market.

1924 Katanga, Belgian Congo, began volume production. Copper Export Association was disbanded.

1926 Copper Exporters, Inc., was organized as international copper cartel.

1927 Domestic copper production was curtailed, contrasting with upward trend in demand.

1928 Copper stocks were reduced to panic-buying levels.

1929 Panic buying of copper continued; stock-market panic occurred, October; copper prices were pegged at 18.3 cents for nearly a year; Canada became large copper producer.

1931 Copper Exporters, Inc., was dissolved; Rhodesian mines gained in volume production.

1932 Duty of 4 cents per pound was placed on raw copper June 21.

1933 NRA (National Recovery Act) Code for Copper and Brass Mill Products Industry was approved in November.

1934 NRA Code for Copper Industry price was set at 9 cents per pound.

1935 NRA was declared unconstitutional. International Copper Control was formed June 1.

1936 Copper production was curtailed by cartel—South America, Africa.

1937 Copper shortage brought panic buying and almost a 50-percent price increase.

1938 Business recession began.

1939 World War II began September 1. International copper cartel was abandoned.

1940 U.S. Government began buying copper through RFC (Reconstruction Finance Corp.) subsidiary.

1941 United States entered World War II December 8. Copper prices were part under voluntary control

1942 Copper prices were fixed at 12 cents, Connecticut Valley, with a Government subsidy for high-cost mines; Premium Price plan set, February 1.

1945 World War II ended.

1946 Price control was ended November 10. Rapid price increases made Premium Price Plan ineffective.

1947 Premium Price Plan was ended June 30.

1948 Geneva Trade Agreement was made. Excise tax was reduced from 4 cents to 2, pending ratification by Chile. (Tax was suspended during the war and suspension was continued to March 31, 1949).

1949 Chile ratified Geneva agreement in February. Tax was further suspended until June 30, 1950.

1950 Korean conflict began June 26. Defense Production Act was passed. Defense Minerals Administration in the Department of the Interior, National Production Authority in the Department of Commerce, and other agencies were created to carry out provisions of the act.

1951 National Production Authority reinstituted Controlled Materials Plan for copper, steel, and aluminum. Ceiling price on copper was established January 26 by the Office of Price Stabilization. World copper (free world countries) was placed under international allocation and quotas were set by the International Materials Conference.

1952 Chile embargoed copper exports to United States, resulting in dual prices in United States. Kelley mine at Butte, Mont., was started in production.

1953 Free trading resumed on London Metal Exchange August 5 after lapse of 14 years and on the New York Commodity Exchange after a recess of 2 years. Initial production came from Yerington mine in Nevada.

1954 Silver Bell, Lavender Pit, and Copper Cities mines in Arizona and the White Pine mine in Michigan reached production stage. U.S. Government purchased 100,000 tons of Chilean copper.

1955 San Manuel started production. Prices increased rapidly.

1956 Mine production and price of copper reached record high in the United States. Three-year labor contracts between principal producers and unions in United States were negotiated in June. Production was started at the Berkeley pit at Butte, Mont.

1957 San Manuel Copper Corp. and White Pine Copper Co. delivered copper to the Government under DMPA contracts negotiated in 1952. Export controls on copper were relaxed by Bureau of Foreign Commerce.

1958 Excise tax on copper imports was reimposed July 1 at 1.7 cents per pound pursuant to agreement at GATT meetings at Geneva, in 1956.

1959 Principal copper mines, smelters, and refineries were closed by longest strike in history. Operations began at the Esperanza mine in Arizona in March. Production was started in April at the El Salvador mine in Chile.

1960 World mine production reached new high. Scheduled production began at the Toquepala mine in Peru, January 1.

BIBLIOGRAPHY

Aitchison, Leslie. A History of Metals. V. 1 & 2, McDonald and Evans, Ltd., London, 1960, 685 pp.

Benton, William. Copper. Para. in Encyclopaedia Britannica, v. 4, Chicago, Ill., 1956, pp. 40 and 240–246.

Bureau of Mines. Copper. MS 6, 1952, 694 pp.

Butts, Allison. Copper, The Metal, Its Alloys and Compounds. Reinhold Pub. Corp., New York, 1954, 936 pp.

Coghlan, H. H. Roman Mining in Britain. Optima, Anglo-American Corp. of South Africa, Ltd., v. 8, no. 1., March 1958, pp. 8–15.

Copper Development Association. Copper Through the Ages. Kendals Hall, Radlett, Hertfordshire, England, 1953, 68 pp.

Encyclopedia Americana. Copper. V. 4, Am. Corp., New York, 1959, pp. 582–589a.

Forbes, Robert J. Metallurgy of Antiquity. E. J. Brill, Leiden, Holland, 1950, pp. 389.

Lathrop, William G. The Brass Industry in the United States. The Wilson H. Lee Co., New Haven, Conn., 1926, pp. 21–22.

Lemmer, Victory F. Prehistoric Copper Mines on the Shores of Lake Superior. Skillings Min. Rev., v. 48, No. 40, Jan. 2, 1960, pp. 4–5 and 15.

Marcosson, Isaac F. Anaconda. Dodd, Mead & Company, Inc., 1957, 370 pp.

Mellor, J. W. A Comprehensive Treatise on Inorganic and Theoretical Chemistry. Longmans Green & Co., London, 1923, v. 3, p. 2; v. 6, p. 399.

Roast, Harold J. Cast Bronze. Am. Soc. Metals, 1953, pp. 360–431.

Singer, Charles, E. J. Holmyard, A. B. Hall, and Trevor T. Williams. A History of Technology. Oxford Univ. Press, Amen House, London, v. 1 & 2 of 5 v. 1956, pp. 802 and 766.

Norway
Sulitjelma   Sulitjelma Gruber A-S
Orkla   Orkla, Grube-Aktiebolag

East Germany
Mansfeld-
Sangerhausen areas   Government

Poland
Konrad, Leona,
Lubichow   Government

Sweden
Boliden group   Boliden Mining Co.

Yugoslavia
Bor   Government
Majdanpek   Do.

Finland
Outokumpu   Outokumpu Oy

U.S.S.R.

Norilsk   Government
Urals   Do.
Kazakstan   Do.
Uzbekistan   Do.
Armenia   Do.

Japan
Hanoka   Dowa Mining Co., Ltd.
Osarizawa   Mitsubishi Metal Mining Co., Ltd.
Hitachi   Nippon Mining Co., Ltd.
Ashio   Furukawa Mining Co., Ltd.
Besshi   Sumitomo Metal Mining Co., Ltd.

India
Mosaboni   Indian Copper Corp.
Badia   Do.

Philippines
Lepanto   Lepanto Consolidated Mining Co.
Santo Tomas II   Philex Mining Corp.
Bagaycay   Marinduque Iron Mines Agents, Inc.
Sipalay   Do.
Toledo   Atlas Consolidated Mining & Development Co

Turkey
Murgul   Government
Ergani   Do.

Israel
Timna   Israel Mining Industries

Cyprus
Mavrovouni   Cyprus Mines Corp.
Skouriotissa   Do.

Spain
Rio Tinto   Compania Espanola de Minas de Rio Tinto

Africa Uganda
Kilembe   Kilembe Mines, Ltd.

Republic of the Congo
Prince, Leopold   Union Miniere du Haut Katanga
Kamoto, Ruwe,
Musonoi, Kolwezi,
Kambove

Northern Rhodesia
Bancroft   Bancroft Mines, Ltd.
Nchanga   Nchanga Consolidated Copper Mines, Ltd.
Rhokana Corp., Ltd.
Chibuluma   Chibuluma Mines, Ltd.
Mufulira   Mufulira Copper Mines, Ltd.
Roan Antelope   Roan Antelope Copper Mines, Ltd.

Southern Rhodesia
Mangula   Messina (Transvaal) Development Co., Ltd.

Republic of South Africa
Messina   Messina (Transvaal) Development Co., Ltd.

Australia
Peko   Peko Mines, N. L.
Mount Isa   Mount Isa Mines, Lt
Mount Morgan   Mount Morgan, Ltd.

Tasmania
Mount Lyell   Mount Lyell Mining and Railway Co., Inc

SOUTH WEST AFRICA

British Columbia

The Anaconda Company

Britannia

Lynn Lake
Flin Flon

Manitoba
Sherritt Gordon Mines, Ltd.
Hudson Bay Mining & Smelting Co., Inc.

Ontario

Geco
Frood-Stobie,
Creighton, Garson,
Levack, Murray
Hardy, Onaping,
Fecunis,
Longvack

Geco Mines, Ltd.
International Nickel Co. of Canada, Ltd.

Falconbridge Nickel Mines, Ltd.

Quebec

Normetal Mines
Horne
Quemont
Waite Amulet
Chibougamau
Opemiska
Gaspe

Normetal Mining Corp., Ltd.
Noranda Mines, Ltd.
Quemont Mining Corp., Ltd.
Waite Amulet Mines, Ltd.
Campbell Chibougamau Mines, Ltd.
Opemiska Copper Mines, Ltd.
Gaspe Copper Mines, Ltd.

Newfoundland

Tilt Cove

Maritimes Mining Corp., Ltd.

Michigan

Montana
Berkeley, Kelley,
Vein mines
The Anaconda Company

Ahmeek, Allouez,
Centennial, Osceola,
Seneca
White Pine

Calumet & Hecla, Inc.
White Pine Copper Co.

Tennessee

New Mexico
Kennecott Copper Corp.

CHINO

Burra-Boyd   Tennessee Corp.

Utah

Utah Copper   Kennecott Copper Corp.

Nevada

Liberty pit
Yerington

Kennecott Copper Corp.
The Anaconda Company

Mexico
Cananea
Cananea Consolidated Copper Co., S.A.

Cuba
Matahambre
Government

Arizona

Bagdad
Christmas
Copper Cities
Copper Queen-
   Lavender pit
Esperanza
Inspiration
Magma
Miami
Mission
Morenci
New Cornelia
Palo Verde
Pima
Ray pit
San Manuel
Silver Bell

Bagdad Copper Co.
Inspiration Consolidated Copper Co.
Tennessee Corp.

Phelps Dodge Corp.
Duval Sulphur & Potash Co.
Inspiration Consolidated Copper Co.
Magma Copper Co.
Tennessee Corp.
American Smelting and Refining Co.
Phelps Dodge Corp.
         Do.
Banner Mining Co.
Pima Mining Co.
Kennecott Copper Corp.
Magma Copper Co.
American Smelting and Refining Co.

Peru

Quiruvilca
Cerro de Pasco,
Morococha,
Yauricocha
Toquepala

American Smelting and Refining Co.
Cerro de Pasco Corp.

Southern Peru Copper Corp.

Chile

Chuquicamata
Mantos Blancos
El Salvador
El Teniente

Chile Exploration Co.
Empresa Minera de Mantos Blancos
Andes Copper Mining Co.
Braden Copper Co.

CHAPTER 3.—PRIMARY COPPER RESOURCES

The world is plentifully endowed with sources of copper. Mines operating today are equipped and have the capacity to more than fulfill current demand. New discoveries coupled with extensions at mines in many copper provinces of the world during the 1950's greatly increased the known world copper resource. Important copper producing regions of the world are: (1) The western United States, (2) the western slope of the Andes in Peru and Chile, (3) the central African Copperbelt in Northern Rhodesia and the Republic of the Congo, (4) the Ural Mountains and the Kazakhstan region in U.S.S.R., (5) the Precambrian area of central Canada, and (6) the Keweenaw peninsula of northern Michigan. Figure 2 shows location of major copper mines in the world.

GEOLOGY

Deposits of copper are widely distributed, both geographically and geologically. Geologically such ore bodies are found in a variety of host rocks of all ages under greatly varied structural conditions. The periods of their formation range from Precambrian to Recent. Many are characteristically associated both in time and location with igneous activity, while a few occur in sandstones and shales.

Types of Deposits

Following is a general classification of types of copper deposits; it should be recognized that gradations will occur with gradual changes from one physical and chemical environment to another: (1) Lenticular deposits in schistose rocks; (2) Sudbury nickel-copper type; (3) native copper types; (4) bedded deposits in sedimentary rocks; and (5) vein, pipe, and disseminated deposits.

LENTICULAR DEPOSITS IN SCHISTOSE ROCKS

The primary mineralization of these deposits consists of sulfides of iron, copper, and zinc commonly as a replacement of schistose rocks. Schistosity may be the result of widespread metamorphism, but the deposits are along zones of shearing. Such deposits seem to have formed at some depth under conditions that favored deformation by shearing rather than by brecciation or shattering. The solutions that deposited the ores were closely confined to the channels and replaced the sheared rock almost completely to form massive sulfide lenses. The minerals are those of moderate temperature range of sulfide deposition, and the deposits may or may not contain precious metals and zinc or lead in important amounts. Such deposits usually are not closely associated with igneous stocks, and none is in stocks.

The deposits of the deeply eroded Appalachian region belong to this classification, as do some of the deposits on the border of the Sierra Nevada and Coast Range batholiths in California and British Columbia, respectively, where erosion has also been deep. The Tertiary deposits of the Rocky Mountain area, which have relatively shallow erosion, are not of this type.

The primary ores are generally a rather low-grade massive sulfide. Pyrrhotite is present in many of the ores, and iron-bearing sphalerite is in most of them. These minerals are notably absent in most of the Tertiary copper deposits, and depth of formation seems to be a factor in their formation.

Supergene alteration in many deposits has produced a zone of enriched sulfides, usually thin and rich as at Ducktown, Tenn., and Iron Mountain, Calif., but sometimes deep and rich as at Jerome, Ariz.

SUDBURY NICKEL-COPPER DEPOSITS

These are associated with the norite member of the Sudbury irruptive. Generally, the ore bodies occur associated with faults at or near the footwall contact between the norite and the underlying volcanics or in offsets of the norite within the volcanic series. Structurally, faulting appears to be the main ore-localizing feature, with ore bodies occurring in zones of shearing and brecciation close to the norite contact. Ore occurs as disseminated and heavy sulfide types containing pyrrhotite, pentlandite, and chalcopyrite as the principal minerals. Gold and silver occur as tellurides and in the native state. The platinum group metals appear as sperrylite, platinum arsenide, and other rare combinations.

NATIVE COPPER DEPOSITS

Included are those in which the copper occurs in shoots in tops of certain lava flows, in conglomerates interbedded with flows, and in sandstones. The rocks in which the copper occurs are characteristically red owing to the

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presence of ferric oxide. The latter may account for the copper being deposited as native metal rather than as sulfide.

Deposits of this type have been mined at vertical depths of more than 6,000 feet in veins and lodes with no notable change in mineralogy. The ores are generally low-grade concentrating ores, although large masses of copper are present.

The Lake Superior district of Michigan is the only large producing region having deposits of this type, although similar deposits occur in New Jersey, Pennsylvania, Maryland, Alaska, and Northern Canada. For many years native copper also accounted for much of the production at the Corocoro mine in Bolivia.

BEDDED DEPOSITS IN SEDIMENTARY ROCKS

These include contact or pyrometasomatic deposits and those formed by replacement of sedimentary rocks, commonly limestone, outward from fissures. The two classes are not sharply separated. The typical contact or pyrometasomatic deposits are replacements of limestone or dolomite by silicates, oxides, and sulfides. Many contact deposits contain large bodies of andradite (the ferric iron garnet) with which are associated other high-temperature silicates, iron oxides (magnetite and hematite), and sulfides of iron and copper. Many such deposits are in limestone at or near the contact of an intrusive body, usually a stock.

The silicates and oxides formed first and the sulfides later. The sulfides, including copper sulfide, are most abundant at many deposits between the silicate zone and the unreplaced limestone. In some deposits the silicate zone is largely lacking, and the sulfide zone is close to the igneous contact. Contact deposits grade into fissure-replacement and bedded deposits, in which certain beds of limestone are replaced by sulfides outward from fissures with or without the formation of silicates.

Deposits of the contact metamorphic type are widely distributed, especially in the eastern Cordilleran region, but with a few exceptions have yielded relatively little copper. Part of the great deposits at Bisbee, Ariz., and some of the limestone replacement deposits at Bingham, Utah, are examples of such occurrences, and many of less importance have been mined.

With such deposits may be grouped those in limestone far from known intrusive bodies; the chalcocite deposits of the Copper River district in Alaska are outstanding examples.

VEINS, PIPES, AND DISSEMINATIONS

These may be grouped as three types of deposits, each having distinctive features but showing such gradation that a definite separation is neither possible nor desirable. The deposits are similar in mineral composition and differ mainly in the character of the fracturing that has controlled deposition.

Vein Deposits.—Perhaps, the simplest type of deposit and the simplest veins are those that occupy a single fault or fissure. Large copper deposits of this character are uncommon but are approached in the Magma vein of Superior, Ariz., and the Old Dominion vein of Globe, Ariz.

Other districts, such as the Butte district in Montana and the Morenci district in Arizona have complex vein systems in which the larger veins can be mined as units, but in parts of these areas a mass of small veins are mined together.

In general, the veins tend to join and decrease in number in depth, and a complex system of veins may merge into only a few root channels.

Volcanic Pipe Deposits.—These are more or less cylindrical bodies formed by volcanic explosion. The rock within the pipe is usually brecciated, although in some a core of relatively unbrecciated rock is surrounded by an envelope of highly brecciated rock. The ore minerals cement and partly replace the brecciated rock. In some pipes, quartz has replaced most of the rock in a central channel, while in others replacement has been slight.

Volcanic pipes range in size from a few feet or tens of feet to hundreds of feet in diameter and in shape from nearly circular to elongated, resembling breccia veins. A few, like the Cactus pipe in Utah and the Colorado pipe at Cananea, Mexico, which have been followed to what seems to be their roots, show a decrease in area and less brecciation with depth. The pipe type of deposit has been particularly productive at Cananea and Nacozari, Mexico. Some of the disseminated deposits may be large pipes such as the Utah copper deposits of Bingham, Utah. In many of the pipes the ore minerals, disseminated as a breccia cement and replacement, comprise only a small part of the ore; in others, like the Colorado and Campbell pipes, parts of the ore bodies are massive sulfide. Most of the volcanic pipe deposits are in the Tertiary of the eastern Cordilleran region and were probably formed at shallow depth.

Replacement Pipelike Deposits.—These occur in limestone and have been productive at many places; the Campbell ore body at Bisbee, Ariz., is an example. These grade into the manto-type deposit common in limestones and are generally formed by solution along intersecting fractures.

The origin of pipelike ore deposits is still open to question. In some the start was doubtless

due to faulting or jointing, which formed brec-
cias. These permeable channels were enlarged
by solution before the ore minerals were de-
posited. This has been referred to as mineral-
ization stoping.

In other pipes the character of brecciation
has suggested that volcanic explosive action
has broken a more or less cylindrical body of
rock. This permeable zone has later been
traversed by mineralizing solutions which have
cemented and partly replaced the breccia.

The importance of pipelike deposits depends
on the distinction that is drawn between them
and other types, but typical pipelike deposits
of both types have produced much ore.

Disseminated (Porphyry) Deposits.—In
North America this is confined to the eastern
Cordilleran region. It occurs characteristically
in and around the tops of intrusive bodies that
probably are upward extensions of batholitic
intrusions. The intrusive bodies may therefore
be simple upward projections in the form of
stocks or cupolas, or they may follow structural
features and be modified forms of stocks or
cupolas. Some of the disseminated deposits,
such as those of the Bingham district, Utah,
and the Ely district, Nev., are in such modified
stocks.

Along the boundary of the stocks both the
intruded and the intrusive rocks have been
jointed and fractured. The fracture zones were
subsequently mineralized, mainly with pyrite
and chalcopyrite in variable amounts and pro-
portions. In some deposits, such as those of
Bingham, Utah, most of the mineralization is
in the stock; but some extended into the in-
truded quartzite, while in others, such as those
of Miami, Ariz., much of the mineralization is
in the intruded schist. The primary mineral-
ization is characteristically low in grade but
with wide variations. The deposits have all
undergone oxidation, which has affected them
differently.

Effect of Oxidation on Copper Deposits

The effect of oxidation on the value of copper
deposits is perhaps greater and more varied than
for any other metal. Several factors influence
oxidation:

1. Character of the sulfide, particularly the
proportion of pyrite to copper sulfide or
copper-iron sulfides.

2. Character of the gangue minerals or
enclosing rock, whether reactive or nonreactive.

3. Type of the deposit, whether massive
sulfide or disseminated sulfide.

CHARACTER OF SULFIDES

Pyrite (FeS_2) in oxidizing is first changed to
ferrous sulfate and sulfuric acid. If the
oxidizing process continues the ferrous sulfate
will oxidize to ferric sulfate, which may break
down to give limonite and sulfuric acid.
Abundant pyrite gives abundant sulfuric acid.
Pyrrhotite or more complex iron-copper sulfides
may oxidize similarly but will yield less sulfuric
acid than pyrite.

Formation of sulfuric acid and ferric sulfate
promotes solubility of metals and is favorable
to their movement in ground water. The
presence of iron sulfide, especially pyrite,
therefore favors leaching of copper.

CHARACTER OF GANGUE

Gangue minerals have been classed as
reactive, intermediate, and inert.

The carbonates, calcite and dolomite, are
most reactive; the micas, feldspars, and other
silicates are intermediate; and quartz and
barite are inert. An inert gangue, such as
quartz, has no chemical influence on the
oxidation products of sulfides and is favorable
to their movement. Reactive gangues, such
as calcite and dolomite, react with all the
oxidation products of sulfides, neutralizing
sulfuric acid, precipitating the metal sulfates
as carbonates, and causing precipitation of
limonite from ferric sulfate. The presence of
calcite or dolomite is unfavorable to the move-
ment of copper and tends to produce copper
carbonate deposits in the upper parts of the
oxidizing body rather than a zone from which
copper has been leached. With an abundant
carbonate gangue there will be little downward
movement of copper. With a quartz or
silicate gangue there may be much downward
movement.

As the copper sulfate, other sulfates, and
sulfuric acid move below the zone of oxidation
they encounter fresh sulfides, and the copper
is precipitated on these as copper sulfide.
Precipitation may result from direct reaction
between the copper sulfate and the sulfide or
hydrogen sulfide may precipitate sulfides from
the sulfate solutions.

TYPE OF DEPOSIT

In massive sulfide bodies, such as the lenses
in schist, the copper is precipitated before the
solutions have penetrated far into the sulfides,
resulting in a shallow zone of supergene
(secondary) sulfides which in some deposits is
very rich. This precipitation is due to the
abundance of fresh sulfides.

In disseminated deposits the sulfate solutions
penetrate deep into the zone of scattered sulfide
grains (mainly pyrite and chalcopyrite) before
all the copper is precipitated, and a thick zone
of secondary sulfide is produced which is usually
not rich.

The zones migrate downward during weathering, and oxidation of the secondary sulfide zone may produce a relatively rich zone of carbonates and oxides. The water table practically limits the downward movement of oxidation.

The net result of oxidation is the formation of four zones, any one of which, except the lowest or *primary zone, may be lacking:*

1. An upper zone from which copper has been leached.

2. A zone in which copper has been fixed as a carbonate.

3. A zone in which copper leached from above has been deposited as secondary sulfide.

4. The zone of primary sulfides.

Mineralogy

COPPER MINERALS

Copper is found in nature in numerous minerals and in various combinations with other elements. Of 165 known copper minerals only about 12 are commercially important; about 6 minerals are the source of more than 95 percent of the copper mined. Principal copper minerals are listed in table 6.

There are three important groups of copper-bearing minerals. The first is that of the primary or hypogene minerals, those deposited at considerable depth in the earth by processes that may be related to igneous activity. Examples of these are bornite, chalcopyrite, and enargite. Some complex minerals are also primary, but minerals of this type are not abundant in economic deposits.

The second group is composed of the supergene copper minerals commonly formed by weathering of copper sulfides. Cuprite, malachite, azurite, and chrysocolla are the chief minerals of the supergene zones of copper deposits when a reactive gangue is present. Brochantite and atacamite are found in the upper portions of ore bodies in arid regions.

The third group, the simple sulfides, chalcocite, and covellite, are predominant in the zone of seconadry enrichment. These minerals may also be found as primary minerals in some deposits.

The copper minerals found most frequently at the surface are the green and blue copper carbonates and the green silicates. These colors are distinctive; however, a rock with a very small percentage of copper may be colored green. Cuprite, tenorite, native copper, atacamite, chalcanthite, and occasionally other rare copper minerals are found in outcrops; these minerals, together with the carbonates and silicate minerals, are formed by weathering of copper sulfides. Chalcopyrite and, to a smaller

TABLE 6.—*Principal copper minerals*

Mineral	Composition	Specific gravity	Copper, percent
Native: Native copper	Cu	8.8 –8.9	100.0
Oxides:			
Cuprite	Cu_2O	6.0	88.8
Tenorite	CuO	5.8 –6.3	79.8
Malachite	$CuCO_3Cu(OH)_2$	3.9 –4.03	57.3
Azurite	$2CuCO_3Cu(OH)_2$	3.77	55.1
Chrysocolla	$CuSiO_3 2H_2O$	2.0 –2.4	36.0
Antlerite	$Cu_3SO_4(OH)_4$	3.39	54.0
Brochantite	$Cu_4SO_4(OH)_6$	3.39	56.2
Atacamite	$CuCl_2 3Cu(OH)_2$	3.75–3.77	59.4
Sulfides:			
Chalcopyrite	$CuFeS_2$	4.2 –4.3	34.5
Bornite	Cu_5FeS_4	4.9 –5.4	63.3
Chalcocite	Cu_2S	5.5 –5.8	79.8
Covellite	CuS	4.9 –5.0	66.4
Enargite	$Cu_3As_2S_4$	4.43–4.45	48.3
Tetrahedrite	$Cu_8Sb_2S_7$	4.7 –5.0	52.1
Tennantite	$Cu_8As_2S_7$	4.7 –5.0	57.0

extent, bornite may exist in outcrops or in float, but when found at the surface in arid districts usually they are as kernels surrounded by alteration products. Copper sulfides are generally associated with pyrite and other iron sulfide minerals.

COPPER ORES

Copper ores vary in type (whether sulfide or oxide), in the valuable metals occurring in them, in tenor, and in the kind of associated gangue present. They are commonly distinguished as oxidized ores or sulfide ores and may range between high grade and low grade, the grade roughly expressing copper content.

A sulfide ore is a natural mixture containing copper-bearing sulfide minerals and associated gangue minerals that are either valueless, as regards their metal content, or of subordinate importance for some other purpose. The gangue may consist simply of the minerals making up the host rock, or it may be composed of various other minerals developed during the general ore-forming process, or it may occur as a combination of these two groups. Thus the gangue might consist of a sulfide, such as pyrite, along with quartz, sericite, calcite, barite, etc. From an ore like this, in addition to the principal value in copper, pyrite might be reclaimed as a byproduct for its sulfur, and barite might be recovered to be used in well-drilling muds.

An oxidized copper ore is similarly a natural *mixture of oxidized copper minerals and gangue.* Examples would be malachite and azurite in dolomite stained by iron oxide, or brochantite and crysocolla deposited among silicates.

Although there are many ores in which copper is dominant and is the principal metal recovered, for some copper-bearing ores much of the value comes from other metals associated with copper or from nonmetallic constituents. Other prod-

ucts commonly contribute to the value of a copper ore.

Commonly associated with copper are minor amounts of gold and silver and locally lead and zinc. Molybdenum is recovered in several plants treating porphyry copper ores, and very minor amounts of platinum, selenium, etc., may be extracted in refining. Nickel, likewise, occurs in certain kinds of copper ore, and cobalt is a valuable associate of copper in some ores of the Belgian Congo and Northern Rhodesia. Mineralogically, a sulfide ore might contain, for example, chalcocite and pyrite dispersed through a gangue of sericite schist; chalcocite, chalcopyrite, and pyrite with a little molybdenite might be associated with gangue minerals derived from the alteration of quartz monzonite rock.

In some ores, copper occurs in coordinate importance with other valuable metals to form a so-called complex ore. By skillful beneficiation, for example, lead, zinc, and copper might be recovered from such material. Sulfide minerals in this kind of ore might be galena, sphalerite, chalcopyrite, and pyrite contained in a gangue of limestone or dolomite. Or, the ore might carry nickel and copper with a sulfide assemblage of pentlandite, chalcopyrite, and pyrrhotite in norite.

Copper itself may occur as a byproduct in some ores, and its value may be only a subordinate item as in some of the large zinc-lead, nickel, and iron deposits.

Copper ores vary widely, and the usual practice is to group them into high-grade and low-grade categories. In the United States most copper comes from the mining and concentrating of ores carrying from 0.4 to 1.20 percent copper, mainly in the sulfide form; the general average content of recoverable copper is about 0.80 percent. High-grade ores commonly contain from 3 to 10 percent copper, although some may be richer, and are generally smelted direct without preliminary concentration.

Among the sulfide copper ores, three outstanding groups should be noted. The first is composed of the porphyry copper and Northern Rhodesian types that carry copper mostly in the form of chalcocite, chalcopyrite, and bornite—iron being contained in pyrite, chalcopyrite, and bornite. Copper ranges in amount from a fraction of one percent to several percent, and iron is generally low to moderate in amount. A second group includes the deposits commonly known as cupriferous pyrite bodies. These contain very abundant pyrite and pyrrhotite and are high in iron and sulfur. Copper is carried in these ores by chalcopyrite generally ranging from 1 to 3 percent. Notable examples are at Jerome, Ariz.; Rio Tinto, Spain; and Cyprus. A third group comprises the ores

carrying some of the copper as enargite or tennantite, containing arsenic. Well-known deposits of this kind are at Butte, Mont.; Bor, Yugoslavia; the Cordilleran region of South America; and Tsumeb, South-West Africa.

Various groupings of oxidized copper ores may be made according to their response to metallurgical treatment. For example, we may recognize (1) the malachite-bearing ores in dolomite of the Congo that are successfully concentrated by flotation; (2) the various mixtures of malachite, azurite, chrysocolla, and tenorite that are subject to treatment by leaching with sulfuric acid; and (3) the unusual oxidized mineral assemblage at Chuquicamata, Chile, consisting principally of brochantite, atacamite, antlerite, chalcanthite, krohnkite, and natrochalcite, also treated by leaching.

In addition, there are various mixtures of oxidized and sulfide copper minerals occurring together in ores that are treated by leaching or by leaching followed by flotation.

There are ores in which copper is carried as the native metal. In these the gangue may be conglomerate or basalt, as in the ores from the Lake Superior district in Michigan, or sandstone, as in the much smaller deposits at Corocoro, Bolivia.

The kind of gangue accompanying a copper ore may be of considerable importance in its mining and treatment. The mineralogy, texture, and physical characteristics of the gangue, which is generally the preponderant material, may dictate the type of mining method to be employed. Similar factors control crushing, grinding, classification, and flotation of the ore during its treatment.

Copper ores are found with diverse textures, depending upon the size, relative abundance, and arrangement of their constituents. Ores in which the sulfide grains are scattered through the host rock are properly termed disseminated ores. However, when the sulfide minerals are aggregated together in rich abundance, the ore is designated as massive sulfide. Or, if the valuable minerals occur in a multitude of veinlets, it is called a stringer ore, or a stockwork.

In summary, the type of copper ore (whether oxidized or sulfide), its richness in copper, the amount of accompanying metals of value, the content of substances detrimental to treatment, the kind of gangue, and the texture and hardness of the ore all influence the selection of methods for its treatment.

WORLD RESERVES

The total reserve of copper in the world is presently estimated to be 212 million tons or enough to last about 50 years at the current

TABLE 7.—*World copper reserves*

Country	Ore reserves copper content, thousand short tons	Country	Ore reserves copper content, thousand short tons
North America:		Asia:	
Canada	8,400	China	3,000
Cuba	200	Cyprus	200
Haiti	75	India	100
Mexico	750	Israel	250
United States	32,500	Japan	1,200
		Philippines	1,000
Total	41,925	Turkey	580
South America:			
Bolivia	55	Total	6,330
Chile	46,000	Africa:	
Peru	12,500	Angola	40
		Republic of the	
Total	58,555	Congo	20,000
Europe:		Northern Rhodesia	25,000
Austria	60	Southern Rhodesia	475
Bulgaria	300	Kenya	20
Finland	750	Mauritania	460
East Germany	500	South-West Africa	525
Ireland	280	Uganda	210
Norway	500	Republic of South	
Poland	11,400	Africa	900
Spain	4,500		
Sweden	700	Total	47,630
U.S.S.R	35,000	Oceania: Australia	1,200
Yugoslavia	2,750		
		Grand total	212,000
Total	56,740		

annual production rate of slightly more than 4 million tons. The tabulation of reserves by country and continent in table 7 is from estimates reported in a variety of sources for significant copper producing areas of the world. All quantities shown were considered to be measured or indicated ores that were minable under technologic and cost-price conditions prevailing in 1960.

Standard definitions for the three categories of reserves agreed to by the Federal Bureau of Mines and Federal Geological Survey were used in appraising the estimates. The definitions are:

Measured Ore.—Tonnage is computed from dimensions revealed in outcrops, trenches, workings, and drill holes, and the grade is computed from results of detailed sampling. Sites for inspection, sampling, and measurement are so closely spaced, and the geologic character is so well defined, that the size, shape, and mineral content are well established. The computed tonnage and grade are accurate within limits which are stated, and no such limit differs from the computed tonnage or grade by more than 20 percent.

Indicated ore.—Tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, measurement, and sampling are too widely or otherwise inappropriately spaced to outline the ore completely or to establish its grade throughout.

Inferred ore.—Quantitative estimates are based largely on broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. The estimates are based on an assumed continuity or repetition for which there is specific geologic evidence of their presence. Estimates of inferred ore should include a statement of the special limits within which the inferred ore may lie.

Physical measurement of ore deposits is only one of the factors in determining the size and grade of reserves. In addition, ore reserves increase as technological advances lower costs and permit the mining and processing of lower grades of ore. And, because the grade of ore mined is also dependent on the market price of copper, a rise in price will automatically make available supplies of ore previously uneconomical to mine. A lowering of the price or any condition or event increasing the cost of production, such as inflation, will have the reverse effect of raising the economic grade of ore that can be mined and reduce the reserve of copper. The grade of ore that can be mined and processed economically differs from country to country, from district to district, and from mine to mine, depending upon the factors that influence cost. Location, water supply, transportation, and metallurgy are some of the conditions that affect costs and may determine whether or not a deposit can qualify as a reserve.

There is little doubt that potential copper resources are very large and, as demand increases and old deposits become depleted, new deposits must be found and developed. An outstanding feature of reserve forecasts over the past three decades has been the upward trend in the quantity available, despite the large tonnage of ore mined. Table 8 shows this progression of copper-reserve estimates for the United States since 1931.

In summary, the copper reserve picture for the world assures a plentiful supply of copper for many years to come.

PRINCIPAL COPPER MINES IN THE WORLD

North America

UNITED STATES

The principal sources of copper in this country are in the Western States. In 1960 copper production was reported from 16 states and reached 1,083,000 tons after falling below

TABLE 8.—*Trend of U.S. copper-reserve estimates*

Year	Tons recoverable copper	Price, cents	Life, years	Annual rate of production used to estimate life, tons	Reference
1931	At least 18, 500, 000__	9	31	600, 000	Barbour, Percy. World Copper-Ore Reserves. Eng. and Min. J., vl. 131, p. 178.
1931	18, 800, 000_____	9	31	600, 000	Rawles, W. P. The Nationality of Commercial Control of World Minerals. AIME, Contr. 41, 1933.
1934	18, 900, 000_____	10	32	600, 000	Barbour, Percy. World Copper-Ore Reserves. Eng. and Min. J., vl. 135, pp. 448–449.
1935	16, 000, 000_____ 23, 500, 000_____	10 12	22 32	750, 000	Leith, K., and D. M. Liddell. The Mineral Reserves of the United States and Its Capacity for Production, Nat. Res. Comm., Washington 1936, p. 55.
1936	23, 700, 000_____	12½	33	725, 000	Do.
1944	20, 000, 000_____	13	25	800, 000	Cannon, R. J., M. Mosier, and Helena Meyer. Mineral Position of the United States. Dept. of Interior, 1947, pp. 240–241.
1945	29, 200, 000_____	13	36	800, 000	Federal Trade Commission Report on the Copper Industry, 1947, p. 4.
1960	32, 500, 000_____	32	30	1, 100, 000	U.S. Geological Survey and U.S. Bureau of Mines.

the million mark in 1958 and 1959; the mines in six states furnished 97 percent of the 1960 output. These were: Arizona, Utah, Montana, Nevada, New Mexico, and Michigan.

Arizona

Bagdad Mine.—In western Yavapai County, this is 27 miles by road from Hillside, a station on the Santa Fe Railway. The ore body at Bagdad is a chalcocite-enriched zone in gray quartz monzonite porphyry that crops out in the form of an irregular stock, roughly 2 miles long (east and west) and 1 mile wide (north and south). It joins a complex of older rocks on the north and is enclosed by a granitic complex on the other sides. Some roof pendants of the older granite are present in the stock of quartz monzonite. The northwest-trending Hawkeye fault cuts through the quartz monzonite and the ore body and dips 70° to 80° in an easterly direction. The chalcocite ore body is present on the dropped hanging-wall block but is absent on the footwall side. Numerous minor fractures cut the quartz monzonite in the vicinity of the ore body. The principal sulfide minerals are pyrite, chalcopyrite, and molybdenite. The minerals generally are scattered as grains through the monzonite but in places occur in veinlets of quartz. Some galena, sphalerite, and bornite are found in the monzonite. In the upper or oxidized portion the quartz monzonite shows alteration by sulfuric acid solutions generated by oxidation of the sulfides.

A change from block caving to open-pit mining was made in 1948 and the annual production of copper advanced from 7,000 tons

that year to 12,000 tons in 1960. In 1954 the Bagdad ore reserve was estimated at 30 million tons of sulfide ore, averaging 0.754 percent copper, and 30 million tons of leachable oxide-carbonate material, containing 0.435 percent copper. A dump leaching plant was built in 1960 to recover copper from a substantial mine dump containing soluble copper minerals.

Christmas Mine.—In Gila County, 8 miles north of Winkelman and 22 miles south of Globe, this mine was opened in 1905 and was worked intermittently for fifty years producing approximately 1,500,000 tons of ore averaging 2.4 percent copper.

The Inspiration Consolidated Copper Co. began intensive exploration and development in 1954 of underground drifting and both surface and underground drilling, developing an estimated proven and probable ore reserve of 20 million tons containing 1.83 percent copper. The chance of developing additional ore is reported to be good.

The mineral deposits are of the contact metamorphic or pyrometasomatic type. The ore body is tabular and ranges from 20 to 100 feet in thickness. Chalcopyrite is the most abundant ore mineral, but variable amounts of bornite, chalcocite, and oxidized copper minerals are generally present. The gangue is chiefly garnet, quartz, magnetite, and unreplaced limestone. The factors that controlled the localization of the ore minerals are: (1) Proximity to the limestone-quartz diorite contact, (2) favorable character of certain limestone

beds, (3) garnetization of the limestone, and (4) postgarnetization fracturing.

Development completed by the end of July 1962 included sinking a hoisting shaft 18 feet in diameter 1,793 feet, a ventilation shaft 12 feet in diameter 1,033 feet, and driving the 1,600 level haulage drift between the hoisting shaft and the ore body. The 4,000-ton-per-day concentrating plant began operating in in August 1962. Concentrates are trucked 37 miles to the Inspiration smelter in the Globe-Miami district.

Esperanza Open-Pit Mine.—The Duval Sulphur & Potash Co. mine, 28 miles south of Tucson, started production in March 1959 after completing an extensive surface drilling and underground exploration program, initiated in 1955, and removing about 6 million tons of waste. Production capacity is expected to reach 12,000 tons of ore per day from a reserve of 49 million tons, averaging 0.75 percent copper and 0.022 percent molybdenum.

The ore body is an enriched blanket averaging 130 feet in thickness and is covered by an average of 95 feet of overburden. The ore is localized in three types of rocks: (1) A series composed largely of graywacke, arkose, and conglomerate breccia; (2) an intrusive andesite; and (3) a quartz monzonite porphyry. There is an unusually regular chalcocite enriched blanket with a capping of about the same thickness, but the protore is rich enough in places to constitute ore, especially where it has a good molybdenum content. The primary mineralization is composed of quartz, clay minerals, biotite, chalcopyrite, molybdenite, and minor pyrite.

Inspiration Mine.—The Inspiration Consolidated Copper Co. mine, formerly an underground block-caving operation, consists of two open-pit properties, the Live Oak and the Thornton, in the Miami-Inspiration ore zone, about 6 miles west of Globe. This ore zone is in Precambrian Pinal schist, which is chiefly a sericite schist consisting of quartz, sericite, and biotite. The principal intrusive masses closely associated with the ore deposits are the Schultze granite and the Lost Gulch monzonite.

The ore occurs in blanket deposits as disseminated chalcocite or oxidized copper minerals or mixtures of these in both schist and porphyry close to the porphyry contact. The porphyry and schist were thoroughly shattered. Then mineral solutions permeated the ore zone through minute, closely spaced fractures and deposited the sulfides together with silica in a very irregular network of veinlets. The predominant sulfide deposited was pyrite, with very subordinate chalcopyrite and molybdenite. The feldspars of the porphyry were attacked and partly sericitized by the solutions. A little sulfide penetrated the walls between fractures,

but most of it was confined to fractures. Subsequent oxidation produced the silicates, carbonates, and oxides—providing a major source of the copper mined.

The conversion from underground block caving to surface open-pit mining, which was started in April 1948 and was completed in August 1954, together with successful development of the Dual Process for treating mixed oxide-sulfide ore by leaching followed by flotation, resolved certain operational problems, lowered costs, allowed profitable treatment of lower grade ores, and increased ore reserves. The high-average ratio of waste removal to ore mined dropped from more than 3 tons of waste per ton of ore to 1.83 tons in 1957 and to 1.18 tons in 1958. Construction of facilities for recovery of byproduct molybdenum was completed, and plant operation commenced in April 1958. The company mining engineers estimated the proven ore reserve in the Inspiration mine to be 444,000 tons of recoverable copper as of December 31, 1958.

Lavender Pit and Copper Queen Mines.—These Phelps Dodge Corp. mines are in the Warren mining district at Bisbee near the southeast corner of Arizona. Since 1880 this area has been one of the major sources of copper in the United States, having furnished over 2½ million tons from within an area of 4 square miles.

At Bisbee Pre-Cretaceous rocks were intruded by the Sacramento Hill granite porphyry stock, and after considerable erosion the Dividend Fault with a displacement of 2,000 to 5,000 feet truncated the porphyry chimney. Mineralizing solutions came after the intrusion in various fracture zones in the limestone and followed porphyry dikes and sills. Oxidation and secondary enrichment in the mineralized granite porphyry produced a siliceous iron bearing gossan underlain by secondarily enriched low-grade ore bodies. Prior to 1917 the porphyry stock was drilled and two ore zones were proved, known as the East and West ore bodies. The West ore body was elliptic in shape with a major axis of 1,200 feet and a minor axis of 800 feet; it contained chalcocite, some bornite, chalcopyrite, and considerable pyrite in a siliceous gangue. This ore body was mined from 1917 to 1929 as the Sacramento open-pit mine, producing 159,000 tons of copper.

Stripping of the East ore body (Lavender pit) was started in 1951, and by 1959, 202,000 tons of copper had been produced.

In 1954 the ore reserves of the Lavender pit were estimated to be 41 million tons of 1.14 percent copper ore and 31 million tons of low-grade sulfide suitable for leaching with an average grade of 0.42 percent. These reserves

were estimated to last 12 years at a production rate of 16,000 tons of ore per day or 38,000 tons of copper per year. Planned expansion of the pit announced in 1959 indicated the development of sufficient ore to last 15 years at the current rate.

Ore bodies south of the Dividend Fault are deep and are mined by underground methods. They are of two distinct types: A pyrite periphery prophyry contact type and an irregular-bed replacement type. Ores of the former have a more massive character, while those of the latter tend to be tabular, since the shape of the ore body is usually influenced by bedding of the limestone replacement. To remove the ore the open-stope, cut and fill, and square-set stoping methods are used.

Mining began in the Copper Queen underground unit in 1880. By the end of 1959 about 3 million tons of copper, 90 million ounces of silver, 2 million ounces of gold, 143 thousand tons of lead and 168 thousand tons of zinc was produced. During the last several years copper has been produced at the rate of about 30,000 tons per year. Reserve data on the Copper Queen mine are not known.

Magma Mine.—This Magma Copper Co. mine is at Superior about 65 miles east of Phoenix in the Pioneer (Superior) mining district, Pinal County. The mine has been successfully operated almost continually since 1910. Production in recent years has been in the range of about 450,000 tons of 5 to 6 percent copper ore, annually yielding from 20,000 to 27,000 tons of copper. Two types of ore bodies are found in the Magma mine. Most of the ore has come from steep dipping vein deposits which are developed for a maximum distance of 9,000 feet along the strike of the vein and to a depth of 4,900 feet below zero level, or to an elevation of 1,370 feet below sea level. The principal ore mineral in the vein ore bodies is chalcopyrite with enargite, bornite, and hypogene chalcocite found in varying abundance. Replacement ore bodies were discovered in beds cut by the vein systems in the eastern end of the mine about 1949. Here the mineralization extends away from the veins along favorable beds for more than 500 feet and in some places as much as 800 feet. The average thickness is around 15 to 20 feet but at times reaches 30 feet. In the replacement ore bodies important copper minerals are chalcopyrite, bornite, and chalcocite. At the present time more than half of the ore comes from the replacement ore bodies. About 400,000 tons of ore or about 20,000 tons of copper is produced per year. The ore also contains a significant amount of silver and gold.

On January 1, 1962, the ore reserves were estimated to be 882,000 tons, averaging 7.06 percent copper. Of this, 729,000 tons of 7.22 percent ore were in the bedded ore body.

Miami Mine.—Now owned by Tennessee Corp. this mine began operating in 1911, and to the end of 1959 it had produced 1.186 million tons of copper. During the last few years of operation, underground mining at this property was carried on, in part, to break up the remaining low-grade area preparatory to producing copper through in-place leaching. In 1960 approximately 10,000 tons of copper was produced by this method. Intensive studies indicated economic advantages in the conversion to a total leaching operation which was accomplished in June 1959. The Castle Dome open-pit copper mine, belonging to the corporation, began operating in 1943; operations were terminated in 1953 after approximately 249,000 tons of copper had been produced. After the mining was terminated the dumps were prepared for leaching, and the company continued to produce about 2,500 tons of copper per year by this method. During 1961 and 1962 plans were prepared and a plant was constructed to leach the mine dumps of the Copper Cities mine by the same method. Production from the dumps of this mine will replace the production from the Castle Dome mine as it is depleted.

Copper Cities Open-Pit Mine.—Also owned by the Tennessee Corp. this is 3 miles north of the Miami mine. The deposit is a typical low-grade dissemination of chalcopyrite and chalcocite in quartz monzonite and quartz-monzonite porphyry. A churn-drilling exploration program, started early in 1946 and completed in 1949, developed an ore reserve estimated at 33 million tons carrying 0.7 percent copper. An additional 9 million tons was discovered by drilling in 1956. The ore reserve as of January 1, 1959, was estimated to be 29.2 million tons. Stripping began in June 1951 and initial mining of ore began in August 1954. Sulfide ore is mined and is concentrated at 12,000 tons per day for an annual production of approximately 18,000 tons of copper.

Mineral Hill Mine.—This is 16 miles south of Tucson and 8 miles west of Sahuarita, in the Pima Mining District. The Banner Mining Co. optioned the property in 1950, obtained a Defense Minerals Exploration loan, and began an exploration program. A substantial ore body was discovered, and the company began a development and construction program that brought the mine into production in 1954. Two ore bodies about 3,000 feet apart, the Mineral Hill and the Daisy, were developed. The Daisy ore body was developed first by diamond-drill holes from the surface then by a 450-foot vertical shaft. Some of the ore was within 25 feet of the surface and was localized along a northeast to east striking fault, which probably

is a segment of the Mineral Hill Fault. Ore occurs in sedimentary rocks in a zone near the contact between Permian limestone and quartzite; the best ore is in the limestone.

Copper deposits at Mineral Hill mine are of the contact metamorphic type. The east-west pre-ore fault traverses the property for about 5,500 feet with some minor offsets in one of about 400 feet. Ore mineralization is localized along this fault zone at and along intersections with cross faults and at or near intrusive contacts in limestone and quartzite. The main deposit is formed in the sedimentary rocks near the fault contact with a granite stock at or near the intersection of two faults. The copper ores occur as replacement in the limestone along shear zones and as disseminations in contact circuits. The main copper mineral is chalcopyrite with small amounts of chalcocite and bornite. Exploration was continued in an area north of the Mineral Hill deposit and a major ore body was discovered. A 5-compartment shaft was sunk 1,700 feet deep on the new deposit, and full production began at the end of 1961 from ore too deep for open-pit mining. Exploration of the Palo Verde ore deposit has developed 58 million tons of ore of a grade comparable to that produced by open-pit mining. Reserves at the Mineral Hill and Daisy mines are unavailable. About 1,000 tons of copper ore is produced per day.

Mission Mine.—The Mission project of the American Smelting and Refining Co.—formerly called the East Pima—is about 15 miles southwest of Tucson where the company started exploring the deposit in 1954. During the following 5-year period 346 holes totaling 190,000 feet were diamond drilled and 2,200 feet of underground workings were driven to test the potentiality of the ore body.

The deposit lies within an extensive zone of porphyry-copper type alteration-mineralization in sedimentary rocks that have been folded, faulted, and intruded by monzonite porphyry. Disseminated pyrite and chalcopyrite pervade all rocks within the zone, but ore-grade copper mineralization occurs principally along certain of the sedimentary horizons, forming gentle to moderately steep-dipping tabular bodies—ranging from a few feet to more than 200 feet in thickness. The ground is intensely broken and is structurally complicated. Leaching and secondary enrichment are confined to a thin layer at the top of the Papago formation completely buried under alluvium and caliche. Pyrite and chalcopyrite are the predominant sulfides. Molybdenite is sparsely and erratically distributed. Hydrothermally altered calcareous rocks, tactite, and hornfels contain the most copper.

The company constructed a 15,000-ton-per-day concentrator, and production of 45,000 tons of copper annually was begun in July 1961. The ore reserve has been estimated at 65 million tons of 0.9 percent copper.

Morenci Mine.—This Phelps Dodge Corp. property near the town of Clifton in southeastern Arizona is the largest producer of copper in the State and ranks second in the United States. Ore is mined at 52,000 tons per day by the open-pit method; it is concentrated and smelted nearby. Blister copper is cast into 700-pound anodes and shipped to the Phelps Dodge refinery in El Paso, Tex. Molybdenite is recovered as a byproduct.

Three types of ore bodies occur in the Morenci district. They are: (1) Irregular or roughly tabular contact-metamorphic bodies in limestone or shale near porphyry; (2) lodes or veins in fault zones; (3) irregular deposits of low-grade disseminated ore in porphyry. Early production of approximately 991,000 tons of copper came from ore bodies of the first two types.

The open-pit ore body, a disseminated deposit of the third type, is in a medium hard monzonite porphyry. The average thickness of the capping was 216 feet, and the maximum thickness was about 500 feet. The ore extends through a vertical range of about 1,300 feet, and the surface outline of the ore body forms a rough oval—covering approximately 900 acres. Some oxidized-copper minerals are present in part of the capping. This is considered leach material and is dumped where provision can be made for extracting the copper content at some future date. The principal sulfide minerals are chalcocite and pyrite. Chalcopyrite and bornite have been identified and covellite is present in minor amounts. The chalcocite occurs as a coating on pyrite and also is present as a dissemination in the gangue.

Mining began in 1942 on the open-pit ore body, estimated to contain 284 million tons—averaging 1.036 percent copper. By the end of 1960 more than 257 million tons of ore had been mined, and about 2.4 million tons of copper had been produced from this ore body. The tonnage and grade of present reserves are not available. However, exploration for new ore bodies has continued, and reserves are estimated to be adequate for many years operation at the current rate of production—approximately 100,000 tons of copper per year.

New Cornelia Mine.—This Phelps Dodge Corp. mine is at Ajo in the southwestern section of the State. Large scale mining operations were started in 1917, when a 5,000-ton-per-day leaching plant was completed. In 1924 a 5,000-ton-per-day flotation plant was built, and

additional plant facilities have since increased processing capacity to more than 30,000 tons of ore daily.

The ore body is described as almost wholly in monzonite porphyry, although the volcanic rocks to the southwest and southeast are also considerably mineralized. It is crudely elliptical in shape, about 3,600 feet long and 2,500 feet wide. Its average thickness is 425 feet, and the maximum is about 1,000 feet. The primary mineralization is chiefly chalcopyrite with some bornite and a little pyrite. The gangue consists of quartz and orthoclase, and the sulfides are distributed both in veinlets and in grains scattered through the altered monzonite; the richest ore accompanies the more intensely altered rock.

The ore body was oxidized to a surprisingly level plane near the water table at an altitude of approximately 1,800 feet. Except for local variations of as much as 50 feet, the transition from sulfide to oxidized zone was about as sharp as could be mined by steam shovel. The depth of oxidized ore ranged from 20 to 190 feet, averaging 55 feet. Minerals of the oxidized ore were malachite with a little azurite, cuprite, tenorite, and chrysocolla. A little chalcocite was found just beneath the bottom of the oxidized zone.

Information regarding ore reserves is not available, however, the mine apparently has many years of production ahead at the recent annual rate of between 60,000 to 70,000 tons of copper.

Pima Mine.—The Pima Mining Co. mine, 50 percent owned by Cyprus Mines Corp., is in the northeastern part of the Pima mining district about 20 miles south-southwest of Tucson. The deposit was discovered by geophysical means beneath 200 feet of overburden.

In November 1951 the Pima Mining Co. was incorporated to explore and develop the new ore body by shaft sinking and underground work. Late in 1955, after extensive surface and underground drilling had delineated a large quantity of fairly low-grade copper ore instead of a smaller quantity of high-grade ore, the company decided to develop the deposit as an open pit mine.

The Pima deposit includes two distinct types of ores: (1) Highly altered limestone, having strong and relatively high-grade mineralization; and (2) low-grade disseminated mineralization in volcanics and sediments. The ore-bearing rocks dip about 45 degrees to the south and trend east. They average about 200 feet in thickness and in the main part of the mine have been developed over a lateral extent of 1,600 feet. The principal ore mineral is chalcopyrite; pyrite and magnetite are accessory minerals.

Chrysocolla (copper silicate) and the black copper oxide, tenorite, are oxidation products occurring for a short distance below the bedrock surface.

An open-pit mine was developed, and a 3,000-ton-per-day concentrating plant was built. It began operating in 1957 on an ore body estimated to contain 6,400,000 tons of approximately 2 percent copper. In 1959 the rated daily capacity of the concentrator was increased to 4,000 tons. An agreement was made with neighboring Banner Mining Co. under which Pima would enlarge its pit to include a certain portion of Banner property and would mine and mill 1.8 million tons of the Banner ore from that area. A substantial tonnage of low-grade material is known to exist.

The company plans to expand the open pit, eliminate the inclined skip, and provide for all truck haulage. The plans include increasing mill capacity to 8,000 tons per day.

Ray Mine.—This property of the Kennecott Copper Corp. is about 80 miles east and slightly south of Phoenix and about the same distance north of Tucson in what is known as the Mineral Creek Mining District. Ore was mined by the block-caving system from 1911 to 1955 when conversion from underground mining to an open-pit operation was completed. The change in mining methods took 7 years and involved removing 43 million tons of overburden.

The rocks that contain the ore body are Precambrian Pinal schist with intrusions of quartz porphyry and diabase. The mineralized diabase formerly too low grade to mine by underground methods has become ore as a result of the change in operations.

The schist and porphyry ores are mineralized principally with supergene chalcocite, but the predominant mineral in the diabase is chalcopyrite. Copper silicates, cuprite, malachite, tenorite, and native copper compose about 20 percent of the schist ore.

Annual recovery of copper from in-place leaching of the caved areas in the old underground mine increased from 3,600 tons in 1954 to 17,700 tons in 1957. New pumps were installed in 1959 for leaching additional caved areas to further production of copper from this source.

Information about the ore reserve of the Ray deposit is not available. Ore is produced from the open-pit mine at the rate of 22,500 tons per day. The ore is concentrated, and the concentrates are smelted in company facilities at Hayden. Blister copper is produced at the rate of 60,000 tons per year.

Lone Star Area.—Northeast of Safford, the area has been drilled by the Bear Creek Mining Co., a subsidiary of Kennecott Copper Corp.

An 800-foot shaft (started in September 1960), and underground development work will allow bulk sampling of the ore body.

San Manuel Mine.—Operated by the San Manuel Copper Corp., this is in the Old Hat mining district, Pinal County, approximately 45 miles northeast of Tucson. The concentrator, smelter, administration building, and other plant facilities are located 7 miles southeast of the mine area at the town of San Manuel.

The San Manuel ore body is a disseminated copper deposit in quartz monzonite, monzonite porphyry, and diabase. The ore lies within a mineralized zone having a width of 8,000 to 9,000 feet and a known length of 9,300 feet.

The upper and western portions of the ore body are separated into two branches by an area of lean, weakly altered hanging-wall rock. These two branches converge at depth so that, in cross section, they may be likened to a U or V leaning to the northwest. The northwesterly limb of ore is referred to as the North ore body and the southeasterly limb, as the South ore body. To the east, the central area of lean material gradually diminishes and disappears as the two limbs join to form a single mass of ore.

The main primary minerals are chalcopyrite, pyrite, and quartz; there are minor amounts of molybdenite. These minerals are distributed quantitatively into three zones known as the ore zone; the hanging-wall zone, which lies between the North and South ore bodies; and the footwall zone, which surrounds the ore and the hanging-wall zones.

Except for the west end the upper portions of the ore body and much of the central lean-hanging-wall zone have been oxidized or partially oxidized. The chief minerals resulting from oxidation and enrichment are chrysocolla, chalcocite, and various iron oxides. Cuprite, native copper, and black copper oxides are often seen where the oxidized-zone grades into the chalcocite zone. Copper carbonates are very rare.

There is complete or nearly complete oxidation of the copper minerals and a relatively sharp line of demarcation between the oxide ore and the underlying sulfide. Little copper was leached from the oxidized zone, and the zone of chalcocite enrichment contains only a relatively small tonnage. Most of the sulfide ore can be mined as clean sulfide with a minimum of mixed sulfide-oxide ore.

A drilling program was completed early in 1948 which proved 479.5 million tons of ore, averaging 0.77 percent copper—consisting of 367.6 million tons of sulfide ore, containing 0.79 percent copper, and 111.9 million tons of oxidized ore, containing 0.72 percent copper.

Silver Bell.—This operation of the American Smelting and Refining Co. consists of two open-pit copper mines and a concentrator located about 40 miles northwest of Tucson.

The two pits, known as the Oxide and the El Tiro, are approximately 2½ miles apart in a northwesterly trending zone of hydrothermal alteration. Alaskite, dacite, andesite, and monzonite porphyry have been enriched by super-gene chalcocite to form the two ore bodies. Within the ore bodies, where alteration is strong, the upper limit of the sulfide zone usually is highly irregular but sharp. There is no transition zone of mixed oxide and sulfide minerals, except locally in fringe ore areas where alteration is relatively weak and rocks are reactive. In general the base of oxidation conforms to modern topography, though local relief exceeds 200 feet. The water table for the most part is now well below the chalcocite zone.

The American Smelting and Refining Co. started developing the two pits in December 1951. Since then and up to January 1, 1957, 21 million tons of waste and 6.4 million tons of ore, assaying about 0.9 percent copper, was removed from the Oxide pit; 15 million tons of waste was stripped, and 1 million tons of ore was mined from the El Tiro pit. At the time production of ore started on March 1, 1954, the two ore bodies had a reserve of 32 million tons of 0.9 percent copper.

Michigan

Calumet & Hecla Mines.—The Calumet division of Calumet & Hecla, Inc., is mining copper ore from six sources, the Seneca, Allouez, Oceola No. 13, Centennial, and two Ahmeek shafts near Calumet in the northern peninsula. Total production from all sources amounts to about 6,500 tons of ore a day, averaging around 1 percent copper, it is treated at the Ahmeek concentrator. The Tamarack reclamation plant at Torch Lake is still retreating old tailings, accounting for 1,500 tons of copper in 1960; total mine output of the Calumet division was almost 16,000 tons of copper in 1960.

Copper deposits in the district are commonly divided into two broad groups—lode and fissure. The first group comprises conglomerate lodes (mineralized beds of felsite conglomerate interstratified with the lavas) and amygdaloid lodes (mineralized vesicular, brecciated, or scoriaceous tops of lava flows). The second group are veins along the fractures that parallel or cross the beds. The economically important lode deposits, with a single exception, are confined to the portion of the Keweenawan series that is composed predominantly of lava flows, but they have a wide stratigraphic range within that portion. The fissure deposits are confined to the same stratigraphic portion of the series, although the valuable fissure deposits and the

lode deposits occur in different areas along the strike of the formation. The Nonesuch lode is a copper-bearing sediment that occurs well up in the sedimentary portion of the series where mineralization is mainly in sandstone and to a less extent in shale. Reserves at the Calumet and Hecla mines are not known.

White Pine Mine.—This mine of White Pine Copper Co., a wholly owned subsidiary of Copper Range Co., in Ontonagon County has become one of the 10 major copper producers in the United States since beginning production in 1955. Facilities include the mine, mill, smelter, refinery, power plant, and a complete town. Initial designed capacity for an annual production of 36,000 tons of copper has been increased to 50,000 tons; production in 1960 was 40,000 tons.

The copper-bearing beds at White Pine are primarily in the lower 20 to 25 feet of the Nonesuch shale. This cupriferous zone is divided into four stratigraphic units which are in ascending order, the lower sandstone, the parting shale, the upper sandstone, and the upper shale. Most of the copper occurs as chalcocite in the upper and parting shales, except in a small area near the White Pine fault where it is abundant in the upper and lower sandstones. It is present in five different layers in values averaging from 1 to 3 percent. Copper content of the shale beds decreases with increasing sand content.

The White Pine ore body still contains at least 300 million tons of ore. Ore found by exploration since production started has substantially exceeded the tonnage mined, and the limits of the ore body have not yet been determined.

The Southwest ore body, discovered in 1959, is between one and two miles southwest of White Pine on the opposite side of the White Pine fault. The ore occurs at a depth of 2,000 feet from the surface and contains about 1.5 percent copper. In November 1961 the new ore body was reached with a development shaft, and the grade of the exposed ore confirmed information obtained from exploratory drilling. The company expects to have this property in full production by early 1964.

Montana

The Butte district mines operated by The Anaconda Company are at Butte in southwestern Montana.

All production before 1952 was from underground-vein mines, but today there are three different methods of mining: Underground vein, underground block caving, and open pit. The underground copper-vein mines in the Butte district are the Mountain Con, Leonard, Belmont, and Steward; the Kelley

mine is the block-caving operation; and the Berkeley Pit is the surface or open-pit mine. From 1880 to July 1, 1959, 300 million tons of ore were mined, recovering nearly 15 billion pounds of copper. In addition, the operations produced more than 28 million tons of zinc ore, yielding 4.5 million pounds of zinc; 8 million tons of manganese ore, containing 3.5 billion pounds of manganese; 618 million ounces of silver; and 2.3 million ounces of gold.

The highly complex ore deposits are localized on the south margin of the Boulder batholith in an area of approximately 6 square miles. Fractured quartz monzonite was altered and mineralized by ore-bearing solutions penetrating fractures. Subsequent faulting and repeated mineralization have created an extremely complex maze of veins and veinlets belonging to at least three systems. The oldest, largest, and most productive veins are in the east-west trending Anaconda system; next in size, age, and productivity are the mineralized faults of the Northwest veins. The Steward or Northeast-fault veins intersect and displace veins of the two earlier systems and in turn are displaced by various post-mineral dikes and faults. Although each system displaces its predecessor veins, all are nearly comtemporaneous and are characterized by identical ores distributed in a concentric zoning pattern. In the central and intermediate zones, mineralization consists of copper sulfides oxidized and enriched to varying depths. In the outer limits of the intermediate zone and in the peripheral zone, copper mineralization gives way to zinc, manganese, and silver. The zonal arrangement is vertical as well as lateral but is masked and complicated by oxidation and supergene enrichment.

Minable low-grade copper ore bodies, of the Kelley mine, locally called ore zones, are in the eastern part of the central zone of mineralization. The high-grade ore occurs in pyrite-enargite veins and ore shoots related to branching and interlacing structures of different ages, which intersect the quartz-monzonite. The most highly productive Anaconda system of fractures has a prevailing east strike and south dip; the Blue vein set of horizontal slip fractures strikes northwest and dips southwest. Lastly, the Steward fault veins range in width from a few inches to a maximum of 50 to 60 feet. Some veins are traced as far as 7,000 feet in length and extend to a depth of several thousand feet. Some veins extend to the surface; some are blind.

The Kelley low-grade ore bodies may be divided into two types. The wide-vein zones consist of closely spaced fissure veins with numerous ore veinlets and stringers traversing the intensely altered quartz-monzonite, making

certain parts of it minable by block caving. The other type is related to the horsetail system of southeast-striking fractures, branching from the high-grade east-west veins, that have replacement copper ore minerals localized to a degree suitable for block caving. 180 million tons of low grade (1.0 percent) copper ore is known to be available for block caving.

The mineralized quartz monzonite that constitutes the Berkeley ore zone is part of the central zone of mineralization created by the hydrothermal processes previously mentioned and enriched by downward percolation of surface waters. It included not only the zone of secondary enrichment but remnants of high-grade ore veins that originally were mined underground. Chalcocite is the principal ore-bearing mineral, with lesser amounts of enargite, bornite, and chalcopyrite present. Post-mineral faulting that complicated underground mining is present within pit limits but is not expected to interfere with open-pit operations.

The enriched zone ranges from 100 to occasionally 600 feet in vertical thickness (the average is about 200 feet) and is about 2,600 feet wide by at least 4,700 feet long. Below the enriched zone replacement minerals grade rapidly into primary sulfides. Barren leached capping, totaling 100 to 300 feet overlies the enriched zone. The leached capping constitutes the bulk of the waste material, but occasional barren or weakly mineralized areas are encountered in the ore zone.

In 1959, the open-pit ore reserve was 123 million tons, averaging 0.75 percent copper and 0.17 ounces silver. To mine this ore required removal of 138 million tons of waste and 40 million tons of leach material. The leach material averages 0.2 percent copper.

The reserve of the vein mines was estimated at 9.6 million tons of ore having a grade of 4.35 percent copper in 1957.

Nevada

Liberty and Veteran Open-Pit Mines.—These mines of the Nevada Mines Division, Kennecott Copper Corp., are in the Robinson Mining District in White Pine County, a few miles west of Ely. The productive portion of the Robinson Mining district, sometimes called the Ruth-Kimberly area, covers an area of about 20 square miles and had produced more than 2 million tons of copper.

The geology of the district shows sedimentary rocks of limestone, quartzite, and shale invaded by monzonite and monzonite porphyry. The principal ore bodies, which are of the disseminated type, occur in the monzonite and have been formed by chalcocitic enrichment of pyrite-chalcopyrite protore. There has also been considerable mineralization of the lime-

stones, but relatively little sulfide ore has been found in these rocks. The ore deposits at Ely are believed to be genetically connected with the intrusion of the monzonite. Supergene enrichment may have been in operation for a long period, possibly beginning before the eruption of the Pliocene lavas.

Underground mining at the Ruth mine ceased in February 1958 with the exhaustion of ore in the Minnesota-Hi ore body. A lower ore body, known as the Deep Ruth with an estimated 25 million tons of ore, averaging about 0.8 percent copper, has been partially developed for mining at some future date.

In 1960 production came wholly from the Liberty pit where 7.35 million tons of ore averaging 0.79 percent copper provided 47,439 tons of copper. The total estimated ore reserve of the Nevada Mines Division is not known.

Yerington Mine.—The Yerington mine of The Anaconda Company is at Weed Heights, Lyon County, 80 miles southeast of Reno. The initial exploration, upon which the Yerington open-pit project was planned, was conducted by the company from 1942 to 1945 and consisted of both surface drilling and underground workings. Stripping of overburden began in June 1952, and the first copper was shipped in November 1953; the planned production of 5 million pounds of copper per month was attained in May 1954.

The Yerington ore body is a disseminated porphyry-copper deposit. Copper mineralization is associated with an intrusion of quartz-monzonite porphyry into a granodiorite mass, the mineralization occurring in the quartz-monzonite porphyry and the bordering granodiorite. Minor intrusions consist of andesite and rhyolite dikes. Chrysocolla is the chief copper mineral, along with minor amounts of malachite and other copper silicates and carbonates. The outlined ore within the deposit averages 0.82 percent copper. The original estimated ore reserve, determined by preliminary exploration, was 35 million tons of 0.97-percent oxide copper underlain by 15 million tons of similar grade sulfide ore.

The Yerington operation comprises stripping and mining the oxidized-copper deposit, leaching the ore with sulfuric acid, recovering the copper by precipitation with shredded iron as cement copper, and shipping to the company smelter at Anaconda, Mont. The sulfuric acid used was manufactured at Weed Heights from sulfur ore mined by The Anaconda Company at the Leviathan mine in Alpine County, Calif., 58 miles west of Weed Heights, until June 1962 when the Leviathan mine was closed.

The production rate of oxide ore was reduced after October 1961 when the new concentrator for sulfide ore treatment began operations.

This plant was operating at its rated capacity of 5,000 tons of ore per day at the end of 1961.

New Mexico

Chino Mine.—This mine of the Chino Mines Division, Kennecott Copper Corp., is a completely integrated unit. Copper ore is mined, concentrated, smelted, and refined; molybdenum is recovered as a byproduct. The Chino mine is at Santa Rita, in the Central Mining District of Grant County; the concentrator, smelter, and fire refinery are at Hurley, approximately 9 miles from the mine.

The geologic detail of the deposit is complex. Extensive sills of quartz diorite intruded sediments, and these in turn were intruded by a stock of granodiorite porphyry. Fracturing and faulting in many directions were intense. Dikes radiate from the granodiorite core. Mineralizing solutions permeated this fractured zone around the relatively unbroken and impervious granodiorite core and deposited the primary ore minerals, copper-bearing pyrite, and chalcopyrite. These minerals were deposited partly in the intrusive and partly in the intruded rock. Silicification and sericitization occurred with deposition of the ore minerals. Kaolinization accompanied decomposition of the sulfide minerals.

Leaching of the upper zone, followed by subsequent secondary enrichment over a wide area, formed an ore body of economic grade that could be mined by open pit.

The total production, from the inception of open-pit mining in 1910 to the end of 1958 was 207 million tons of copper ore. The 1960 production rate was 22,500 tons of ore and 45,000 tons of waste per day. The average copper content of the ore is approximately 0.84 percent. The monthly output of the smelter is about 12 million pounds of copper. Ore reserves are not known.

Miser's Chest Group of Mines.—This group owned by the Banner Mining Co., is in Hidalgo County, 6 miles south of Lordsburg. The mines and mill were closed in October 1957 owing to the low price of copper, but development work continued. Operations were resumed in 1959 and in 1960 copper production was 1,847 tons.

The exposed rocks of the Lordsburg mining district consist chiefly of basalt and rhyolite flows, plugs, and associated fragmental material belonging to two groups of different age and character. An irregular stock of porphyritic granodiorite intrudes the rocks of the earlier group, and both the stock and earlier rocks are cut by several varieties of dikes that are separated from the later volcanic rocks by an unconformity. The ore deposits are siliceous veins in fault fissures and shear zones con-

centrated in and around the northern and eastern margins of the granodiorite stock. There are two main groups of these veins, one striking northeast and the other almost due east. Oxidation and secondary enrichment have affected the ores to extremely variable depths. In places, almost unaltered sulfides are found at the surface, yet in other sections oxidation and leaching extends to the 1,400 foot level.

The Bonney vein has been developed by the Banner Mining Co. to a vertical depth of 1,560 feet and for 2,000 feet along the strike. The minimum stoping width is 4 feet, and the maximum vein width is about 20 feet. The reserves of ore are unknown.

Tyrone Deposits.—This Phelps Dodge Corp. property, a potential resource of copper, is in the Big Burro mountains of Grant County, approximately 10 miles southwest of Silver City. The deposits have all the characteristics of a chalcocite porphyry ore with the chalcocite either disseminated regularly throughout large masses of fractured country rock or concentrated along exceptionally strong veins or shear zones. The overburden is of variable thickness, either barren of copper or containing carbonates and other oxidized copper minerals. The principal ore bodies lie within a northeastward trending zone of fracture between Leopold and Tyrone. They are very irregular in size and shape, ranging from roughly blanketlike masses—grading upward into oxidized ground and downward in primary pyrite—to strong veins or shear zones—which are likewise barren near the surface and too lean to be of value below, where chalcocite gives way to primary pyrite.

In 1949 Phelps Dodge Corp. initiated a drilling program at Burro Mountain to develop a large low-grade ore body, minable by open-pit methods. This development drilling was continued into 1958 when it was suspended. Encouraging results were reported each year.

North Carolina

Ore Knob Copper mine.—In Ashe County, this mine, operated by Appalachian Sulphides, Inc., a wholly owned subsidiary of Nipissing Mines Co., Ltd., produced ore at the rate of 24,000 tons per month, equivalent to 450 tons of copper in 1960.

Discovered in 1855, the Ore Knob began operating as a mine in the spring of 1873; in the next ten years it produced 12,500 tons of copper. Federal Bureau of Mines drilling in 1942 and 1943 disclosed a substantial reserve in the mine. Nipissing Mines Co., Ltd., leased the mine in 1953 and started an exploration program that indicated more than a million tons of ore. Shaft sinking started in 1955, and production of ore commenced in 1957.

The steeply dipping fissure vein in gneiss which contains the ore is as much as 50 feet wide and more than 1,500 feet long. There are vein splits that branch into the hanging wall from the main ore zone at a low angle (branching upward), and these are mainly pyrite with some chalcopyrite.

The vein is parallel to the strike but steeper than the dip of foliation. It is along a shear or fault that cuts across the dip of gneiss and locally formed breccia fragments. Coarse biotite in the hanging wall and foot wall is related to vein formation.

Ore minerals are pyrite, chalcopyrite, and minor quantities of sphalerite, while the gangue contains biotite, albite, calcite, and many silicate minerals.

The ore reserve estimated in 1957 at 1,300,000 tons, averaging 3.0 percent copper was exhausted in 1962 and the mine was closed.

Pennsylvania

Cornwall iron mine near Lebanon, operated by the Bethlehem Steel Co., is primarily an iron mine but it has been a source of copper since 1920 after successful flotation experiments with the ore in 1919. Copper production increased significantly in 1928 as a result of further flotation-plant improvements. Flotation of pyrites was started in 1940 because the pyrite carried recoverable cobalt. From that time until recently, Cornwall ore accounted for nearly all U.S. cobalt production.

The ore is a contact replacement. type. High-grade, iron-bearing solutions coming from or through the traprock sill caused replacements of the limestone at its contact with the upper side of the traprock intrusion. The ore consists of magnetite with pyrite and chalcopyrite. It contains recoverable copper, gold, silver, cobalt, and sulfur. The gold and silver apparently are in solid solution in the chalcopyrite, and cobalt occurs similarly in the pyrite.

Tennessee

Tennessee Copper Division of the Tennessee Corp., operating the Boyd, Calloway, Eureka, and Mary mines in the Ducktown Basin mining district in Polk County is the largest producer of copper in the eastern United States. The ore, concentrated in two flotation mills, yields copper, pyrite, and zinc concentrates; the pyrite concentrate is roasted to produce sulfur dioxide for use in manufacturing sulfuric acid and iron oxide; the iron oxide is sintered for use in iron and steel plants. Annual production of copper from 1955 to 1960 has ranged from 9,000 to 13,000 tons.

The deposits are great lenses of massive sulfide ore. They extend, with interruptions, for thousands of feet on the strike and are up to 300 feet thick. Except where faulted they parallel the foliation of the schists which are closely folded.

The country rock consists of schist and graywacke intruded by gabbro. The great group of granitic batholiths of the southern Appalachian Mountains lies a few miles east, and the deposits may have been formed in connection with the intrusion of the granite, probably near the end of the Paleozoic era.

The minerals include pyrrhotite, pyrite, chalcopyrite, magnetite, garnet, amphibole, and chlorite. The ore near the surface is altered to iron oxide or gossan, which extends to a depth of about 100 feet; below the gossan at the water level a zone of chalcocite from 3 to 8 feet thick was mined, containing 5 to 25 percent of copper. The primary copper ore carries about 1 to 3 percent of copper. Reserves of ore are unknown.

Utah

The Utah Copper mine of the Kennecott Copper Corp., sometimes referred to as the Bingham Pit, is the second largest copper producer in the world; an average of approximately 90,000 tons of ore is mined and milled each day. The mine is near the town of Bingham in Salt Lake County and along with two nearby concentrators (the Magma and Arthur mills) and a smelter and electrolytic refinery at Garfield, the Utah Copper Division, Kennecott Copper Corp., has a fully integrated operation from ore to refined copper.

The Bingham District has been a steady and large producer of copper since 1896 and has one of the largest reserves of developed ore in the United States.

The igneous rocks in this district are intrusives of monzonite in the forms of stocks, dikes, and sills and andesite flows and tuffs. The monzonite intruded sedimentary quartzites, limestone, and shales to form two extensive irregular stocks—the Bingham and the Last Chance. Dikes connect these major masses and, together with smaller stocks, traverse sediments to the south and east of the main stocks. Mineralization is more intense in and around the Bingham stock and most of the ores within the stock are copper ores. In the sedimentary rocks close to the stocks copper ores also prevail, but they grade outward into zinc-lead-silver ores that give an irregular but definite zoning of the metals.

The ores of the Bingham stock are typical of the disseminated copper deposits. The monzonite has been intensely shattered and altered. The primary ore minerals are pyrite and chalcopyrite, which occur as grains and veinlets disseminated through the deposit. Molybdenite is present locally.

Based on recent activities of the Kennecott Copper Corp. at its Utah Copper Division such as, building the electrolytic refinery, purchasing the Garfield smelter of the American Smelting & Refining Co., driving the 5,490 level railroad tunnel underneath the pit, and announced plans for modernizing the smelter; it is evident that Kennecott has assigned a life expectancy to the Utah operation of at least 20 years, a reasonable plant amortization period. An ore production of 32 million tons was reached in 1956 and the grade of ore averaged 0.86 percent copper from 1957 through 1960. Using these criteria the reserve in 1961 is estimated to be 640 million tons of ore averaging 0.85 percent.

CANADA

Newfoundland

Tilt Cove Mine.—Owned and operated by Maritimes Mining Corp., Ltd., this mine is on tidewater at Tilt Cove on the Eastern side of the Burlington Peninsula. An intensive prospecting program started in 1954 resulted in blocking out sufficient ore to justify preparation for mining. In 1956 a 2,000 tpd mill was completed, and shaft sinking was started. Milling was started in August 1957, and the concentrates have been smelted by Gaspé Copper Mines, Ltd., at Murdochville, Quebec.

Chalcopyrite and pyrite occur in a chloritized zone along a contact between pillow andesites and dykelike intrusions of quartz-diorite porphyry. Two types of ore bodies occur on the property—a massive type, consisting of lenticular pods of fine-grained pyrite and chalcopyrite, and a disseminated stockwork type in which pyrite and chalcopyrite form irregular clusters, networks of stringers, and irregular veins. The massive sulfide type of deposit is lenticular, has sharp, clearly defined boundaries, and occurs usually at the contact between the quartz-diorite porphyry and the chloritized andesite. The disseminated type of ore body usually occurs some distance from the intrusive-volcanic contact, is lower in grade than the massive type, and the mineralization extends into the wall rocks—so that ore limits are by assay boundaries.

Mining is now only by underground operations; production averaged 2,232 tons per day during 1961.

Ore reserves estimated on December 31, 1961, totaled 3,050,000 tons of unstated grade. Production of 814,748 tons of ore in 1961 averaged 1.68 percent copper and returned 5,146 ounces gold.

Buchans Mine.—This is in the central part of the island 3 miles north of Red Indian Lake and is operated as the Buchans Unit of American Smelting and Refining Co. Geo-

physical prospecting and diamond drilling led to discovery of two deposits in 1927, and two more ore bodies in the northwest were revealed by diamond drilling from 1947 through 1950.

The ore bodies occur on the north flank and around the westward plunging nose of a dome-shaped structure, at the east end of a northeasterly trending anticline, and are replacements of volcanic breccias and associated tuffs. The shape of the ore bodies ranges from lensoid to tabular. The ore can be divided into three main groups from hand specimens: Breccia, containing numerous rock fragments; baritic, containing abundant barite; and normal, from massive sulfides with few fragments. Ore minerals present are galena, sphalerite, chalcopyrite, pyrite, gold, silver, and minor amounts of bornite and other minerals. The gangue minerals are quartz, barite, and calcite.

Ore reserves reported on December 31, 1961, were 4,627,000 tons of 1.25 percent copper.

Quebec

Horne Mine.—The mine and the smelter of Noranda Mines, Ltd., are at Noranda in Rouyn Township, northwestern Quebec. The mine and plant have produced continuously since 1927, and the smelter has been enlarged so that it is now able to treat custom ores and concentrates from most of the copper mines in western Quebec, Ontario, and a mine in Manitoba.

The ore bodies at the Horne mine occur in metamorphosed rhyolitic flows, tuffs, and agglomerates as massive sulfide replacements of brecciated rhyolite and as disseminated pyrite and chalcopyrite mineralization in massive-appearing rhyolite rocks. Faulting and shearing are considered to be the main structural factors controlling the emplacement of the ore bodies. Principal metallic minerals present are pyrite, pyrrhotite, chalcopyrite, sphalerite, gold, silver, and associated tellurides.

Approximately 95 percent of the total output of the Horne mine comes from cut-and-fill stopes, the remainder from blast hole stopes. The production rate at the Horne mine in 1959 was 3,810 tons per day. As of January 1, 1960, the sulfide ore reserve at the Horne mine was 9,303,000 tons of 2.29 percent copper and 0.19 ounce gold per ton.

Gaspé Copper Mine.—Deposits are owned and exploited by Gaspé Copper Mines, Ltd., a subsidiary of Noranda Mines, Ltd. The mine site is in the northwest corner of Holland Township, Gaspé North County, in the Gaspé Peninsula, 25 miles from the south shore of the Gulf of St. Lawrence.

Two major ore zones have been outlined on the company-owned property—the Needle Mountain ore bodies and the Copper Mountain

ore body. The former consists of bedded replacement deposits of chalcopyrite and pyrrhotite, while the Copper Mountain ore body consists of late-stage pyrite—chalcopyrite mineralization filling fractures in a siliceous host rock. The Needle Mountain ore bodies, where most of the mining and development has been done, are higher in grade and more uniform in constitution than the Copper Mountain deposit.

The production schedule calls for 8,000 tons of ore per working day from the mine; 90 percent is supplied from the underground operation, and the remainder is from the open pit. The ore reserve reported on January 1, 1960, was 63,710,000 tons, having a copper content of 1.29 percent.

Quemont Mine.—This property of Quemont Mining Corporation, Ltd., is contiguous to, and directly north of, the Horne mine of Noranda Mines, Ltd. The property lies within the city limits of the City of Noranda. Mine and mill began production in June 1949 at its rated capacity (2,000 tpd); it has continued to produce at this or a slightly higher rate.

The ore bodies at the Quemont mine consist of sulfide replacements of a brecciated rhyolite, and three types are mined: Massive sulfide replacement ore; chloritized breccia ore, containing disseminated sulfides; and shattered breccia ore, containing sulfide stringers. The ore occurs in a dome-shaped structure in which the permeable rhyolite breccia is the core, and a thick series of impermeable porphyritic rhyolite forms the capping. Three main ore bodies have been exploited, the West, the East, and the Southwest. Exploration at depth is contemplated to explore the ore zone.

As of December 31, 1959, the ore reserves totaled 6,220,000 tons, containing 1.32 percent copper, 2.71 percent zinc, 0.176 ounce of gold per ton, and 1.06 ounces of silver per ton.

Waite Amulet Mine.—Owned and operated by the Waite Amulet Mines, Ltd., this is in Duprat and Dufresnoy Townships, approximately 9 miles north of Noranda.

The ore bodies occur as two types of massive sulfide lenses, lying along flow contacts in early Precambrian extrusives. The structure of the region is complicated, and the structure localizing the ore bodies is not easily recognized. The massive sulfide lenses consist of (1) pyrite and sphalerite, containing 10 to 11 percent zinc and low copper; and (2) massive pyrrhotite lenses, carrying 4 to 7 percent copper and low zinc. Minor metals in the ore are lead, silver, and gold. The ore reserve reported on December 31, 1959, was 550,000 tons of approximately 4.4 percent copper and 3.6 percent zinc. The reserves have become depleted, and mining will cease by the end of 1962; the company has been absorbed by Noranda.

Sullico Mines, Ltd.—This property is in the township of Bourlamaque, 5 miles southeast of Val d'Or. Two large sulfide replacement ore bodies occur in a series of siliceous volcanics which strike north 60 degrees and dip steeply to the south. The ore has been localized in a quartz-chlorite alteration zone that has been folded, fractured, and in places, replaced by the sulfides. The zone of alteration has been traced over a length of 1,500 feet, is 600 feet wide, and persists beyond the bottom level of the mine. Ore bodies within the zone of alteration have lengths of from 300 to 500 feet, widths up to 95 feet, and range in vertical height from 600 to 800 feet. The principal minerals are chalcopyrite, sphalerite, and pyrite with small values in gold and silver.

Mine production in 1959 was 2,600 tons per day from stopes above the 2,500-foot level. Seven levels are being developed below the 2,500-foot level. The ore reserve reported December 31, 1959, was 3 million tons—averaging 1.0 percent copper, 0.73 percent zinc, and 0.008 ounce of gold and 0.35 ounce of silver per ton.

Normetal Mining Corporation, Ltd.—A copper-zinc property that consists of 12 mining claims in Desmeloizes Township, 13 miles north of Dupuy station. Development and exploration of the mine has been accomplished by means of four shafts; three are in use today.

The ore bodies are massive sulfide replacement lenses in sheared rhyolite agglomerate which has been intruded by various igneous rocks. Principal sulfide minerals in order of abundance are: Pyrite, sphalerite, chalcopyrite, and pyrrhotite and minor amounts of galena, arsenopyrite, chalcocite, and bornite. Silver and gold are also found in the ore. The sulfides are not evenly distributed in the ore bodies, since it appears that the copper values are concentrated along the hanging-wall side while the massive pyrite and sphalerite occur along the footwall. The ore body varies in length and width but on the 2,600 level shows a length of 1,025 feet and average width of 18.4 feet. Ore reserves as of December 31, 1959, were reported as follows:

	Tons	Copper, percent	Zinc, percent
Copper-zinc ore	1, 405, 900	4. 00	3. 22
Zinc ore	192, 800	. 37	19. 00
Total	1, 598, 700	3. 59	5. 05

Golden Manitou Mine.—Operated by Manitou-Barvue Mines, Ltd., this is in Bourlamaque Township, Abitibi County, in northwestern Quebec, and consists of 48 claims with an area of 1,850 acres.

The ore bodies occur in sheared Keewatin volcanics which lie along the southern contact

of the Bourlamaque batholith. The ore shoots vary greatly in length and are from 6 to 60 feet wide. No definite bottom has been established for the ore zone, which has been developed down to the 2,950 level. The zinc ore bodies contain sphalerite, pyrite, arsenopyrite, chalcopyrite, and galena, while the copper ore body is a mixture of chalcopyrite and pyrite, containing low but recoverable values in gold.

Ore reserves at the end of 1959 were: Copper ore (above the 1,870-foot level): 797,238 tons—averaging 1.14 percent copper, 0.25 ounce silver, and 0.017 ounce gold. Zinc ore (above the 3,000-foot level): 438,172 tons—averaging 7.14 percent zinc, 0.98 percent lead, 5.63 ounces silver, and 0.03 ounce gold.

Campbell Chibougamau Mines, Ltd.—The mines are at Doré Lake, Obalski Township, Abitibi-East County, in the Chibougamau District of northwestern Quebec. In addition to the main property on Merrill Island, the company owns the Cedar Bay property in McKenzie Township, 6 miles from the mill, and has leased approximately 120 acres adjoining the main mine from Merrill Island Mining Corp., Ltd.

The Merrill Island ore body, which extends southeastward into the property of Merrill Island Mining Corp., Ltd., is localized along a shear zone in altered and silicified anorthosite. The shear zone ranges in width from 100 to 600 feet, is steep dipping, and has been traced for a strike length of 3,000 feet. The ore bodies occur in the shear as massive-sulfide replacements with pyrrhotite, chalcopyrite, pyrite, and sphalerite—the principal minerals, occurring in that order of abundance. The ore widths average 40 feet but range from 12 to 85 feet. Contacts are gradational, and the ore limits are determined by assay boundaries. A new ore occurrence has been encountered at depth by diamond drilling, and an internal shaft is being sunk below the 1,900 level for exploration of this zone.

Proven and probable reserves as of June 30, 1959, totaled 10,191,134 tons, averaging 2.5 percent copper and 0.068 ounce of gold per ton.

Opemiska Copper Mines, Ltd.—At Chapais, the company owns 58 claims in Levy Township, Abitibi-East County, in the Chibougamau District. The mine is serviced by three shafts: Springer No. 1 shaft, 2,320 feet deep; Springer No. 2, shaft, 2,414 feet deep; and Perry exploration shaft, 2,000 feet deep. Springer No. 2 is the ore hoisting shaft. Mill capacity was expanded to 2,000 tons per day in November 1959.

The ore bodies consist of veins localized in a composite basic sill, intruded into acid and intermediate volcanic rocks of Keewatin age, the whole assembly having been upturned and folded after emplacement of the sill. Massive stringers and lenses mineralized with chalcopyrite, pyrite, and magnetite and minor amounts of pyrrhotite, gold, silver, and molybdenite constitute the ore. Sulfide minerals also occur as fracture fillings and are disseminated in the wall rocks. The east-west veins have been the most productive. Mineable widths range from 3 to 73 feet; lengths are variable (to 3,000 feet); and the vertical extent is controlled by the contact between the gabbro sill and the underlying rhyolites; one zone was traced for a vertical distance of 1,200 feet.

Reserves as of December 31, 1959, totaled 5,278,700 tons of positive and indicated ore, averaging 2.97 percent copper.

Ontario

The International Nickel Company of Canada, Ltd.—The original discovery of the Sudbury District copper-nickel ores was made when the Canadian Pacific Railway, building its transcontinental right-of-way, cut through a mineralized outcrop near the present site of the Murray Mine in 1883. Most of the important ore bodies were discovered within the next few years.

The ore bodies in the Sudbury District are associated with the norite member of the Sudbury irruptive. The contact of the norite with the underlying rocks is the primary structure which determines the location of ore deposition. Essential but secondary structures are: Depressions in the relatively smooth norite footwall; shearing that roughly coincides with the base of the norite; contact breccias at the base of the norite; and norite dykes or offsets penetrating the footwall rocks. The common ore minerals are pyrrhotite, pentlandite, and chalcopyrite. Minor amounts of cobalt, selenium, tellurium, and the platinum metals also occur in the ore.

During 1959, the tonnage mined was mainly produced from underground operations at the five producing mines, Frood-Stobie, Creighton, Garson, Levack, and Murray; the balance was produced from the Frood Open Pit. Thirteen operating shafts serve these mines for hoisting ore and handling men and supplies.

The ore reserve, reported December 31, 1960, is as follows:

Tons of ore:	Nickel-copper content, tons
290,273,000	8,715,300

Falconbridge Nickel Mines Ltd.—In 1928, Ventures, Ltd., acquired a group of claims of the Sudbury Basin and incorporated Falconbridge Nickel Mines, Ltd., to develop the property. Additional property in the Sudbury area was acquired in later years, and an active explora-

tory program was maintained. New ore bodies were discovered and developed.

Underground operations are carried on at the Falconbridge, East, and McKim mines in the Falconbridge area on the south rim of the basin and at the Hardy, Boundary, Onaping, Fecunis Lake, and Longvack mines in the Onaping area on the north rim. All these ore occurrences lie at or directly below the contact of the norite with the footwall rocks. Pyrrhotite, pentlandite, and chalcopyrite are the main ore minerals. Ore reserves at December 31, 1959, were as follows:

	Tons	Nickel, percent	Copper, percent
Developed ore: Falconbridge, East, McKim, Hardy, Onaping, Fecunis Lake, and Longvack mines	22,200,050	1.56	0.87
Indicated ore: Sudbury District only	23,982,400	1.34	.77
Total	46,182,450	1.45	.82

Geco Mines, Ltd.—The property of Geco Mines, Ltd., 1 mile north of Manitouwadge Lake in the Thunder Lake District, Port Arthur Mining Division of Ontario, consists of 57 mining claims. Preliminary diamond drilling indicated the presence of a sizeable copper-zinc ore body and Geco Mines, Ltd., was incorporated in October 1953 to explore and develop the property. The first ore was produced from an adit driven into the A zone in 1955.

The Geco ore body occurs in a band of quartz sericite schist at the contact between the schist and granite or garnetiferous hornblende-biotite gneiss. Mineralization consists of pyrite, chalcopyrite, sphalerite, and pyrrhotite, with minor amounts of gold and silver. A lenticular core of massive sphalerite-pyrrhotite mineralization occurs close to and parallel with the south wall of the orebody. This core has a maximum thickness of 60 feet but tends to lens out along the strike and at depth. The high-grade zinc core is surrounded by a wide envelope of disseminated sulfides including chalcopyrite, sphalerite, pyrite and pyrrhotite.

Ore reserves at the end of 1959 were as follows:

Ore body:	Tons	Copper, percent	Zinc, percent	Silver, ounces per ton
A	3,016,300	1.94	1.93	1.00
B	7,098,400	1.57	4.51	1.79
C	5,927,800	2.38	5.33	3.28
Total	16,042,500	1.94	4.38	2.19

These reserves allow for 10 percent dilution and include ore to 25 feet below the 1,450 level.

Wilroy Mines, Ltd.—This company owns 30 mineral claims to the west of, and adjoining the property of, Geco Mines, Ltd. Diamond drilling from the surface indicated sufficient ore to warrant bringing the mine into production, and shaft sinking started in 1956. The mine plant and mill were completed in July 1957, having a capacity of 800 tons per day. By the end of 1959 the shaft had reached a depth of 1,562 feet, and mill capacity had been increased to 1,100 tons per day.

The Wilroy deposits occupy the same band of metasediments in which the Geco ore bodies were found. Massive sulfides replace an iron formation to form a lenticular ore body which strikes east-west, dips north at 70°, and plunges east at 44°. The ore consists of massive pyrite, pyrrhotite, sphalerite, magnetite, chalcopyrite, and galena. Zoning of the sulfides is apparent, and copper is more abundant in the center of the ore mass while sphalerite predominates on the borders. Ore reserves on December 31, 1959, were:

Tons:	Copper, percent	Zinc, percent	Silver, ounces per ton
2,708,066	1.63	4.85	1.58

Reserve figures allow for 10 percent dilution.

Manitoba-Saskatchewan

Flin Flon mine.—This mine of Hudson Bay Mining and Smelting Co., Ltd., second largest copper producer in Canada, is on the border of Manitoba and Saskatchewan 87 miles north of The Pas, Manitoba.

The Flin Flon deposit is in a sheared greenstone-quartz porphyry complex. The lenticular ore bodies are in a footwall and a hanging-wall zone separated by a band of barren schist as much as 100 feet thick. The average dimensions of the massive pyritic ore lenses are as much as 900 feet long, 70 feet wide, and range from 1,500 to 2,500 feet in depth. Three types of ore occur in the mine—massive sulfide; disseminated pyrite and chalcopyrite; and disseminated ore containing bands of massive sulfides. The massive-sulfide type ore is characterized by sharp boundaries either against the wall rock or against the disseminated ore. The disseminated ore does not have a sharp cut-off, and ore limits are established by assay boundaries. Minerals present are pyrite, chalcopyrite, galena, gold, and silver. The ore contains a small amount of arsenic, selenium, and cadmium. Ore reserves December 31, 1959:

Location: At or near	Tons	Copper, percent	Zinc, percent	Gold, ounces per ton	Silver, ounces per ton
Flin Flon	11,842,100	3.12	3.4	0.066	0.93
Near Snow Lake	5,615,500	1.68	8.5	.051	1.13

Lynn Lake Mine.—This property of Sherritt Gordon Mines, Ltd., was discovered in September 1941. Surface diamond drilling and under-

ground exploration programs led to development of two ore zones. The mine plants, the mill, and the branch line of the Canadian National Railways were completed in 1953, and production started in 1954. The Sherritt Gordon nickel refinery is at Fort Saskatchewan, Alberta. The plant was completed in May 1954 with a designed yearly capacity of 16.8 million pounds of nickel. Subsequent expansions have raised this yearly capacity, and in December 1959 production was at the rate of 30 million pounds of nickel a year.

The ore bodies at Lynn Lake occur in faulted basic rocks which have been intruded as irregular stocks or plugs into Precambrian metasediments and volcanics. These plugs—consisting of varying amounts of diorite, gabbro, amphibolite, and norite—are numerous in the area but only two are known to contain ore bodies. Faulting is common with the major faults trending north and a secondary group of faults striking northeasterly. Movement along the faults has caused shearing and brecciation, and the ore minerals occur as bodies of massive sulfides, as disseminated sulfides, and as stockworks of sulfide stringers in the fractured zones. The two principal ore bodies are the A and the EL. The A ore body contains 4,975,000 tons, averaging 1.22 percent nickel and 0.64 percent copper, and is a disseminated-sulfide type. The EL ore body has massive-sulfide mineralization and stockworks of sulfide stringers filling fractured zones in the basic rock. Combined tonnage and grade of the two types of ore in the EL ore body totals 2,445,000 tons, averaging 2.50 percent nickel and 0.93 percent copper.

The copper precipitate recovered at the nickel refinery is shipped to Noranda, Quebec, or Tacoma, Wash., for smelting. Production in 1959 was 12,406 tons of nickel, 5,171 tons of copper and 314,343 pounds of cobalt. Ore reserves at the end of 1959 were reported as 14,158,000 tons, averaging 0.96 percent nickel and 0.54 percent copper.

British Columbia

The Britannia mine, of the Howe Sound Co. is on the east side of Howe Sound, approximately 30 miles north of Vancouver, and is one of the oldest and largest base-metal mines in Canada. From 1939 through 1957 mine production averaged about 4,200 tons of ore per day; then the property closed early in 1958 because of low base-metal prices. It was reopened in January 1959 and operated for the balance of the year at an average rate of 1,200 tons a day.

The ore bodies occur in a northwesterly trending shear zone in a roof pendant imbedded in the Coast Range batholith. The shear zone has been traced for a distance of eight miles,

has a maximum width of 2,000 feet, strikes roughly north 70° west and has an average dip of 70° to the south. Three distinct types of ore bodies occur at Britannia: Large siliceous replacement, containing disseminated pyrite chalcopyrite, and some sphalerite; stringer-lodes, containing chalcopyrite and pyrite; and the barite-sphalerite type of zinc.

Ore reserve figures are not published, but it is estimated that the reserve is sufficient to maintain a 1,200 ton per day operation for at least five years.

CUBA

The major copper producer in Cuba is the Matahambre mine in the Province of Pinar del Rio, 100 miles west of Havana. Ore zones lie between, under, and sometimes in members of four fault systems. The zones are tubular or pipelike, each containing many shoots occurring as vein segments, lenses, and wedges. Major ore zones are elongated vertically, and converging at depth they plunge generally toward the northwest. The ore minerals are chalcopyrite and pyrite. Mill heads run 6 percent; a 32 percent copper concentrate is produced with a 98 percent recovery.

In 1957 daily production was curtailed from 1,100 tons per day to 700 tons per day and annual output of copper has declined from 20,000 tons in 1955 to 10,000 tons in 1959. Reserves are not known but the company is optimistic about encountering a sizeable ore body at depth.

HAITI

Consolidated Halliwell, Ltd., of Canada holds title to a 100-square-mile-mining concession, through its Haitian subsidiary, Sedren, S.A., in the northwestern part of the Republic of Haiti. The district is known as Terre-Neuve, and is 180 miles north of the capital, Port-au-Prince, and 13 miles from tidewater at Gonaives. Almost 81,000 feet of diamond drilling was completed at the Meme, Casseus, and Brasillac showings, including 11,290 feet of underground drilling in the main Meme workings. Two distinct types of ore have been explored, surface secondary oxides and underground primary sulfides.

The surface-oxide mineralization occurs as rich pockets of erratic distribution which are easily accessible and appear limited in tonnage. The copper minerals are entirely oxides and carbonates, principally malachite and azurite. The primary ore is entirely beneath the surface and contains chalcopyrite and bornite in nearly equal proportions, with minor chalcocite in local instances. The principal gangue minerals

are epidote, garnet, calcite magnetite, chlorite, and quartz.

At 30 cents per pound for copper, ore reserves after dilution have been estimated at 4,031,400 tons, averaging 1.88 percent copper.

MEXICO

A little more than half of Mexican copper production comes from copper ore deposits, and the balance is recovered from complex lead-zinc-copper ores. Reserves of copper ore are not well known for Mexico; however, under favorable economic conditions most of the large operating mines apparently have ample reserves to supply continuing operations at present production rates.

Cananea Consolidated Copper Co., S.A.— This subsidiary of Greene Cananea Copper Co., owned by The Anaconda Company, operates underground and open-pit mines, a concentrator, and a smelter near Cananea, Sonora, 50 miles southwest of Bisbee, Ariz.

The ore bodies are relatively homogeneous deposits, occurring in breccia pipes associated with porphyry and granite intrusives. The mineralized pipes occur in an area 6 miles wide and 15 miles long. The principal ore mineral is chalcocite; the ore gangue is an intensely siliceous, kaolinized, sericitized, fine-textured volcanic rock. The ore bodies are limited by assay boundaries.

Ore reserves are believed to be large at the present operating grade of 0.7 to 0.8 percent copper, but they are not blocked out in advance of requirements. Production in 1959 was 32,182 tons of copper, 433,771 ounces of silver, and 9,030 ounces of gold.

Boleo Mine.—This Lower California mine was operated by Compagnie du Boleo from 1886 until 1954. Compagnie Minera Santa Rosalía, S.A., a semiofficial agency of the Mexican Government, is conducting studies to develop low-cost methods to concentrate and smelt complex low-grade copper ores from numerous but small occurrences in the district. The ore series includes three beds of conglomerate averaging 100 feet in thickness. The ore minerals are principally chalcocite with minor bornite and chalcopyrite. Reserve information is not available, however, there apparently is enough ore in the area to encourage continuing research on economical recovery methods.

South America

BOLIVIA

Corocoro Mine.—Operated by the Bolivian Government, it is of unusual interest because of the presence of native copper that constituted much of the production for many years. In recent years chalcocite has supplemented the native metal as the principal mineral in the ores. The deposits are found in beds of two unconformable clastic formations known as the Ramos and Vetas. The Ramos beds dip eastward while the Vetas dip westward, and they are separated locally by the steep dipping Corocoro fault. Native copper is localized in certain sandstone beds of the Ramos and in the oxidized zone of the Vetas. Throughout the Vetas chalcocite is interstitial with sand grains and rock fragments in certain red sandstone and arkose beds. The localization of chalcocite in a particular bed seems related to processes of sedimentation.

It is estimated that approximately 7 million tons of ore, containing 350,000 tons of copper, have been taken from the district. A reserve of 750,000 tons of 2.6 percent copper ore was estimated in 1957.

Chacarilla Copper Mine.—This mine 90 miles southwest of LaPaz, is being developed by six Japanese companies. The reserve has been estimated at 900,000 tons of 4 percent copper ore, and if development is successful a monthly production of 700 to 1,000 tons of ore is scheduled. The concentrates will be shipped to Japan.

CHILE

The copper mining industry of Chile consists of three sectors: American great mining companies, medium mining companies, and small copper-mining companies. The great mining enterprises are those producing 25,000 tons or more of copper per year. At present there are three companies in this category: Chile Exploration Co., Chuquicamata mine; Andes Copper Co., El Salvador mine; and Braden Copper Co., El Teniente mine. Medium mining companies produce less than 25,000 tons of copper annually but have capital of more than 15 million pesos. Small mining companies are firms or individuals with capital not greater than 15 million pesos. The great mining companies account for about 90 percent of Chilean copper output, and the balance is divided between the medium and small producers.

Chuquicamata, 160 miles northeast of the port of Antofagasta by rail, is considered to be the largest economic copper deposit in the world. It is owned by the Chile Exploration Co., which is controlled by the Chile Copper Co., a subsidiary of The Anaconda Company. The mine lies at an altitude of 9,300 feet in the northeastern part of the Atacama Desert, one of the most arid regions in the world. Because of the extreme aridity, large quantities of sulfate and other water-soluble minerals have accumu-

lated that normally would have been carried away by rainfall.

The deposit is a pear-shaped porphyry mass about 2 miles long, 3,600 feet wide, and with unknown depth. It is in the midst of a large body made up of granodiorite and monzonite porphyry. Mineralization is related to extensive fracturing and shearing, accompanied by much silicification and sericitization, and occurs in veins and innumerable networks of veinlets. The ore body contains three types. Oxide ore occurs in the upper section and has averaged 1.63 percent copper. Below the horizon of oxide ore there is an area of mixed ore lying above the huge reserve of sulfide ore, averaging slightly less than 2.00 percent copper. The principal oxide mineral is antlerite, $Cu_3(SO_4)(OH)_4$, and the most important minor oxide minerals are atacamite, brochantite, chalcanthite, and krohnkite. The sulfide minerals in the upper part of the sulfide zone are mainly chalcocite and covellite; enargite is found at increasing depth.

Approximately 190,000 tons of ore and waste is mined daily. The annual output of the sulfide plant and the oxide plant is 300,000 tons of copper. At this rated capacity, proven ore reserves at Chuquicamata are sufficient for more than 50 years. From the start of operations in 1915 through December 1959 Chuquicamata produced 6,835,000 tons of copper; 1959 production totaled 306,500 tons, a record output.

El Salvador Mine.—This mine of Andes Copper Mining Co., a subsidiary of The Anaconda Company, is a major porphyry copper deposit in the great copper belt of the Chilean Andes. The mine is in Atacama Province, 65 miles inland from the Pacific port of Chanaral and 20 miles north of Potrerillos. Production that was started in April 1959 replaces that of the old mine and concentrator at Potrerillos which discontinued operations June 10, 1959. The mine, concentrator, and smelter were designed for copper production of 100,000 tons per year; production in 1960 was 87,000 tons.

A broad band of intrusive rhyolite dikes and sheets and monzonite porphyry extends for several miles to the southwest of Indio Muerto Peak, making a major structural axis along which the El Salvador center of intrusion and mineralization occurs. Mineralizing activity along this axis is in a belt 3 miles long and 1 mile wide. Within the zone several centers of better-grade copper mineralization and more intense alteration are localized by intrusive rocks. El Salvador is in the southernmost and largest of these centers. Occasional small, scattered argentiferous lead-zinc veinlets fringing the copper mineralization indicate weak,

widespread metal zoning. The ore body is a zone of secondary enrichment that ranges in thickness from a few feet at the edges to more than 950 feet in its central portion. Chalcocite is the principal ore mineral.

Block caving is the mining method used, and the mine has capacity to produce 24,000 tons of ore per day. The average grade of ore is 1.5 percent copper, and the reserve is estimated at 375 million tons.

La Africana Mine.—Operated by the Santiago Mining Co., a subsidiary of The Anaconda Company, this is 15 miles west of Santiago at an altitude of about 3,000 feet. Production started in 1957. Expansion of mill facilities has increased annual production capcity to 6,000 tons of copper in the form of concentrates.

The ore body is a fault fissure 10 to 60 feet wide in coastal grandiorite batholith of Cretaceous Age. The oxidized-leached zone reaches a depth of 160 feet; below is a zone of secondary enrichment 30 to 100 feet in depth, assaying 6 to 12 percent copper with chalcocite being the chief mineral. In the primary ore zone the principal copper mineral is chalcopyrite. Ore mined averages 3.2 percent copper from which a 29-percent copper concentrate is produced. Ore reserves are estimated at 2.5 million tons, averaging 3.5 percent copper.

Braden or El Teniente Mine.—Braden Copper Co., a subsidiary of Kennecott Copper Corp., owns this mine, in the Chilean Andes, 7,000 feet above sea level at the town of Sewell, 80 miles southeast of Santiago.

The Braden deposit is unique among first ranking copper producers of the world for being located in and around an old explosive volcanic vent. Only underground mining has been practiced, and only sulfide ore has been treated. Molybdenum is recovered. Annual production of copper is between 180,000 and 190,000 tons. Mine production in 1960 was 11.5 million tons of ore, averaging 1.95 percent copper. In 1947 reserves were estimated at 200 million tons of 2.18 percent copper.

Medium and Small Mines.—Of the medium and small mining companies, 10 account for about 60 percent of the production of this group. The nine long established firms in this group are: Cia. Minera Tocopilla, Cia. Minera du M'Zaita, Cia. Minera Disputada, Cia. Minera Cerro Negro, Cia. Minera Tamaya, Cia. Minera Aysen, Sali Hochschild, Renacimiento Aurifero, and Capote Aurifero. The 10th, the Santiago Mining Co. La Africana mine, started production in 1957.

Two other important copper deposits have been developed and are nearing the production stage. The Cerro de Pasco Corp. has been engaged in an exploration program on the Rio Blanco copper property, 35 miles northeast of

Santiago, since 1955. Reserves of 121 million tons of ore, averaging 1.6 percent copper, have been delineated, and the company is studying the economic feasibility of the project. An 11,000-ton-per-day block caving operation is planned. The next step will be driving a 3-mile adit under the ore body from which further exploration and development of the ore body can be conducted at depth.

The Empressa Minera de Mantos Blancos, a subsidiary of the Mauricio Hochschild organization, has explored and developed the Mantos Blancos deposit, 28 miles east of Antofagasta at an elevation of about 3,000 feet. Drilling since 1952 has proven the first 10,500,000 tons of about 2 percent copper ore. The several ore bodies lie within an area 2.5 miles long by 1,640 feet wide and extend in depth below 164 feet; many of the drill holes are bottomed in ore. The mineralization consists of copper oxides—mainly the copper oxychloride, atacamite.

The leaching plant was officially inaugurated March 27, 1961. The annual capacities of the open-pit mine and leaching plant were designed for producing 18,000 tons of fire-refined copper.

PERU

Until 1960 the Cerro de Pasco Corp. was the principal producer of copper in Peru. The corporation operates copper-producing mines at Cerro de Pasco, Morococha, Yauricocha, and the recently reopened San Cristóbal mine; concentration plants located at Cerro de Pasco, Casapalca, Morococha, and Mahr Tunnel; and a smelter at Oroya. Northern Peru Mining Co., a subsidiary of the American Smelting and Refining Co., produces copper from its Quiruvilca mine. Other copper producers include several Peruvian-owned organizations, operating mines and concentration plants, and the French-owned Campagnie des Mines de Huaron. In addition, small copper mines, mainly Peruvian-owned, sell their ores to the principal producers or other ore buyers for processing or export.

In 1960 Peru advanced to one of the major copper-producing countries of the world when more than 145,000 tons was recovered from the Toquepala mine of Southern Peru Copper Corp. The Quellaveco, Cuajone, Antamina, and Cobriza deposits have been explored and contain large reserves of low-grade ore.

Cerro de Pasco.—Deposits are localized in and around the eastern and southern portion of a body of pyroclastic rock, agglomerate, and quartz-monzonite porphyry that appears to be a deeply eroded volcanic vent. A large cresent-shaped replacement mass of pyrite and silica is contained in the contact zone between the volcanic rocks of the vent and the limestone east of the major fault. At intervals within the massive pyrite body are found economic concentrations of copper-silver ore, silver ore, and lead-zinc silver ore. The copper ore bodies vary greatly in size, and few persist more than 300 feet vertically. Enargite-famatinite, luzonite, and tetrahedrite-tennantite are the usual ore minerals but bornite, chalcocite, and chalcopyrite dominate in localized occurrences. Vein mineralization accounts for about half the ore tonnage. Some veins follow fractures within the pyrite mass, and others extend for hundreds of feet west into the volcanic rocks of the vent. Most of the copper and zinc in the mantle of oxidized material, known locally as pacos, were leached—except in small areas adjacent to the limestone where these metals occur as carbonates. Rich secondary sulfide ores, chalcocite, and covellite extended below the pacos 500 feet to the primary pyrite material; mining today is limited to primary ores.

Morococha.—Deposits, mined by Cerro de Pasco Corp., a subsidiary of Cerro Corp., are about 100 miles east of Lima at an altitude of 14,500 feet. They are of several types: (1) Fissure veins in altered limestone, volcanic rocks, and quartz-monzonite; (2) limestone replacement mantos, pipes, and sills; and (3) contact metamorphic zones adjacent to the intrusion. Mineral and metal zoning is well exhibited at Morococha, both on a horizontal or district pattern and on a vertical scale in the separate ore bodies. Enargite in the central zone gives way to chalcopyrite and bornite in the surrounding limestone. Reserves are unknown but are understood to have decreased in recent years.

Yauricocha Mine.—This mine of Cerro de Pasco Corp. is in the Department of Lima, Province of Yauyos. High-grade sulfide ores in steeply dipping irregular mantos in limestone and an extraordinary pipe of rich oxidized ore, all carrying silver and gold as well as copper, comprise the valuable deposits. The ore bodies are replacements in limestone adjacent to and probably genetically related to a stock of monzonite porphyry. The principal ore mineral is enargite. Only direct smelting ore is mined; the grade of ore shipped in 1959 was 5.8 percent copper, 1.2 percent lead, 2.3 percent zinc, and 4.5 ounces of silver per ton. Reserves have been estimated at about 700,000 tons of ore.

San Cristobal.—The deposit, in the Department of Junin, mined by Cerro de Pasco Corp., consists of fissure vein systems of zinc-lead-copper-silver ore. Ore extensions found in the early 1950's replenished depleted reserves. In 1960 the mine was reported to have 1.5 million tons of measured ore—averaging 1.2 percent copper, 9 percent lead, and 12 percent zinc. Sphalerite is the dominant ore mineral within the mine area and chalcopyrite is next, but

silver, lead, and gold contributed about half the total value.

Quiruvilca.—This mine of Northern Peru Mining Co., a subsidiary of American Smelting and Refining Co., is in the Province of Santiago de Chuco, in the Department of Liberstad, about 80 miles from the port of Salaverry at an altitude of 13,000 feet. The copper veins occur in a small portion of a large mineralized area within which there is considerable diversification in the content of the veins. There are three mineralized areas in the larger district—a southern group of silver veins with quartz filling, a central group of enargite and tetrahedrite veins, and a northwesterly group of silver veins with barite gangue. Operations have been centered largely on the copper group. Those copper veins are notable for their persistence in strike and mineralization, and the ore averages more than 5 percent copper. Copper production ranges between 5,000 and 6,000 tons a year. Reserves are unknown.

Toquepala.—This Southern Peru Copper Corp. mine is in southern Peru, 55 airline miles north of the small city of Tacna and the same distance inland from the port of Ilo. The Toquepala ore body lies in a mineralized zone, elliptical in shape and about two miles long, which has been the locus of intense igneous activity. Several small intrusive bodies having irregular forms occur within and adjacent to a centrally located large breccia pipe. The mushroom-shaped ore body consists of a flat-lying enriched zone of predominant chalcocite with a stemlike extension of hypogene chalcopyrite ore in depth within and around the pipe. Hydrothermal alteration is pervasive in the zone. The principal sulfides, pyrite, chalcopyrite, and chalcocite, occur mainly as vug fillings in the breccia, and as small grains scattered through all the altered rocks.

Operations at Toquepala of the Southern Peru Copper Corp. began January 1, 1960, and the scheduled rate of production was reached in March. Mine production of ore and waste averaged 166,897 tons per day. Ore milled averaged 26,052 tons per day and contained 1.73 percent copper, which is substantially higher than the average grade of the ore body. According to published data the deposit contains 400 million tons of open-pit ore, averaging a little more than 1 percent copper. Production in 1960 was 145,115 tons of blister copper.

Cuajone.—This porphyry copper deposit is the northernmost of a group of three deposits in southern Peru controlled by the Southern Peru Copper Corp.—Toquepala, Quellaveco, and Cuajone—all within a 20-mile belt. The principal ore body, the Chuntacala, is on the southern part of the northeastward-dipping Cuajone quartz-monzonite stock. It is roughly

circular in plan and about 3,000 feet in diameter near the surface. With depth the ore body divides into two distinct roots; the northern portion tapers downward as an inverted cone; and the southern portion follows down the south contact zone of the quartz-monzonite intrusive. Sulfides within the ore body occur as small veinlets that may be as much as 2.5 inches wide but are usually 0.5 inch wide, or less, in closely spaced array. Pyrite is the most abundant of the sulfide minerals; chalcopyrite is the principal ore mineral. Minor quantities of bornite, galena, sphalerite, and enargite were noted in the drill samples. Molybdenite is common but spotty. Supergene enrichment processes have formed chalcocite and covellite. Oxide minerals and local concentrations of native copper occur where the zone of oxidation encroaches upon enriched ore. Reserves have been estimated at 500 million tons of slightly more than 1.1 percent copper ore.

Quellaveco.—The deposit is the smallest of the three having an estimated reserve of 200 million tons of ore of approximately the same grade as the other two.

Antamina.—This deposit, approximately 100 miles northwest of the town of Cerro de Pasco, has an indicated ore body of approximately 100 million tons of 1.5 percent copper ore. Exploration and development activities by Cerro de Pasco Corp. at this deposit were temporarily suspended in 1957.

Cobriza.—The deposit, 115 miles southwest of La Oroya, was explored by the Cerro de Pasco Corp., and the tonnage of 3.5 percent copper ore developed in 1961 would support mining for an annual production of approximately 9,000 tons of copper in concentrates for about 10 years.

Europe

AUSTRIA

The Mitterberg mine, near Salzburg, is the most important of 22 copper mines listed for Austria. Two others, the Schwaz and the Untersulzbach, are intermittent producers but only Mitterberg operates continuously. It has been known as an important source of copper since ancient times. The ore averages 1.6 percent copper and occurs in veins with siderite, andesite, and calcite as matrix; chalcopyrite is the principal ore mineral. The reserve is estimated at 3 million tons of ore, averaging 2 percent copper.

BULGARIA

Bulgaria has more than doubled its production of electrolytic copper in the last five years. Recently discovered deposits at Panagyuriste,

in the Stara Planina mountains, and near Malko Turnovo provide sources for further expansion of mine capacity. The lead-zinc ores of the Rhodopt region have important copper values. Information regarding production, reserves, and descriptions of the operating copper mines is not available. It is known that smelting and electrolytic refining capacities are being increased and a large expansion of Bulgarian copper production is planned for 1962. Refinery output was 14,000 tons of electrolytic copper in 1960.

FINLAND

Outokumpu.—This mine is in eastern Finland, in the Province of North Karelia, about 30 miles northwest of the town of Joensuu. The massive sulfide ore deposit, about 3,800 meters long, 300 to 350 meters wide, and 7 to 9 meters thick, is part of a beltlike complex of quartzite and serpentine rocks. The ore minerals are chalcopyrite, pyrite, and pyrrhotite; the average metal content of the ore is 3.7 percent copper. Reserves are estimated at about 15 million tons.

Paronen.—The mine is in the Parish of Ylöjärvi about 12 miles northwest of the town of Tampere. In the area the rocks are mainly volcanics which are strongly broken in places, forming two almost parallel breccia zones. The ore deposit forms part of the larger zone, and the breccia differs considerably from the surrounding volcanics. The angular fragments of discolored tuffite and porphyrite have been cemented by a mass of ore-bearing, fine-grained, needlelike tourmaline. Chalcopyrite and arsenopyrite occur in varying amounts, and commercial ore bodies are likely to be limited, often irregular in form, and without any definite geological contacts. The ore is low grade, containing about one percent copper, minor amounts of silver and gold, and in places about 0.1 percent tungsten trioxide (WO_3). The ore reserve is estimated at 1 million tons.

Vihanti mine.—This is approximately 30 miles south of the town of Oulu on the Gulf of Bothnia. The ore bodies are located in a schist zone where the main rocks are mica schist, quartzite, and marble—the latter largely altered to a skarn. Mineralization occurs in at least two separate and parallel zones in an area which is about 4,000 feet long and 650 feet wide. Sphalerite is the most abundant ore mineral, galena and chalcopyrite occurring as a weak dissemination throughout the ore. Accessory minerals are cubanite, tetrahedrite, and tennantite. Copper production in 1959 was 3,227 tons contained in concentrates.

Kotalahti.—This mine, in the center of Finland south of Jyväskylä, started operations in October 1959 and produced 40,652 tons of ore that year; 4,742 tons of nickel concentrate and 390 tons of copper concentrate were recovered.

Pyhasalmi.—This deposit in the Province of Oulu has been developed for an annual production rate of 600,000 tons of ore; the expected recovery of copper, beginning in 1962, is 16,500 tons. The deposit contains copper, zinc, pyrite, and precious metals.

EAST GERMANY

Most of the German copper has come from the Mansfeld area. The ore body is a thin but extensive cupriferous bed of black shale, 30 to 40 centimeters thick, which contains as much as 2 percent copper. Comprehensive exploration in the Sangerhausen basin developed a 35-centimeter-thick copper-slate deposit, having higher copper content than that of the Mansfeld basin. Copper production will shift from the Mansfeld to the Sangerhausen area as the newly developed Niederroeblingen copper mine reaches the planned production rate. As a consequence copper production of East Germany is expected to be 70 percent higher in 1965 than in 1958 or about 41,000 tons.

IRELAND

Avoca Mine.—This property of St. Patrick's Copper Mines, Ltd., is 43 miles south of Dublin and 7 miles from the small harbor of Arklow. The outcrops in the area were first worked many centuries ago, but the earliest underground mining was in the 1750's. Mining was carried on continuously until around 1880. Workings in a strike length of about 3 miles, having more than 100 abandoned shafts as much as 700 feet deep, indicate the intensity of historical mining activity.

The ore bodies are sulfide-rich sheets, lenses, and associated tonguelike zones lying within altered schistose rocks. The ore minerals in order of abundance are: Pyrite, chalcopyrite, sphalerite, and galena. Reserves of about 20 million tons of ore, assaying 0.95 to 1.36 percent copper, have been developed.

Mountain Mine.—Near Allihies in County Cork, this is being developed by Can-Erin Mines, Ltd., of Canada. Limited drilling has proved a total of 2,350,000 tons of ore, averaging about 2 percent copper.

NORWAY

The cupriferous iron pyrites mine of Orkla Grube-Aktiebolag is at Medalen. Deposits are massive pyrite lenses in gabbro and contain about 2 percent copper. Smelting and refining plants at Thamshamn in Orkedalsfjorden are operated by Orkla Metal-Aktieselskap, a subsidiary; the head office is in Løkken Verk. In

1959, 342,303 metric tons of ore was mined of which 245,616 tons was smelted, yielding 3,800 tons of copper and 78,349 tons of sulfur. The ore reserve in mid-1959 was estimated to be sufficient for about 15 years production.

The other principal producer of copper in Norway is Sulitjelma Gruber, A/S, which operated cupriferous pyrite mines at Sulitjelma about 60 miles east of Bodø. The ore, in lenses, consists of copper-bearing pyrite and pyrrhotite, disseminated through mica schist. Some of the pyrite is massive. Copper production in 1958 was approximately 3,800 metric tons. Ore reserve data are not known.

POLAND

The principal copper-mining districts are the Złotoria and Bolesławiec-Grodziec basins—estimated to contain 200 million tons of ore, averaging approximately 0.7 percent copper. Production in 1957 came from three mines, the Konrad, Lubichow, and the Leona and amounted to 1,256,000 tons of ore, containing 0.61 percent copper. Full capacity of these three mines is scheduled for an annual output of 2.5 million tons of ore. The 1960 annual rate of copper production was about 10,000 tons.

Exploration revealed new reserves in the Lublin-Głogów area in Lower Silesia, and development of three mines in this area will start in the next five years. The first ore is expected from the Lublin mine in 1964 with production on an industrial scale scheduled for 1966. Development of the Polkowice I will start in 1963 and at the Polkowice II in 1965. These three mines will be in full production in the 1970's. The smelter and refinery planned to treat the Głogów ore will be put into operation by 1968–70.

Deposits near Głogów are of the sedimentary type in an area of about 200 square kilometers, containing an estimated reserve of about 10 million tons of copper. The average copper content of the ore has been estimated between 1.4 and 2.0 percent.

SPAIN

The copper-bearing massive pyrite deposits in southern Spain have been worked for more than 3,000 years and have been an important source of copper and pyrite since Roman times. The most important producer is the Compania Espanola de Minas de Rio Tinto, formerly a British company, but now controlled by the Spanish government. The Rio Tinto mining area is about 37 miles north of the port of Huelva in Huelva Province and accounts for more than 90 percent of the output of copper from Spanish mines. Cupreous pyrite is mined by surface (open cast) and underground

methods. Copper is recovered from the ore by leaching and cementation; by direct smelting to produce a copper matte and sulfur dioxide for making sulfuric acid; and by flotation followed by smelting. Reserves in 1958 were estimated at 4.5 million tons of copper (content of ore).

SWEDEN

Most copper mining in Sweden is carried on by the Boliden Mining Co., which operates several mines in a 200-mile strip from the Gulf of Bothnia in the east to the Lapland mountains in the west. The mines are grouped into four distinct areas near Boliden, Kristineberg, Adak, and Laisvall. There are three mines in the Boliden area—Boliden, Renström, and Långsele. Kristineberg is 60 miles west; Ravliden, a small producer, is 3 miles further west. The Adak district, 80 miles northwest of Boliden has two producing mines, the Adak and Rudtjebacken. These are State owned but operated by the Boliden company. The Laisvall mine is 140 miles west of Boliden, near the Lapland mountains where lead-bearing ore is mined.

The three areas of Boliden, Kristineberg, and Adak comprise the Skellefte District with mineralization occurring in Precambrian rocks. The principal ore bodies are massive sulfide ores—the most common being compact pyrite with subordinate pyrrhotite, chalcopyrite, sphalerite, and galena in varying amounts. Some of the ore bodies are in drag folds, while others are in shear zones or at the junction between bedding and shear zones. The dips of the ore bodies are generally steep and the wall rocks are mostly sericitic or chloritic schist. All the mines are small producers, working at shallow depths. In 1958 reserves were estimated at approximately 70 million tons or sufficient to insure 40 years production at present capacity.

U.S.S.R.

Principal copper deposits occur in five general areas, Kazakhstan, the Urals, Uzbekistan, Noril'sk, and Armenia. Quantitative data showing total copper production of the U.S.S.R. or any of its regions are not available. However, annual percentage increases to 1955 and results of the Fifth Five Year Plan have been published. This information formed the basis for developing production data in tonnage. Annual copper production increased 25 percent between 1955 and 1959 (from 385,000 to 480,000 tons), and the U.S.S.R. Seven Year Plan (1959–65) proposes to almost double 1959 production by 1965. The copper content of reserves in the Soviet Union was estimated to have been about 35 million tons at the

beginning of 1959. A reserve goal approaching 45 million tons by 1965 is anticipated in the Seven Year Plan. The quality of the reserves are not known and they may include low-grade protore uneconomical to mine at present values.

Kazakhstan, accounting for more than half of the U.S.S.R. output of copper ore, has producing mines in three areas, Dzhezkazgan-Karsakpay, Balkhash-Kounradskiy, and the Altai in East Kazakhstan. The Dzhezkazgan deposits are copper-impregnated sandstones, of the Rhodesian type, forming gently dipping lodes extending outward from feeder faults. The ore averages 1.62 percent copper and contains chalcopyrite, bornite, and chalcocite as the principal ore minerals. The most recent reserve estimate of 3.5 million tons of copper content is now considered too modest. The porphyry copper-type deposits in the Balkhash-Kounradskiy area are extensive and low grade. They consist of disseminated yellow sulfides and supergene chalcocite in small porphyry masses and invaded rocks; small quantities of molybdenum, gold, and silver are also present. The shallow enriched zones, averaging about 1 percent copper can be economically mined. The copper content of reserves in the region estimated at 2 million tons are believed to include areas of protore carrying only 0.5 to 0.7 percent copper. In the Altai region of eastern Kazakhstan, a major lead-zinc center, complex ores contain recoverable copper. Other main deposits in Kazakhstan are at Bozshakul' and Uspenkiy.

The principal copper mines in the Urals are at Krasnoural'sk, Kirovgrad, Degtyarsk, Karabash, Sibay, and Blyava. Copper-bearing pyritic lodes yield most of the copper ores mined in this region. The copper content of the ore is less than 3 percent, but it also contains extractable quantities of zinc and has values in gold and selenium. In 1950 the total reserve in the Urals was estimated at 4 million tons of copper contained in 246 million tons of ore. There are low-grade copper deposits in the western foothills of the Urals and in the Donets Basin that have not been developed. The Gay (section of Orsk) deposits discovered in 1958 contain five ore bodies having values in copper and zinc sulfides. Two of the ore bodies (1 and 3) now being developed average 4 percent copper; in some sectors the ores are 11 to 12 percent, and samples assaying 30 percent copper have been reported.

The Noril'sk district is east of the lowlands of the Yenisey River and borders the northwestern section of the Siberian massif. Magmatic copper-nickel deposits occur as intrusive stocks, sills, and dikes. Pyrrhotite, chalcopyrite, and pentlandite form low-grade dis-

seminations and veinlets in peridotite to form 1- to 3-percent copper ore, carrying 0.5 percent nickel and some of the platinum group metals.

In Armenia copper ore is mined on a small scale at four localities, Akhtala-Shamlug, Dastakert, Kadzharan, and Kafan. Copper reserves in the Caucasus although lowgrade are extensive (600,000 to 1,200,000 tons copper content).

In Northern Karelia and the Kola Peninsula are significant deposits of copper-nickel ore containing cobalt and platinum group metals. Reserves in central Asian U.S.S.R. are important. The largest at Almalyk, 56 miles southwest of Tashkent is reported to have a copper content of 3 million tons.

YUGOSLAVIA

Bor Mine.—Situated about 75 miles from Belgrade, this is the principal source of copper in Yugoslavia. Annual production of the Bor mine is about 30,000 tons of copper, scheduling operations at 4,000 tons per day of 1.8 percent copper ore and 500 tons per day of 5 to 6 percent copper ore. The important deposits at Bor are three massive bedlike ore bodies in altered andesite porphyry. The largest contains pyrite and enargite with minor amounts of luzonite and famatinite. The ore, enriched by secondary chalcocite and covellite, carries from 5 to 6 percent copper and some gold. The second ore body also yields 5 to 6 percent copper ore, and the third contains pyrite and chalcopyrite with a copper content of 1 to 2 percent.

Majdanpek Mine.—In the eastern part of Yugoslavia, this is actually in the northeastern part of the Republic of Serbia. It is about 75 miles from Belgrade, 9 miles from the Danube, and 25 miles north of the Bor mine. Exploration by the Yugoslavian Government in 1949 led to the development of extensive copper ore deposits at Majdanpek, which is one of the oldest mining districts in Yugoslavia, having about 2,000 years of mining history. This copper-bearing area is in the most northern part of the andesite mass of East Serbia. Magnetite, pyrite, and chalcopyrite occur as either fine disseminations or as irregular veins. Chalcopyrite is the principal copper ore mineral. Ore reserves of the Bor-Majdanpek combine have been estimated at 275 million tons averaging about one percent copper; Majdanpek accounted for 221 million tons, containing an average of 0.824 percent copper.

The Majdanpek mine is expected to produce 12,000 tons of ore per day for a recovery of 25,000 tons of copper annually. When the whole Bor-Majdanpek copper mining complex is put into operation in 1962, it is expected to produce annually 55,000 tons of copper, 3 tons

of gold, 34 tons of silver, 400,000 tons of pyrite concentrate, 230,000 tons of sulfuric acid, 575,000 tons of superphosphate, and 6,000 tons of synthetic cryolite.

Asia

CHINA

There are many copper occurrences in China. A Chinese source reports that discoveries of more than 3,000 copper deposits have been made mainly in Liaoning, Hopeh, Shansi, Kansu, Szechwan, Yunnan, Kweichow, and Anhwei Provinces. Intensive prospecting, exploration and development are credited for the more than tenfold increase in mine production and additions to resources of copper during the past decade. The most significant producing areas are in the southwestern provinces of Yunnan, Szechwan, and Kweichow.

Approximately 80 percent of the copper deposits in China are of four types: Bedded deposits, stringer-vein disseminated deposits, contact replacements, and copper-bearing pyrite. The Tung-ch'uan and I-men mines in Yunnan Province are in bedded deposits; the Chung-t'iao Shan mine in Shansi and the Te-hsing mine in Kiangsi are in stringer-vein disseminated deposits; the ore bodies of the T'ung-kuan-shan mine in Anhwei and the T'ien-pao-shan mine in Kirin are contact-replacements; the Pai-yen mine in Kansu exploits cupriferous-pyrite bodies; the deposits of the Chao-yüan mine in Shantung are of the vein type; the Kuang-t'ung mine in Yunnan, the Hui-li mine in Szechwan, and the Wei-ning mine in Kweichow work copper-bearing shale rock; and the deposit at the T'ao-k'o mine in Shantung is the copper-nickel type.

In 1959 the copper ore reserve of China was estimated between 3 to 5 million tons of copper (content of ore).

CYPRUS

Cyprus was perhaps the earliest important producer of copper smelted from sulfide ores. The metal was undoubtedly produced and delivered to the Egyptian kings before the Island was conquered by Thotmes III about 1500 B.C. The Roman occupation of Cyprus in 58 B.C. followed 14 centuries of control by Egyptians, Phoenicians, Assyrians, Persians, and the Ptolemies. Roman production was probably active until A.D. 200. There was no interest in metal mining from the Roman period until 1878 when Great Britain gained control of Cyprus. Casual prospecting was carried on between 1882 and 1914. More intensive exploration was resumed in 1919, and cupreous pyrite has been actively mined since 1922, except during World War II.

Skouriotissa Mine, Cyprus Mines Corp.— The property situated at the foot of Foucassa Hill was probably the largest of the ancient producers. The massive sulfide ore body is flat-lying, lenticular, roughly elliptical in plan, about 2,000 feet long, and as much as 600 feet wide—tapering from a thickness of a few feet at the edge to 150 feet near the center. It is intensely impregnated with sulfates of copper, iron, and zinc—including chalcanthite, brochantite, and melanterite. The ore reserve estimate as of December 31, 1960, was 2,750,000 tons, containing 2 percent copper and 40.0 percent sulfur.

Mavrovouni Mine, Cyprus Mines Corp.—The ore body is a large flat-lying body of cupreous pyrite. It is almost completely surrounded by andesitic lava that has been hydrothermally altered for hundreds of feet on all sides of the ore. The ore body shows extensive secondary alteration of chalcopyrite to chalcocite, covellite, and bornite. There are localized variations in copper content from 1 to 10 or 12 percent or more in bodies of sizeable tonnage. Crude ore production in 1960 totaled 873,000 tons, compared with 913,000 tons in 1959. The ore reserve at the end of 1960 was estimated to be 2 million tons, averaging 3.6 percent copper and 47.5 percent sulfur.

The Cyprus Sulphur and Copper Co., Ltd., a British company, produced cement copper from leach solutions at the inactive Limni mine from May 1958 until August 1959, when mining operations were resumed. In 1959 the company continued stripping operations at its Kinousa opencast mine, mining about 80,000 tons of ore—averaging 0.88 percent copper. Ore reserves at the end of March 1959 were 162,500 tons, averaging 0.26 percent copper at the Kinousa mine and 2,235,000 tons of 1.30 percent copper at the Limni mine.

INDIA

The Mosaboni and adjacent Badia mines of the Indian Copper Corp., Ltd., have been the only significant producing copper mines in India. Exploration and development are being conducted by the corporation at the Surda mine and Pathagarah prospect northwest of the present working area. The Mosaboni and other mines are in the Singbhūm district of the State of Bihār, 15 miles south of Ghatsila, 135 miles from Calcutta. Ore reserves as of December 31, 1958, were estimated at 4 million short tons, averaging 2.51 percent copper. The principal ore mineral is chalcopyrite, which occurs in veins in altered granite.

Occurrences of copper reported in various states are in Botang, near Rangpo in Sikkim; Khetri and Daribo in Rajasthan; Baragonda in the district of Hazaribagh; and Bairukhi in

Santhal Parganas, both in Bihār; Belliguda in the Chitaldrug district of Mysore, Almora; and the Tehri-Garhwal districts of Uttar Pradesh. Other than the Singbhum belt only the deposits in Sikkim and Rajasthan are said to be promising. Investigations of the deposits at Khetri and Daribo in Rajasthan by the Indian Bureau of Mines and Geological Survey have been encouraging.

ISRAEL

The Timma mine of the Israel Mining Industries, 15 miles north of the Red Sea port of Eilat, is in the area of the fabled King Solomon's mines. Operations began in April 1958 and in 1959, 495,000 tons of ore, containing 1.45 percent copper, was processed to yield 4,930 tons of copper. Proved reserves are estimated at 17 to 18 million tons of ore, averaging 1.4 percent copper. New deposits discovered in April 1960 are expected to double the potential ore reserve.

The Timmá deposit is a sedimentary one with copper mineralization diffused in sandstone and shaly sandstone rock. The copper occurs mainly in the form of chrysocolla (copper silicate). Ore is mined by the open-pit method and is leached with sulfuric acid. Copper is precipitated from the acid copper sulfate solution with scrap iron and recovered as cement copper, averaging about 80 percent copper.

JAPAN

Deposits of copper are widely distributed in Japan. A reserve estimate of 85 million tons of ore, averaging 1.4 percent, was made from a Japanese Government survey of 184 mines in 1956. Most of the ore bodies contain less than a few hundred thousand tons but they commonly occur in clusters. The bedded cupriferous pyrite-type deposit is the most productive in Japan. These are massive, pyritic replacements of green schist. The sulfide minerals present are pyrite, pyrrhotite, chalcopyrite, and sphalerite. Another type of replacement is known as black ore; these deposits contain galena, sphalerite, chalcopyrite, and pyrite replacing tuffs and shales. Many Japanese mines obtain copper from fissure veins containing chalcopyrite associated with pyrite or pyrrhotite. Some of the larger Japanese mines are:

Besshi Copper Mine.—This mine of the Sumitomo Metal Mining Co., Ltd., is in Ehime-ken in Shikoku. The ore is cupriferous pyrite, averaging 2.16 percent copper in a reserve of about 7 million tons.

Kosaka Mine.—Operated by the Dowa Mining Co., Ltd., this is at Kazuno-gun, Akita-ken. In 1913 this property was considered the largest copper mine in Japan. Mining was suspended in 1946 because all workable ore had been depleted; however, copper is being recovered by leaching and cementation. The reserve is estimated at 20 million tons of ore with a grade of 0.4 percent copper. Annual production is about 1,000 tons of copper.

Hanoka Mine, Dowa Mining Co., Ltd.—This is 2.5 miles north of Odate station, Akita-ken, and is known as the largest typical massive kuroko or black ore deposit in Japan. The deposit consists of more than 10 irregularly massive ore bodies distributed in an area 3 miles long and 1½ miles wide. Ore minerals are copper, lead, zinc, and iron sulfides with some gold and silver. The ore reserve as of September 1958 was 1.85 million tons, containing 1.2 percent copper.

The Mitsubishi Metal Mining Co., Ltd., operates 14 mines and produces about 20,000 tons of 21 percent copper concentrates annually. Eleven of these mines have narrow vein deposits, and three work massive, cupriferous pyrite deposits.

Hitachi Mine.—The Nippon Mining Co., Ltd., property is in Ibaraki-ken. The deposit consists of about 60 cupriferous pyrite ore bodies occurring in an area 2,500 meters long and 1,000 meters wide. The company operates 17 other mines in various sections of Japan, and the ore reserves of all operating properties in April 1957 were about 40 million tons.

Ashio Copper Mine.—This famous mine, operated by the Furukawa Mining Co., Ltd., is about 70 miles north of Tokyo in the Tochigiken. The deposit is one of the largest of the quartz-vein type in Japan and has been worked since 1620. Besides the true fissure veins, irregular deposits of chalcopyrite called kajika are found at the intersection of numerous radial veinlets or near the axis of folding in sedimentary rocks. Production amounted to 28,500 tons in 1959.

PHILIPPINES

Copper mining replaced the production of gold as the leading mineral industry of the Philippines in 1959 owing to the expansion of operations by the four major producers.

Toledo Mine.—This open-pit mine of the Atlas Consolidated Mining & Development Corp. in Cebu is the largest in the Far East. It accounted for 43 percent of the total copper output in the Philippines in 1959. Almost 4 million tons of ore, averaging 0.62 percent copper, was delivered to the mill for processing. The known ore body is 1,000 feet wide and 400 feet deep and is a disseminated porphyry type deposit. Chalcopyrite is the principal ore mineral. The total reserve at the end of 1959

was 10 million tons of ore with a grade of 0.695 percent copper.

Lepanto Mine.—The Lepanto Consolidated Mining Co. is in the Mountain Province of the Philippines. It is the largest vein copper mine in the country and ranks second in production. The ore body consists of a stockwork of veins and veinlets in a block 1,500 meters long, 30 meters wide, and 50 to 150 meters high. Luzonite, enargite, and precious metals are the ore minerals. The grade varies greatly but only material about 2 percent copper is considered ore. Ore reserves at the end of 1959 totaled 4.7 million tons averaging 3.39 percent copper and 0.129 ounce of gold per ton.

Sipalay Mine.—This property of the Marinduque Iron Mines Agents, Inc., is an open-pit operation of a large low-grade disseminated deposit on the west coast of Negros Occidental. Measured and indicated reserves in 1959 totaled 20.6 million tons of ore, averaging 0.79 percent copper.

Bagaycay.—The property on Samar Island, also owned by the Marinduque Iron Mines Agents, Inc., was discovered in 1955, and mining started in June 1956. One type of ore occurs in carbonaceous shale and in low-rank coal. Ore reserves at Bagaycay in 1959 were about 32,000 tons of direct shipping ore, containing 17.14 percent copper and 3.2 million tons of milling grade ore, averaging 3.55 percent copper.

Santo Tomas II Mine.—Near Baguio, Mountain Province, and owned by the Philex Mining Corporation, the mine started production July 5, 1958, and 1 million tons of ore was milled by the end of May, 1960; open-pit mining produced 800,000 tons and underground block-caving 200,000 tons. Ore reserves on January 1, 1960, were estimated at 22 million tons having a copper content of 0.81 percent. Most of the ore was found in a highly shattered zone of andesite alongside a large tongue of blocky diorite.

TURKEY

Copper production in Turkey is derived almost wholly from two mines which are owned and controlled by the Government. From 1936 through 1950 the Ergani deposit was the principal source of copper output; production at the Murgul mine began in 1951.

Ergani Mine, Mill, and Smelter.—These are in the Elazig II near Maden on the west bank of the Tigris River. Reserves consist of about a million tons of high-grade direct smelting ore in massive pyrite, averaging 10 percent copper and 3 to 4 million tons of disseminated ore underneath the massive pyrite that carries 4 to 5 percent copper. The ore body is bordered by serpentine to the south and is in contact

with schist or diabase to the north. It is oval in horizontal section, the long axis is about 360 meters, and the short axis is 170 meters. The ore minerals are pyrite, chalcopyrite, covellite, and some chalcocite.

Murgul Mine.—This is in the Coruh II near the village of Damar, southeast of the Black Sea Port of Hopa. The deposit consists of several irregular, subhorizontal lenses and stockworks of veins in a thick complex of andesite dacite. The largest lense is Cankara, others are the Sosveni and Satep. The Cankara ore body is 300 meters long, 50 to 80 meters wide, and increases in size with depth. The Sosveni measures 100 by 50 meters in the open pit and the Satep 100 by 60 meters. Ore reserves have been estimated at 16 million tons of about 2 percent copper. Ore minerals are pyrite and chalcopyrite occurring as a stockwork in dacite. The ore is mined by the open-pit mining method at a rate of 1,500 tons per day.

Africa

ANGOLA

The Empresa de Cobre de Angola, a subsidiary of Companhia Uniao do Fabril of Lisbon, Portugal, operates the only producing copper mine. The property is in northwestern Angola at Mavoio in the Uíge District in the Congo area. Production in 1960 amounted to 3,500 tons in ore, matte, and ingots. The ore mined averages between 10 to 14 percent copper. A water-jacket smelting furnace near the mine has a daily capacity of 5 tons of ingots.

REPULIC OF THE CONGO

Copper produced in the Republic of the Congo comes from concessions of the Union Minière du Haut-Katanga in Katanga Province.

The important mines are concentrated in an area about 200 miles long paralleling the Northern Rhodesia border. Producing units are referred to as belonging to the western group around Kolwezi, the central group near Jadotville, and the southeast group near Elizabethville.

Kamoto.—This is an open-pit mine in the western group, producing copper-cobalt ore. The ore body consists of flat-dipping beds of copper oxide ores. Outcrops are cellular quartzite containing vugs and seams of malachite. This mine is the principal supplier of siliceous oxidized ores for the Kolwezi concentrator.

Musonoi.—This open-pit mine of the western group operated only 4 months in 1961 when it furnished the major portion of mixed oxide-sulfide ores for the Kolwezi concentrator. Thereafter operations consisted of stripping overburden, mainly in the southern zone.

Ruwe Mine.—This is an open-pit operation in the western group. The ore body consists of deeply buried flat-lying shale beds containing nodules and seams of malachite. Zones of secondary enrichment occur as veinlets and small masses in the underlying breccia. The pit was extended to the south to mine the breccia in the lower levels. Stripping overburden from the two south extensions was continued.

Kolwezi Mine.—Located in the west, the mine is an open-pit operation. Mining in 1961 was sporadic due to construction of a skip hoist. Stripping overburden from the south end of the mine continued.

Kipushi (Prince Leopold) Mine.—This is an underground mine and is the principal producer in the southeastern group. The deposit is a pipelike sulfide lode along a faulted contact between limestone and members of the Mines series. The lode is 500 to 600 meters long, 20 to 60 meters wide, and dips steeply. It is rich in copper, zinc, and iron sulfides—containing values in silver, cadmium, and germanium. The ore body averages 11 percent copper and 19 percent zinc. The principal ore minerals are chalcopyrite, bornite, and sphalerite. A new top-slicing technique using metal props has been extended to the whole mine.

Kambove-West Open-Pit Mine.—Found in the central group this is being developed and is expected to be a substantial producer. Construction of the new concentrator near Jadotville to process Kambove ore was completed in April 1961.

Ore reserves of copper at the mines in the Congo are not available. The deposits are extensive, however, and the total reserve is believed to be large. An estimate of 20 million tons of recoverable copper is considered to be conservative.

FEDERATION OF RHODESIA AND NYASALAND

The Federation advanced to second place in world mine production of copper in 1959 and maintained that rank in 1960 with record mine output in both Northern and Southern Rhodesia. (The Federation ceased to exist on December 31, 1963, dividing into the Protectorate of Northern Rhodesia and the Colony of Southern Rhodesia.)

Northern Rhodesia

Six companies operate six underground and two open-pit mines within the Northern Rhodesian Copperbelt. These companies fall into two groups, one controlled mostly by British and American interests (Rhodesian Selection Trust, Ltd.) and the other predominantly by South African interests (Anglo-American Corporation of South Africa, Ltd.).

Anglo-American Corp. Group:
  Bancroft Mines, Ltd.
  Nchanga Consolidated Copper Mines, Ltd.
  Rhokana Corporation, Ltd.

Rhodesian Selection Trust Group:
  Chibuluma Mines, Ltd.
  Mufulira Copper Mines, Ltd.
  Roan Antelope Copper Mines, Ltd.

The Copperbelt has been described in great detail by a number of geologists. Briefly, the copper deposits are found in a number of synclinal structures along a belt 140 miles long and about 40 miles wide. They occur as disseminations in sedimentary shale and sandstone. These copper-bearing beds range from a few multiples of 10 to 100 feet in thickness; they extend on strike for a few thousand feet to a few miles and persist for at least several thousand feet in depth. The ore minerals are chalcocite, bornite, and chalcopyrite with minor amounts of other metallic minerals. Most of the ore bodies consist of sulfide ore but there is some mixed oxide-sulfide ore in places, and considerable oxide ore is mined in certain sections.

Bancroft Mines, Ltd.—The property is at the northwest end of the Northern Rhodesian Copperbelt. It is bounded on the north by the Belgian Congo border and on the south by Nchanga Consolidated Copper Mines, Ltd. Ore reserves June 30, 1959, were approximately 105 million tons, averaging 3.82 percent copper in three ore bodies:

	Short tons	Copper, percent
Kirila Bomwe South ore body	52, 122, 200	4. 5
Kirila Bomwe North ore body	20, 690, 900	4. 18
Konkola ore body	32, 073, 400	2. 48
Total	104, 886, 500	3. 82

Production was resumed on April 1, 1959, after a stoppage of 1 year, and by the end of the year output was at the annual rate of 55,000 tons of copper.

Nchanga Consolidated Copper Mines, Ltd.—This is the largest producer of copper in the British Commonwealth. Ore comes principally from the Nchanga West underground mine, but two open-pit mines—the Nchanga and the Chingola—are in operation. Ore reserves on March 31, 1960, totaled 180 million tons, containing 4.65 percent copper.

	Short tons	Copper, percent
Nchanga West and Nchanga ore bodies	163, 556, 000	4. 64
Chingola ore body	14, 183, 000	4. 89
Nchanga River lode	2, 280, 000	4. 21
Total	180, 019, 000	4. 65

Rhokana Corp., Ltd.—The corporation operates the Nkana mine near Kitwe. Ore is mined from three ore bodies where reserves were estimated to total 120 million tons with 3.07 percent copper as of June 30, 1960:

	Short tons	Copper, percent
Nkana North ore body	24, 328, 000	3. 06
Nkana South ore body	14, 913, 000,	2. 62
Mindola ore body	81, 106, 000	3. 15
Total	120, 347, 000	3. 07

Chibuluma Mines, Ltd.—The property is in the Nkana South Limb area about 7 miles west of Kitwe and about 34 miles southwest of the Mufulira mine. Ore reserves at June 30, 1960, were:

	Short tons	Copper, percent	Cobalt, percent
Chibuluma main ore body	7, 420, 000	5. 03	0. 21
Chibuluma West	2, 370, 000	4. 47	. 07
Total	9, 790, 000	4. 89	0. 18

Mine production for the year ending June 30, 1960, was 575,436 tons of ore, averaging 4.65 percent copper and 0.23 percent cobalt; 76,362 tons of copper concentrate were treated by Mufulira Copper Mines, Ltd., yielding 26,465 tons of copper; 36,524 tons of cobalt copper concentrate was treated at the Ndola plant, producing 9,778 tons of cobalt matte, containing 9.46 percent cobalt and 11.61 percent copper. The cobalt matte is shipped to Société Générale Metallurgique de Hoboken in Belgium for refining.

Mufulira Copper Mines, Ltd.—The company operates the second largest Copperbelt mine and the largest in the Rhodesian Selection Trust group. The property is 26 miles northeast of Kitwe. There are three parallel mineralized beds at Mufulira, separated by 25 to 45 feet of waste. The two lower beds coalesce toward the northwest to give a thickness of more than 100 feet in places. Ore reserves on June 30, 1960, were 179 million tons, averaging 3.35 percent copper. A major expansion of the producing capacity of Mufulira was being developed at Mufulira West and was designed to increase capacity by 50 percent.

Roan Antelope Copper Mines, Ltd.—The deposit is in the southwest corner of the Copperbelt near the town of Luanshya. The deposit is large and is important for the great length of ore developed around the nose and along the limbs of the fold in an ore bed 25 to 30 feet thick. Ore reserves at June 30, 1960, totaled 94.6 million tons of 3.04 percent copper. Mine output for the year ending June 30, 1960, was 6.66 million tons of ore, averaging 1.85 percent, and production of copper was 103,422 short tons.

Southern Rhodesia

The areas north and west of Sinoia contain many deposits of copper minerals, and it is from this belt that future production is expected. Copper development in Southern Rhodesia has been controlled by The Messina (Transvaal) Development Co., Ltd., (incorporated in the Union of South Africa) directly or through its subsidiaries M.T.D. (Mangula), Ltd., and M.T.D. (Sanyati), Ltd.

Mangula.—The largest copper mine in Southern Rhodesia is in the Lomagundi District, 30 miles north of Sinoia and 115 miles northwest of Salisbury. The ore is disseminations of sulfide minerals, mainly bornite and chalcocite, in folded structures of arkoses and shales. Mineralization extends for 3,000 feet with a width at the center on the surface of 300 feet. Ore reserves in 1960 were 26 million tons, averaging 1.36 percent copper.

Other important deposits in this area are the replacement ore bodies of the Copper King and Copper Queen mines in the Sebungwe District, 60 miles west of Sinoia; the Alaska mine 12 miles west-southwest of Sinoia, where copper minerals occur in a replacement deposit in dolomite, and quartzites and ore reserves are estimated at 5 million tons of 1.87 percent copper; and the copper-gold carbonate lodes 32 miles west of Sinoia. Another producing property is the Umkondo mine in the Sabi valley, Bikita district, where copper minerals occur in beds of shale and quartzite. Umkondo reserves on September 30, 1959, were 360,160 tons of ore, containing 3.35 percent copper, sulfide, and oxide.

KENYA

The Macalder-Nyanza copper mine, operated by Macalder-Nyanza Mines, Ltd., south of Nyanza near Lake Victoria, produced 1,855 tons of copper in 1960. It has been estimated that the known ore body will support this rate of production for approximately 12 years.

MAURITANIA

The principal copper deposit in Mauritania, known as the Guelb Moghrein mine, is about 2.5 miles from Akjoujt. About 9 million tons of oxidized ore, averaging 2.3 percent copper, is in the upper part of the ore body, and almost 17 million tons of cupriferous pyrite with 1.5 percent copper is in the lower part. The deposit is being developed by Société des Mines de Cuivre de Mauritanie (MICUMA). Lack of water and mineral composition present problems in concentration of the ore.

SOUTH-WEST AFRICA

The Tsumeb mine of the Tsumeb Corp., Ltd., is the major source of copper. Copper minerals—including tennantite, enargite, chalcocite, and bornite—are associated in substantial amounts with lead and zinc ores in massive replacement lenses in dolomite. Ore reserves above the 3,150-foot level on June 30, 1959, totaled 8,165,000 tons—averaging 5.29 percent copper, 14.36 percent lead, 4.45 percent zinc, and 0.013 percent germanium. Drilling below the 3,150-foot level has indicated a minimum of 2 million additional tons of ore—averaging 4.6 percent copper, 7.1 percent lead, and 1.9 percent zinc. Exploratory drilling on the Tsumeb Asis claim near Kombat has disclosed two ore bodies of about 1 million tons each, averaging 11 percent combined copper and lead; production started in April 1962.

The Tsumeb Corp., Ltd., was erecting a copper smelter at Tsumeb which was completed and began production of blister copper in November 1962.

UGANDA

The Kilembe mine, operated by Kilembe Mines, Ltd., is in the foothills of Mount Ruwenzori in the Toro district of Western Uganda. Two ore bodies are being worked. Both deposits are irregularly shaped and occur over a strike length of about 7,000 feet and an average thickness of 40 feet. The upper part of the East deposit is near the surface and is mined by the open-pit method; the lower section of the East deposit and the North deposit are mined by underground methods. The ore contains a wide variety of minerals which occur as disseminations, stringers, seams, and small blebs within the granulitic host rock. Ore minerals are cuprite, malachite, chrysocolla, and chalcocite in the upper zone and chalcopyrite, pyrite, pyrrhotite, and linnaeite in the lower. Ore reserves estimated at the end of 1958 were 8.2 million tons, containing 2.31 percent copper and 0.18 percent cobalt, and 1.57 million tons, containing 1.23 percent copper and 0.22 percent cobalt. Blister-copper production was 17,200 tons in 1962.

REPUBLIC OF SOUTH AFRICA

Most of the copper production of the Republic of South Africa comes from two widely separated deposits, the O'okeip properties near Springbok in Namaqualand (Cape of Good Hope Province) and the Messina mine in the Northern Transvaal near the Southern Rhodesian border.

The principal mines operated by the O'okeip Copper Co. are the Nababiep, East O'okeip, Narrap, Wheal Julia, Nababiep West, and the Carolusberg. The ore bodies occur in metamorphic rocks that consist chiefly of paragneiss, granulite, quartzite, and schist. Mixtures of the oxide minerals, brochantite, chrysocolla, and malachite show in some of the outcrops. Below the oxidized zone, usually less than 70 feet, are bornite, chalcopyrite, and chalcocite. These minerals occur in finely disseminated form, in coarser aggregates with interconnecting veinlets, and in massive form. Sulfide ore reserves on June 30, 1959, totaled 27.8 million tons, averaging 2.19 percent copper.

Artonvilla, Campbell, Harper, and Messina.—In Transvaal, these four producing mines, operated by Messina (Transvaal) Development Co., Ltd., are near the town of Messina, 6 miles south of the Limpopo river which forms the boundary between the Union of South Africa and Southern Rhodesia. The ore bodies are veins and breccia pipes lying in a zone striking northeast for about 10 miles along the Messina fault. The principal ore minerals are chalcopyrite, bornite, and chalcocite. Ore reserves on September 30, 1961, totaled 6,003,810 tons, containing 1.57 percent copper. The Campbell mine presently accounts for about half of the ore produced.

Palabora Mining Co., Ltd.—Controlled by the Rio Tinto Co., Ltd., of London and the Newmont Mining Corp. of New York, this company is developing a long known copper deposit at Loolekop, Palabora, northeast Transvaal about 75 miles east of Pietersburg. Drilling confirms the existence of a low-grade ore body suitable for open-pit mining. Ore reserves have been estimated at 5.7 million tons of 1.26 percent copper and 24.9 million tons of 0.51 percent copper per hundred feet of depth. Production of 2,265 tons of copper in concentrates was reported in 1961 for the first time.

Australia

During the 1950's Australia became one of the major world sources of copper. Mine production of 1950 had increased eightfold to 122,000 tons by 1960, mainly because of continuing expansion at the Mt. Isa mine. Four mines produce approximately 90 percent of Australian copper. The balance is obtained from small copper mines and byproduct recoveries at the large silver-lead-zinc mines.

Mt. Isa Mine.—This property of the Mt. Isa Mines, Ltd.—in which the American Smelting and Refining Co. of the United States has a controlling interest—is the largest producer, accounting for 60 percent of the 1961 output from Australia. The deposits are situated at Mount Isa, 600 miles west of Townsville, in

Queensland. The host rock for the ore bodies is known broadly as a silica-dolomite rock, which consists of brecciated dolomitic and pyritic shales with massive and, at times,, structureless siliceous and dolomitic rocks. The chief ore mineral is chalcopyrite, occurring as fracture and vein filling in the silica-dolomite bodies. Oxidized and secondarily enriched copper ores occur in the oxidized zone near the surface. The estimated reserves of copper ore on June 30, 1961, were 24.5 million tons, averaging 3.7 percent copper.

Mount Morgan Mine.—Operated by Mount Morgan, Ltd., it is also in the State of Queensland, 23 miles southwest of Rockhampton, on the Dee River. The ore-body, including the Sugarloaf section, is an irregular quartz-pyrite mass, having a maximum length of 2,100 feet and a maximum width of 900 feet. The principal ore mineral is chalcopyrite. Production in 1961 was 9,300 short tons of copper and 54,000 ounces of gold in concentrates. Reserves as of June 30, 1961, amounted to 13.8 million tons of ore, containing 0.116 ounce gold per ton and 1.10 percent copper.

Mount Lyell.—This group of mines is about 18 miles from Macquarie Harbour on the west coast of Tasmania. The mines are in a narrow belt extending from the northern side of Mount Lyell to the northern side of Mount Owen on the south, a distance of 2½ miles. The ore deposits are associated with the schist-conglomerate contact. Massive sulfide ore bodies occur on or near the contact and low-grade disseminated ore bodies away from the contact. The ore consists mainly of chalcopyrite with some bornite. Ore reserves on June 30, 1962, were 23.5 million tons, averaging 0.80 percent copper. Mine production of copper in fiscal 1962 was 12,332 tons.

Peko Mine.—This property of Peko Mines N.L. at Tennant Creek is the only copper mine in Northern Territory. The ore body or lode contained some copper in the oxidized zone; high-grade copper ore, containing native copper, cuprite, and malachite was found near the water table. Primary ore—containing 4 to 11 percent (average 8.5 percent) copper and 0.05 to 1.25 ounce gold per ton—occurred at 350 feet and continued in depth. The copper occurs chiefly as chalcopyrite, while the gold is fine grained and is closely associated with the chalcopyrite. Ore reserves in mid-1959 were estimated at 1 million tons with an average of 6.3 percent copper. Production of copper in 1959 was 7,500 tons in concentrates.

Foreword

The Materials Survey series being prepared by the Bureau of Mines, Department of the Interior, under the sponsorship of the Office of Emergency Planning, is designed to present in separate documents a compilation of comprehensive fundamental information for those metals and minerals essential to National Security. These Surveys summarize the demand-supply position in the United States and include information on production, consumption, imports, exports, capacities, substitutes, and pertinent history, usually in some detail back to 1925. The properties and uses of the commodity and its principal alloys and compounds are described. Exploration, mining, metallurgical, and fabrication methods are discussed. Domestic and foreign primary and secondary resources and reserves are covered. An extended presentation of the structure of the industry, employment and productivity, research and development, legislation, taxes, and Government wartime controls is included. Other special data are presented for particular commodities.

This Copper Materials Survey was prepared in the Division of Minerals under the direct supervision of Paul F. Yopes, Chief, Branch of Nonferrous Metals, and Donald R. Irving, Assistant to the Chief, Division of Minerals. The manuscript was reviewed, in whole or in part, by specialists in the Bureau of Mines, Geological Survey, and various segments of the copper industry.

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BIBLIOGRAPHY

The Anaconda Company. This is Anaconda. New York, 1960, 59 pp.

Bureau of Mines. Copper. MS 6, 1952, 556 pp.

Butler, B. S., and S. W. Burbanks. The Copper Deposits of Michigan. U.S. Geol. Survey Prof. Paper 144, 1929, p. 101.

Butts, Allison. Copper, The Metal, Its Alloys and Compounds. Reinhold Pub. Corp., New York, 1954, pp. 21–62.

Callaway, H. M. Lead, A Materials Survey. BuMines Inf. Circ. 8083, 1962, 194 pp.

Edwards, A. B. Geology of Australian Ore Deposits. 5th Empire Min. and Met. Congr., Australia and New Zealand, v. 1, 1953, 1290 pp.

Gardner, E. D., C. H. Johnson, and B. S. Butler. Copper Mining in North America. BuMines Bull. 405, 1938, 300 pp.

Geological Survey of Finland. The Mines and Quarries of Finland. 1954, 123 pp.

Grant, M. North. The Copper Industry in Yugoslavia. Copper Division BDSA, U.S. Dept. of Commerce, v. 5, No. 2, Autumn 1958, pp. 10–11.

Hardwick, W. R. Open-Pit Mining Methods and Practices at the Chino Mines Division, Kennecott Copper Corp., Grant County, N. Mex. BuMines Inf. Circ. 7837, 1958, 64 pp.

Killin, A. F. Survey of The Copper Industry in Canada. Miner. Inf. Bull. MR47, Miner. Res. Division, Dept. Mines and Tech. Surveys, Ottawa, Canada, 1961, 107 pp.

Lasky, S. G. Geology and Ore Deposits of the Lordsburg Mining District, Hidalgo County, N. Mex. U.S. Geol. Survey Bull. 885, 1938, 62 pp.

McWilliams, John R. Mining Methods and Costs at The Anaconda Company Berkeley Pit, Butte, Mont. BuMines Inf. Circ. 7888, 1959, 46 pp.

Mining Journal (London). Russian Copper Reserves. V. 255, No. 6541, Dec. 30, 1960, p. 736.

Muskiet, T., A. Bem, and T. Plodowski. Mining Activities in Poland. Min. and Quarry Eng., v. 25, No. 4, April 1959, pp. 166–168.

Nahai, Lotfollah. The Mineral Industry of Turkey. BuMines Inf. Circ. 7855, 1958, 140 pp.

Northern Rhodesia-Chamber of Mines. Yearbook. Rhodesian Printers Ltd., Salisbury, Southern Rhodesia, 1959, 93 pp.

Parsons, A. B. The Porphyry Coppers. AIME, New York, 1933, 581 pp.

Peets, Robert G., Mining History at Cornwall, Pa. Min. Eng., v. 9, No. 7, July 1957, pp. 741–744.

Pelletier, J. D. Geology of the San Manuel Mine. Eng. and Min. J., v. 9, No. 7, July 1957, pp. 760–762.

Peterson, Nels P., and Roger W. Swanson. Geology of the Christmas Copper Mine, Gila County, Ariz., Ch. in Contributions to Economic Geology, 1956. U.S. Geol. Survey Bull. 1027(h), 1956, pp. 351–373.

Popoff, C. C. Block Caving at Kelly Mine, The Anaconda Company, Butte, Mont. BuMines Inf. Circ. 7758, 1956, 102 pp.

Schmidt, Harrison A. The Copper Province of the Southwest. Min. Eng., v. 11, No. 6, June 1959, pp. 597–600.

Sixteenth International Geological Congress. Copper Resources of the World. V. 1 and 2, George Banta Pub. Co., Menasha, Wis., 1935, 855 pp.

Skinner, Walter E. Mining Yearbook 1961. Walter E. Skinner and Financial Times, London, 1961, 791 pp.

Smith, M. Clair. Methods and Operations at the Yerington Copper Mine and Plant of the Anaconda Co., Weed Heights, Nev. BuMines Inf. Circ. 7848, 1958, 37 pp.

Spence, W. I. The Messina Copper Mine. Paper for 7th Commonwealth Min. and Met. Cong., Union of South Africa, April 1961, pp. 1–19.

Thurmond, Robert E., and Walter E. Neiwicks, Jr. Geophysical Discovery and Development of the Pima Mine, Pima County, Ariz. AIME Transactions, Min. Eng., February 1954, pp. 197–202.

Tuck, Frank J. Stories of Arizona Copper Mines. Ariz. Dept. Min. Res., Phoenix, Ariz., 1957, 120 pp.

White, Walter S., and James C. Wright. The White Pine Copper Deposit, Ontonagan County, Mich. Econ. Geol., v. 49, No. 7, November 1954, pp. 676–716.

CHAPTER 4.—SECONDARY COPPER RESOURCES

A large reserve of secondary copper has been accumulated and is continually being augmented because of its indestructibility and the consumption pattern of the metal and its alloys. Every use of new copper in any form becomes a potential part of this resource except those few applications where reclamation is impossible, for example, copper in chemical compounds used as fungicides and copper and brass powders employed in making paint pigments. Each year this reservoir provides approximately one fourth of the copper consumed in the United States in unalloyed and alloyed products. The collection and processing of copper and copper-base alloy scrap into secondary metal constitutes an important segment of the copper industry in all major consuming countries.

NATURE

Copper recovered from copper scrap, copper-alloy scrap, and other copper-bearing scrap materials as copper metal, or as copper content of alloys, chemicals, or compounds is known as secondary copper. As defined, the term includes the copper content of copper-base and other copper-bearing alloys recovered in alloy form, as well as refined copper produced in processing various kinds of scrap. Some brass scrap is smelted for separate recovery of its copper content; however, a large percentage of reclaimed brass scrap is remelted into brass ingots that are sold as brass in the secondary market. Inasmuch as the copper content of such brass substitutes for primary copper in industrial and commercial products of consumption, it contributes to the available supply of copper and may validly be covered by the term "secondary copper." The same reasoning is applicable to other copper alloys and chemicals.

KINDS OF SCRAP

Secondary copper is recovered from two principal classes of scrap, these are old and new. However, only old scrap is available from the accumulating reserve of secondary copper through reclamation of copper, brass, bronze, and other alloy products that have been used and then discarded because they are obsolete, worn out, or damaged. Such articles are wire, fired cartridge cases, pipe, automobile radiators, bearings, valves, screening, lithographic plates, etc.

The new scrap generated in fabricating and manufacturing semifinished and finished products from primary copper and copper-base-alloy forms does not form a reservoir supply to supplement production of primary copper. Examples of new scrap are defective castings, clippings, punchings, turnings, borings, skimmings, drosses, and slag. Such scrap represents a circulating quantity of copper previously accounted for as supply of primary copper and returned to the fabricating process without reaching the product stage. However, data on the movement of new scrap have significance as indicators of business activity in fabricating and scrap reclamation industries.

SOURCES

Collection of scrap is carried on in almost every community. In many there are collectors or dealers that sort, bale, and otherwise prepare the salvaged scrap for shipment and sale to primary smelters, secondary smelters, ingot makers, brass mills, chemical plants, or other fabricators. However, the principal sources of copper and copper-alloy scrap are the heavily populated industrial centers; most of the plants engaged in recovering secondary copper are located in these areas. Distribution of the primary and secondary smelters and brass mills (1958) producing most of the secondary copper in the United States is as follows:

State:

Primary and secondary smelters:	No. of plants
Illinois	15
Ohio	12
New York	12
New Jersey	10
California	9
Massachusetts	5
Michigan	5
Brass mills:	
Connecticut	15
Illinois	6
Michigan	6
Pennsylvania	6
New Jersey	5
New York	5
California	3

Types and quantities of old scrap consumed by producers of secondary copper, 1956-60, are shown in table 9.

TABLE 9.—*Consumption of old scrap, short tons*

Type of scrap	1956	1957	1958	1959	1960	Total	1956–60 (average)	Percent
Aluminum bronze	1,240	925	833	1,142	843	4,983	997	0.16
Auto radiators (unsweated)	57,815	51,819	46,682	51,561	44,111	251,988	50,398	8.15
Bronze	24,758	21,665	18,659	23,926	23,408	112,416	22,483	3.64
Cartridge cases and brass	16,607	14,139	11,634	15,964	14,340	72,684	14,537	2.35
Composition or red brass	71,449	59,074	55,343	65,737	58,793	310,396	62,079	10.04
Low brass	2,242	2,841	2,038	1,575	1,311	10,007	2,001	.32
Low-grade scrap and residues	142,887	149,540	149,771	127,537	144,346	714,081	142,816	23.11
Nickel silver	3,395	2,799	2,637	2,638	2,913	14,382	2,877	.47
No. 1 wire and heavy copper	86,689	87,368	83,636	110,322	90,302	458,317	91,663	14.83
No. 2 wire and mixed heavy and light copper	114,158	99,540	111,872	121,609	117,071	564,250	112,850	18.26
Railroad-car boxes	63,174	55,931	44,895	56,674	48,727	269,401	53,880	8.72
Yellow brass	68,913	62,810	58,803	63,236	53,636	307,398	61,480	9.95
Total	653,327	608,451	586,803	641,921	599,801	3,090,303	618,061	100.00

MAGNITUDE OF INDUSTRY

Near the beginning of the 20th century, the secondary metal industry comprised a group of independent junk collectors and dealers who gathered, sorted, and sold scrap metals and waste materials to relatively uncertain markets. As more scrap metal sources were developed and collection became better organized, some of the collectors and dealers sought to increase their margin of profit by remelting their scrap and producing commercial ingots. These ingots were generally poor in grade, by present standards. Having no precedent with which to base expansion and improvements, this embryonic industry added technical personnel and began educating itself in the art and technology of reclaiming values from scrap and waste materials. The quality of products and the degree of specialization in all segments of the industry soon improved considerably. Collecting, marketing, melting, refining, and alloying became individual operations, each contributing to the general advance of the industry as a whole.

As a result, the secondary copper industry has grown to be the largest of the nonferrous secondary metal industries. Approximately 25 percent of the total copper consumed in the United States is produced from old scrap. The industry consists of several thousand collectors and dealers, several hundred foundries, about eighty ingot makers and secondary smelters, around fifty brass mills, and a dozen primary smelters which process some scrap. These processors also recover an equivalent amount of copper from new scrap and the total from new and old scrap often approaches recoverable mine production.

ACCUMULATING POOL

Consumption of new copper in the United States from the beginning of recorded production in 1845 through 1960 totals approximately 51 million tons. In the period from 1908 through 1960, more than 16 million tons of secondary copper was recovered from old scrap. An important aspect of these data is that an ever increasing resource is being created that is capable of providing about one-third of the national requirements.

There is no statistical series showing the amounts of copper consumed in end-use items that would indicate the quantity in service in nondissipative uses; nor is there any related quantitative data about consumption and reclamation by an industry classification. Although it is generally known that more old scrap is salvaged from capital goods such as buildings, ships, railroads, bridges, and industrial machinery than from consumer goods such as automobiles, radios, and cooking utensils, there are no continuing statistics showing significant uses of copper in units of capital goods that would support the construction of an estimated reserve of copper in use.

However, table 10 presents an interesting development showing expansion of the copper-in-use pool in the United States from 3 million tons at the end of 1907 to almost 33 million tons at the end of 1960. These figures are based on the estimate that three-fourths of the new copper is consumed in the manufacture of reclaimable products.

Although adequate data for adjustment of the annual increments are not available, there are some factors that should be considered in the accumulation of the in-use pool. Examples of such factors are imports and exports of copper-bearing manufactured products and imports and exports of new and old copper scrap that are not included in calculating apparent consumption. However, the error introduced by exclusion of this foreign-trade material is believed to be consistent with the limits of accuracy in other portions of the problem.

WORLD RESERVE

The major uses of copper in the principal foreign consuming countries are about the same

TABLE 10.—*Estimated accumulation of copper-in-use in the United States, 1845-1960*

(thousands of tons)

Year	Consumption, apparent			Pool of copper-in-use	
	New copper	Old scrap	Total	Increase=75 percent of total minus old scrap	Total at end of year
1845-1907	[1] 6, 083				[2] 3, 000. 0
In use through 1907					
1908	240. 0	[1] 50. 0	290. 0	167. 5	3, 167. 5
1909	344. 3	[1] 55. 0	399. 3	244. 5	3, 412. 0
1910	366. 2	64. 5	430. 7	258. 5	3, 670. 5
1911	340. 9	76. 0	416. 9	236. 7	3, 907. 2
1912	388. 0	107. 0	495. 0	264. 2	4, 171. 4
1913	406. 1	91. 5	497. 6	281. 7	4, 453. 1
1914	350. 8	87. 9	438. 7	241. 1	4, 694. 2
1915	568. 3	121. 2	689. 5	395. 9	5, 090. 1
1916	739. 4	175. 0	914. 4	510. 8	5, 600. 9
1917	697. 4	194. 9	892. 3	474. 3	6, 075. 2
1918	830. 8	176. 7	1, 007. 5	578. 9	6, 654. 1
1919	457. 2	152. 6	609. 8	304. 8	6, 958. 9
1920	526. 9	169. 0	695. 9	352. 9	7, 311. 8
1921	305. 5	132. 0	437. 5	196. 1	7, 507. 9
1922	448. 3	202. 8	651. 1	285. 5	7, 793. 4
1923	650. 2	270. 9	921. 1	419. 9	8, 213. 3
1924	677. 4	266. 2	943. 6	441. 5	8, 654. 8
1925	700. 5	291. 0	991. 5	452. 6	9, 107. 4
1926	785. 1	337. 3	1, 122. 4	504. 5	9, 611. 9
1927	711. 5	339. 4	1, 050. 9	448. 8	10, 060. 7
1928	804. 3	365. 5	1, 169. 8	511. 8	10, 572. 5
1929	889. 3	404. 4	1, 293. 7	565. 8	11, 138. 3
1930	632. 5	342. 2	974. 7	388. 8	11, 527. 1
1931	451. 0	261. 3	712. 3	272. 9	11, 800. 0
1932	259. 6	181. 0	440. 6	149. 4	11, 949. 4
1933	339. 4	260. 3	599. 7	189. 5	12, 138. 9
1934	322. 6	310. 9	633. 5	164. 2	12, 303. 1
1935	441. 4	361. 7	803. 1	240. 6	12, 543. 7
1936	656. 2	382. 7	1, 038. 9	396. 5	12, 940. 2
1937	694. 9	408. 9	1, 103. 8	419. 0	13, 359. 2
1938	407. 0	267. 3	674. 3	238. 4	13, 597. 6
1939	714. 9	286. 9	1, 001. 8	464. 4	14, 062. 0
1940	1, 008. 8	333. 9	1, 342. 7	673. 1	14, 735. 1
1941	1, 641. 6	412. 7	2, 054. 3	1, 128. 0	15, 863. 1
1942	1, 608. 0	427. 1	2, 035. 1	1, 099. 2	16, 962. 3
1943	1, 502. 0	427. 5	1, 929. 5	1, 019. 6	17, 981. 9
1944	1, 504. 0	456. 7	1, 960. 7	1, 013. 8	18, 995. 7
1945	1, 415. 0	497. 1	1, 912. 1	937. 0	19, 932. 7
1946	1, 391. 0	406. 5	1, 797. 5	941. 6	20, 874. 3
1947	1, 286. 0	503. 4	1, 789. 4	838. 7	21, 713. 0
1948	1, 214. 0	505. 5	1, 719. 5	784. 1	22, 497. 1
1949	1, 072. 0	383. 5	1, 455. 5	708. 1	23, 205. 2
1950	1, 447. 0	485. 2	1, 932. 2	964. 0	24, 169. 2
1951	1, 304. 0	458. 1	1, 762. 1	863. 5	25, 032. 7
1952	1, 360. 0	414. 6	1, 774. 6	916. 4	25, 949. 1
1953	1, 435. 0	429. 4	1, 864. 4	968. 9	26, 918. 0
1954	1, 235. 0	407. 1	1, 642. 1	824. 5	27, 742. 5
1955	1, 336. 0	514. 6	1, 850. 6	873. 3	28, 615. 8
1956	1, 367. 0	468. 5	1, 835. 5	908. 1	29, 523. 9
1957	1, 239. 0	444. 5	1, 683. 5	818. 1	30, 342. 0
1958	1, 157. 0	411. 4	1, 568. 4	764. 9	31, 106. 9
1959	1, 183. 0	471. 0	1, 654. 0	769. 5	31, 876. 4
1960	1, 148. 0	429. 4	1, 577. 4	753. 6	32, 630. 0
Total	51, 084. 3	16, 481. 7	61, 483. 0	29, 630. 0	

[1] United States production.
[2] Estimated.

Source: Merrill, C. W. The Significance of the Mineral Industries in the Economy. Ch. in Economics of The Mineral Industries, AIME, New York, 1959, p. 26.

as those in the United States with a similar
relationship existing between consumption of
scrap and total copper. Further, scrap is
collected, marketed, and processed in foreign
countries in much the same manner as it is in
the United States. This is particularly true
in Europe where 40 to 45 percent of the World
copper is consumed, principally in Great
Britain, West Germany, U.S.S.R., France, and
Italy. Because of this implied parallelism,
the accumulated copper that is recoverable
but still in use in the World is estimated by
applying the ratio of World to United States
consumption for the period 1908-60 to the
resource pool as developed for the United States.

$$\frac{\text{World secondary resource}}{\text{U.S. secondary resource}} = \frac{\text{World consumption}}{\text{U.S. consumption}}.$$

$$\text{World secondary resource} = \frac{32,630,000 \times 113,562,000}{45,001,300},$$

World secondary resource = 82,000,000 tons.

World consumption of 113,562,000 for the
period 1908-60 was compiled from various
sources. Smelter production is used for the
periods 1908-21 and 1939-45, as there are no
World consumption data for those years;
consumption data appearing in the American
Bureau of Metal Statistics 1933 yearbook are
used for 1922-25; Bureau of Mines data are
used for the other years. Although the con-
sumption statistics used in developing an es-
timate of the World secondary copper reserve
are not strictly comparable, the differences are
not great, and the result indicates the probable
magnitude of such a resource.

BIBLIOGRAPHY

Barbour, Percy E. Secondary Copper. Min. and Met. Soc. of America, New York, 1936, 85 pp.

Bureau of Mines. Mineral Resources. Copper Chapters, 1910–31.

Bureau of Mines. Minerals Yearbook. Copper Chapters, 1932–60.

Cochran, Ray. Secondary Copper and Copper Alloys. Ch. in Copper, The Metal, Its Alloys and Compounds, by Allison Butts, Reinhold Pub. Corp., New York, 1954, pp. 352–359.

Merrill, Charles White. The Significance of the Mineral Industries in the Economy. Ch in Economics of the Mineral Industries, AIME, New York, 1959, pp. 24–27.

Spendlove, Max J. Methods of Producing Secondary Copper, BuMines Inf. Circ. 8002, 1961, 41 pp.

CHAPTER 5.—TECHNOLOGY

Technology is defined as the science of systematic knowledge of the industrial arts. The expansion of nearly all phases of industry is due largely to development of the science through research and application of scientific knowledge through engineering. Management must continually seek improved technology to increase efficiency of operations and to lower costs.

PROSPECTING AND EXPLORATION, CURRENT ORE SEARCH TECHNIQUES

Prospecting is the search for ore occurrence, and exploration is the work involved in gaining knowledge of the size, shape, position, and value of an ore body. These two activities, although quite different in purpose, use the same tools. Many writers consider prospecting as a phase of exploration and refer to the search for ore as exploration for new ore bodies; others use the terms interchangeably; such as, geochemical exploration and geochemical prospecting or geophysical exploration and geophysical prospecting. The separation of prospecting from exploration is difficult, and consequently prospecting and exploration of copper deposits are treated as a unit in this report.

Historically, mineral deposits have been discovered by prospectors, local groups organized to prospect and explore a single property, and small companies. The most advanced techniques now used—comprising the expensive requirements of mapping, drilling, underground workings, sampling and assaying, metallurgical testing, and other related activities—are beyond the financial capacity of individuals or small companies. The major companies, for the most part, find operation of small mines uneconomic and generally do not become interested in a mining property or a mineralized area unless it promises to become a large-scale producer.

Two fundamental approaches to exploration are common, basic exploration and property acquisition, as well as combinations of the two in varying proportions.

Basic exploration as practiced by large exploration companies consists of several stages. First, an area is selected by a study of the regional geology, using photogeology as a tool. Next, the area often is mapped by airborne magnetic and electromagnetic methods on a fairly close spacing. Some reconnaissance geochemical work may also be carried out. Results obtained are used to determine the extent of stage three, detailed ground surveys. This includes checking of geophysical anomalies using corresponding ground techniques, detailed geochemical work and geological mapping and possibly shallow drilling, pitting, stripping, and trenching. The extent of the final drilling program is in turn conditioned by the quality of these results.

The favored exploration approach by some companies is the acquisition of prospects by option. It also fits into the basic exploration programs of others.

Generally the planning and organization of a large-scale exploration program consists of four main stages:

1. Obtaining an overall picture of the geology by aerial photography, photogeological interpretation, and field reconnaissance; preparation of a large-scale geological map; and development of the geological history of the area.

2. Airborne geophysical exploration, geochemical survey of silts and soils along streambeds and other prominent exposures, and field investigations for mapping widely spaced traverses to locate all superficial mineralization that can be found easily.

3. Geological mapping and geophysical, geochemical, and mineralogical prospecting of promising areas on a close grid.

4. Proving the deposit—essentially, delineating and sampling the ore. More detailed mapping and geophysical and geochemical studies may be necessary, but the fieldwork consists mostly of obtaining samples from the ore body exposed by pitting, trenching, drilling, and shaft sinking.

Geophysical prospecting is used to measure and interpret anomalous physical or physico-chemical phenomena within the crust of the earth. For example, magnetic material in the earth increases the strength of the normal magnetic field, and the resulting magnetic anomaly can be measured with sensitive magnetometers. Hence this method has been important in locating magnetic iron ores (magnetite) and has been helpful in tracing geological formations and certain types of nonferrous mineral deposits having a greater or smaller content of magnetic minerals than the enclosing or adjacent rocks. Various other geophysical methods, such as resistivity and radio frequency, depend on electrical phenomena. The petroleum industry has been eminently successful in developing and applying gravitational and seismic methods to location

of underlying oil-bearing structures. Gravitational methods are based on gravity differences between different types of rocks, and seismic methods depend upon differences in the speed of refracted and transmitted seismic waves through different rock layers.

Data obtained by exploration are required for determining ore reserves, planning mining methods and equipment, projecting the scale of operations, and other technological and economic factors essential to establishment of a mining enterprise.

These objectives postulate that the ore body be penetrated by boreholes and/or underground workings at appropriately spaced intervals and that the exploratory work be carried far enough to permit sound planning of subsequent operations. For example, porphyry-copper deposits usually are explored by a network of boreholes or underground workings on a 100-to-200-foot spacing over the entire area of the deposit, in advance of mining, to predetermine the factors that predicate a heavy expenditure for equipment and to avoid placing surface structures on ground that may later be mined or caved. However, exploration of a vein deposit, especially if it is to be mined by supported stopes, need only be carried far enough to demonstrate the existence of a workable ore body of sufficient extent and value to justify relatively moderate expenditures for development and equipment. Deferred interest charges on advance exploration and the cost of maintaining underground workings limit the amount of exploration that may be done in advance of mining; hence, in many underground mines the developed ore reserve rarely exceeds 10 years and often is less than 3. A continuing program of exploration and development is relied upon to add reserves to replace mined-out ore.

Boring

Boring is employed extensively, either as the principal exploratory method or to supplement exploration by underground and surface workings. Several types of drills are used for this purpose, the principal ones being churn drills, diamond drills, rotary drills, and hammer drills.

Churn Drills.—These are employed when solid cores are not required, stratigraphic thicknesses do not have to be measured accurately, and only vertical holes are desired. In churn drilling, a string of tools with a cutting bit at the lower end is suspended from a rope or cable and is alternately raised and dropped by a machine near the collar of the hole. The release of tension in the cable as the bit strikes the bottom of the hole imparts a slight rotation or churning action to the bit. Water is used

in the bottom of the hole during drilling, and the cuttings are brought to the surface with a bailer at regular measured intervals for examination and analysis.

Diamond Drills.—These are used widely in exploration because of their speed, adaptability for directional drilling, and portability, and because they provide accurate information about the rock types drilled. The diamond drill is a rotary drill designed to cut an annular groove about a central core of the rock penetrated. The bit is typically a hollow metal cylinder set with industrial diamonds on its inner, outer, and downward surfaces; it is joined to hollow steel connecting rods that are rotated by mechanical equipment at the collar of the hole. Water circulating through the hollow rods cools the bit and flushes the fine cuttings from the hole. The core formed as the drill advances is recovered in sections for inspection and testing.

Rotary Drills.—These rotate alloy steel or tungsten carbide drill (tricone) bits in the bottom of the hole to achieve a constant chipping, scaling, and grinding action. Downward pressure of the drill bit is supplied by the weight of the string of metal rods that connects the bit to the rotary table at the collar of the hole. Cuttings are either bailed or conveyed to the surface by a circulating fluid such as water, air, or a heavy medium mixture (fig. 3).

Hammer Drills.—Their best application is in exploratory drilling underground, where relatively short holes are to be drilled, and core recovery is not required. The maximum depth of drill holes for hammer drills is about 250 feet, but the most efficient range is less than 150 feet. These are common heavy rock drills, using sectionalized, hollow drill steel and standard bits.

Underground Exploration

Most prospects and small mines rely principally upon underground methods of exploration. A typical example in exploration of narrow veins is sinking a shaft in or alongside the vein and then driving underground-horizontal workings known as drifts or levels in the vein at convenient intervals. Exploration for parallel ore bodies or to find faulted segments of the main ore body is done by diamond drilling or by making horizontal openings (crosscuts) that penetrate the enclosing country rock at an angle to the prevailing strike of the vein or stratified country rock. In suitable topographic situations, a horizontal opening or adit may be driven into a hillside to reach or follow a vein that crops out at a higher level.
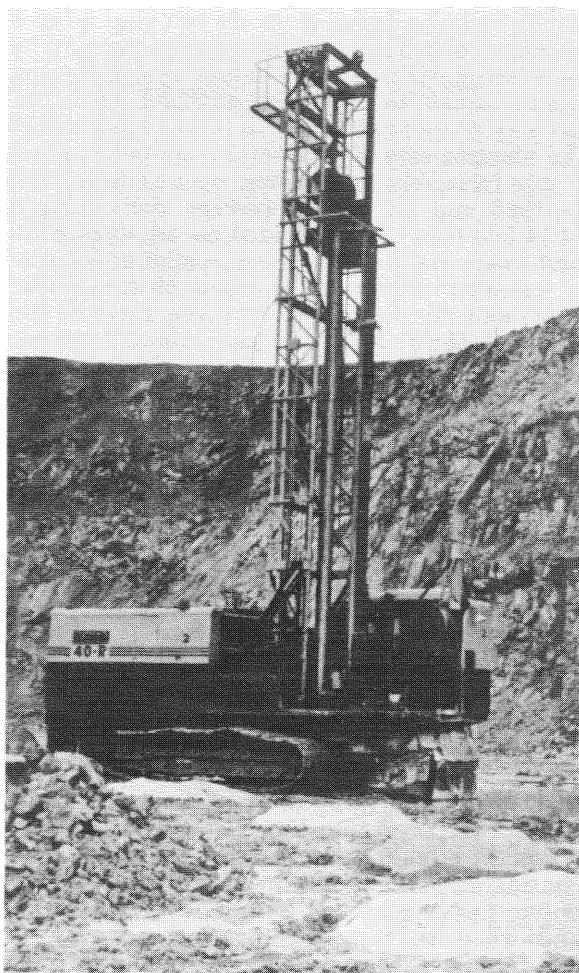
FIGURE 3.—Rotary Drill.

MINE DEVELOPMENT

The term "mine development" is employed to designate the operations involved in preparing a mine for ore extraction. In underground mines these operations include, principally, preparation of openings to and into the ore body—such as tunnels, crosscuts, drifts, raises, and shafts.

In most mines, both exploration and development continue after ore extraction has begun, often nearly to the cessation of mining. Although exploration and development work are similar, the emphasis is placed on ore finding in exploration; whereas in the development, operations are mainly preparation for removing ore from the mine.

The major operations in developing an open-pit mine are stripping, (removal of barren or low-grade overburden) and establishment of mine transportation systems.

Development of an underground mine involves the following: An efficient entry, whether a shaft or adit; auxiliary entries, required for safety and ventilation considerations, if workings extend beyond depths specified by various State laws; and such networks of drifts, crosscuts, and raises (or winzes) at various levels and intervals that may be required by the form of the deposit and the details of the mining method. Transportation systems for removing ore to the treatment plant and pumping infiltrated waters are common to both surface and underground operations.

The cycle of operations in metal-mine development underground consists of drilling, blasting, and removing broken rock. Drilling is generally done with compressed-air hammer drills or, more rarely, with diamond drills. Blasting is done with various types of explosives, detonated by electric caps or by fuses and caps.

Compressed-air or electric-powered shoveling and loading machines are used for removing broken rock and loading in headings and shafts in most important underground operations. Scrapers are used in inclines, and gravity is employed in steeper raises (usually more than 38°) for transfer of broken rock to mine haulageways. Transportation of broken ore and waste rock is commonly by car and track haulage, using storage-battery or compressed-air locomotives for short hauls beyond the range of hand tramming and either storage-battery or trolley locomotives for longer distances. Ore is also transported by shuttle cars, diesel trucks, and conveyor belts.

Some companies consider stope preparation a step in mine development. This operation includes all of the auxiliary workings directly related to preparing a given block of ore for mining and is confined to main entries and haulageways. This is an important distinction in mining massive deposits covering an extensive horizontal area, as in the porphyry-copper mines to be mined by caving methods, where advance preparation of stopes is a definite and extensive phase of operation that can be charged against a specific block of ore; but in narrow veins, thin beds, and relatively small irregular ore bodies there is usually no essential difference between development and preparation.

MINING

There are two principal types of copper ore deposits, those near the surface and those deep below the surface. Veins and other deposits of tabular or irregular form are usually deep and are mined by underground workings; methods employed are aimed at complete extraction of ore and proper support to prevent

loss of adjacent or overlying ore bodies and destruction of surface structures. The large disseminated deposits are mined by open-cut power-shovel methods, if they lie close enough to the surface, and the cost of removing the overburden is not excessive. If the overburden ratio is too high, these deposits can be mined by caving systems which are amenable to mass-production operation at minimum cost.

In recent years improvements in underground methods have consisted mainly of modifying existing techniques. Progress has been reflected in the growth of mechanization. Improved drilling equipment, loading equipment, and mechanical ventilating systems have been introduced.

Production from open-pit mines in the United States increased, in comparison to underground mines, from 1939 to 1952; the decrease from then until 1960 is due to opening of large underground mines in Arizona, Michigan, and Montana. Table 11 shows the relative importance of the two major copper-mining methods and points to the progressive increase in the proportions of copper ore and copper extracted by open-pit methods.

Mining Methods

The principal mining methods employed in modern copper mining are:

Open pit:
    Conventional.
    Glory-hole.
Caving:
    Block.
    Top slicing.
    Sublevel.
Supported stopes:
    Naturally.
    Artificially:
        Shrinkage.
        Cut-and-fill.
        Timbered.

Open pit.—Open-pit methods are applicable to mining ore deposits at or near the surface. If the top of the ore body is below the surface, the overburden and barren capping overlying the ore must be removed in advance of ore recovery. Removal of this material is known as stripping and is part of development.

The choice between open-pit and underground mining of a given ore deposit is based upon factors such as size, shape, and depth of ore body; relative costs of mining by open pit or by an underground method applicable to the deposit; dilution of ore with waste in relation to ultimate recovery of ore; topography and surface improvements; climate; availability of skilled labor (for underground mining); probable continuity of operation; and available capital. An important consideration in choosing the open-pit method is preliminary stripping, which must be done before ore can be produced at planned capacity. The average amount of rock stripped at several mines indicates that waste equal to about a fifth of the total estimated ore reserve has to be removed before the planned rate of production can be attained. Three relatively new mines, opened from 1955 to 1960 and having an average daily output of 14,000 tons of ore, yielded no ore during the first year; yielded ore at the rate of 25 percent of planned output by the end of the second year; and, at the rate of 75 percent by the end of the third year. These mines reached designed capacity during the fourth year, after about 30 percent of the total waste had been removed. It is generally planned to do a major part of preliminary stripping while treatment plants are being built so as not to delay full production.

Among the advantages of open-pit mining are its flexibility, the ability to obtain mass production, the ease with which the rate of production can be increased or decreased once the pit has been developed fully, small shutdown expense, the ability to mine selectively to meet requirements for certain grades of ore, virtually complete extraction of the ore inside the pit limits, the comparatively small labor force required, and elimination of hazards inherent in underground-mining operations.

However, certain disadvantages may outweigh the advantages and affect direct economic considerations. Large open-pit operations involve heavy capital outlay for equipment, and when the amount of overburden to be removed is extensive, a correspondingly high capital expenditure is required for stripping. This capital is nonproductive until ore mining is begun, and during the stripping period interest charges accumulate. The time elapsing before production begins may in itself be a serious disadvantage, especially if exploration is undertaken when ore prices are favorable and the demand for ore is strong. Disposal of the waste from stripping operations sometimes becomes a serious problem, especially when the terrain is flat or dump areas near the mine have a high real estate value. Climatic conditions may limit or necessitate complete closing of operations during certain months, and in areas where torrential rains are prevalent the pits may become flooded, hampering mining operations. Figures 4 and 5 show the Utah mine in Utah and the Chuquicamata mine in Northern Chile.

The mining cycle for an open-pit operation consists of drilling, blasting, loading, and transportation or ore and waste. Drilling is usually divided into primary and secondary. Primary drilling is the sinking of vertical or near-vertical blastholes behind the open face of

TABLE 11.—*Copper ores and copper produced in the United States, distributed by principal mining methods*

| Year | Production | | | Distribution by mining method | | | | | |
| | Ore | | | Open pit | | Block caving | | Other | |
	Thousand short tons	Copper, thousand short tons	Copper, percent	Ore, percent	Copper, percent	Ore, percent	Copper, percent	Ore, percent	Copper, percent
1939	55,239	714	1.29	59	42	20	15	21	43
1943	98,120	1,069	1.09	69	55	16	14	15	31
1946	62,232	595	.96	66	60	17	15	17	25
1947	87,864	832	.95	73	69	14	12	13	19
1948	84,729	818	.97	76	69	14	11	10	20
1949	76,033	731	.96	78	72	12	9	10	19
1950	94,586	886	.94	81	76	8	7	11	17
1951	95,494	901	.94	84	76	7	7	9	17
1952	99,947	901	.94	85	77	7	6	8	17
1953	101,065	906	.90	83	75	8	7	9	18
1954	93,654	816	.87	83	75	10	15	7	10
1955	112,550	979	.87	83	77	5	10	12	13
1956	131,776	1,082	.82	78	73	9	13	13	14
1957	129,716	1,061	.82	77	72	11	14	9	14
1958	114,824	959	.84	76	71	13	15	11	14
1959	103,716	807	.78	79	74	9	12	12	14
1960	134,994	1,058	.78	80	75	10	10	10	15
1961	142,722	1,142	.80	80	74	9	11	11	15
1962	150,217	1,203	.80	81	75	9	11	10	14
1963	146,450	1,187	.81	81	74	9	12	10	14

an unbroken bank. Secondary drilling is required for breaking boulders too large for shovels to handle or blasting unbroken points of rock that project above the digging grade in the shovel pit.

Benches of open-pit copper mines normally are from 25 to 60 feet high. The height is selected to give maximum digging efficiency but sometimes is limited for ore-waste separation. The height may be reduced where the rock is hard and difficult to break and increased where it is softer.

Blastholes range in diameter from 6 to 12 inches and are drilled from 2 to 15 feet below grade. The holes usually are drilled with a rotary drill, using multicone-type cutting bits. Smaller diameter holes spaced more closely are used in harder ground. Distances between holes range from 12 feet for 6-inch holes in very hard ground to as much as 50 feet for 12-inch holes in easily broken ground. In very hard ground, percussion-type drills with tungsten carbide bits are used. Performance of heavy percussion-type drills ranges from about 60 to 150 feet per shift; that of rotary-type drills is from about 150 to 500 feet per shift. Bit performance varies greatly, depending on the type of rock; for 12-inch bits it ranges from 600 to 3,000 feet per bit. Figures 6 and 7 show plan and section of primary blasthole arrangements.

For secondary drilling, portable compressors and wagon drills have been replaced at most mines by self-contained units with compressor and drill installed as integral parts. The drills are mounted on hydraulically controlled arms that can be adjusted to drill horizontal or

vertical holes. When extremely dense boulders are encountered, a "drop ball" is used. Where adaptable, this is an efficient tool for breaking oversize boulders. In mines where only a few boulders occur, mudcap or adobe shots, which require no drilling, are used.

Blasting is of primary importance. Good breakage depends on the proper selection of explosives and the proper arrangement of drill holes. Gelatinized nitroglycerine explosives are used in wet holes and the ammonium nitrate type in dry holes. Usually the charges are placed at the bottom of the hole and exploded by detonating fuse, but often small additional charges, commonly called deck charges, are placed high in the hole. Delays are used where toe holes or multiple rows of holes are fired in the same blast.

Commercial ammonium nitrate, sensitized by addition of diesel fuel or some other suitable petroleum product, has been adopted as a low-cost efficient blasting agent and is used widely. The ready supply of the chemicals and the ease with which they can be formed into an efficient blasting agent in the field enables the blaster to prepare the explosive on the job.

Ore and waste are loaded by full-revolving electric and diesel powered shovels. At most mines shovels with 4-to-6-cubic-yard capacities are used, but larger mines have shown an increasing tendency to use 7-to-13-cubic-yard shovels. Four-to-six-cubic-yard shovels generally load 3,000 to 6,000 tons per operating shift, and the larger shovels load as much as 15,000 tons per operating shift. The larger shovels are particularly useful in waste areas where large boulders can be loaded and dumped

FIGURE 4.—Utah Copper Open-Pit Copper Mine, Utah.

(Courtesy, Kennecott Copper Corp.)

FIGURE 5.—Chuquicamata Mine, Atacama Desert, Northern Chile.
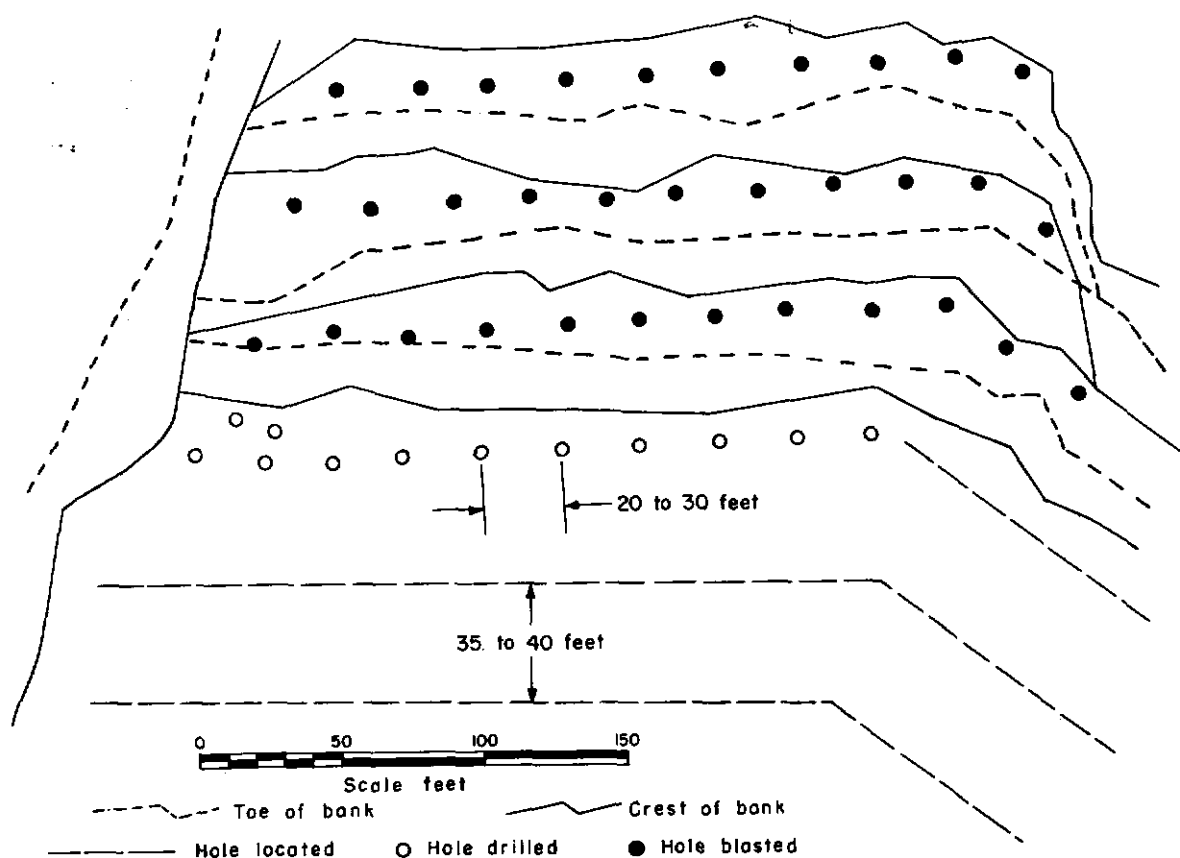(Courtesy, The Anaconda Company)

FIGURE 6.—Plan of Located, Drilled, and Blasted Holes.

without secondary blasting. In the ore areas, boulders must be broken to a size that the crushing facilities can handle.

Before World War II, the cost of breaking ground in open-pit mines far exceeded the combined cost of loading and hauling ore. Today the cost of loading and hauling has increased while the relative cost of breaking ground has decreased. It is now about half the total cost of mining. Loading and transporting ore are closely related. The sizes and number of shovels used are determined by daily production, type of material handled, type of haulage, and working conditions such as height of bank, angle of swing, working radius, and clearance. On the other hand, selection of haulage equipment is influenced by the type of loading equipment used.

In general, large mines, or those with long hauls, have rail haulage; medium-sized mines have truck haulage or truck haulage in combination with some other type of haulage. (Rail haulage has been replaced by truck haulage in a number of the larger mines.) Some types of auxiliary haulage are inclined skips (Figure 8), vertical skips, inclined-belt conveyors, standard-gage railroad cars, large capacity tractor-type trucks, and tractor-drawn scrapers.

Bulldozers are used to push materials for distances to 200 feet; scrapers are used from 200 to 1,500 feet; trucks are used for distances ranging from 600 feet to 1 mile; and rail haulage is used for distances of more than 1 mile. Performance with bulldozers ranges from 500 to 1,500 tons per shift, that of scrapers from about 400 to 1,500 tons per shift, that of large trucks from 500 to 2,000 tons per shift, and that of rail haulage from about 1,000 to 3,000 tons per locomotive shift for distances of about 3 miles.

Glory Hole.—In some instances when open-pit operations extend to depths that cause the stripping ratio to become uneconomic, the glory-hole mining method has been used. This method has also been used for mining relatively small ore bodies or the upper parts of bodies remaining at or near the surface. In glory-hole mining, the ore is broken around one or more raises extending upward from an underground haulageway driven below the ore or beneath

NOTE:

B = distance from hardtoe to crestline
T = distance from hardtoe to drill hole
C = distance from crest to drill hole

Angle between crest-
toe and vertical is
measured with clinometer.

Digging grade

Toe

Powder

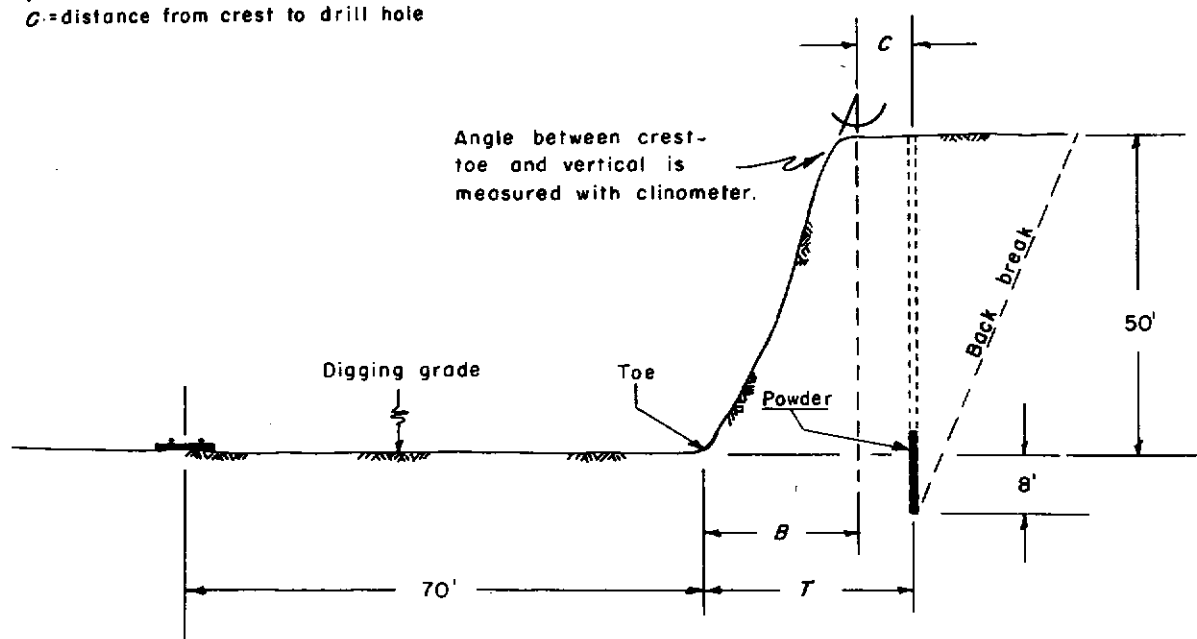Back break

50'

8'

B

70'

T

FIGURE 7.—Vertical Section of Primary Drill Hole.

the ultimate bottom of the glory hole or pit. It is thus a combination of surface and underground mining with the ore breaking done on surface but removed through underground workings. Under favorable conditions gloryhole mining is as economical as open-pit mining for the same scale of operations.

Caving Methods

Caving methods are of three distinct types, block, top slicing, and sublevel:

Block Caving.—This method is amenable to mass production and is efficient and low cost—permitting scientific planning, centralized management, labor specialization, and modern mechanization. The successful use of this method requires detailed engineering study of the nature of an ore body by systematic and detailed exploration and sampling. Adequate development, preparatory work, transportation, ore drawing, and allied mining operations are equally important.

Block caving consists of dividing suitable ore bodies into blocks of predetermined size and undercutting each to induce rock stresses to cave and crush the ore to sizes that can be readily handled. Block caving is applicable to homogeneous and rather weak ore bodies of regular outline with enough horizontal area to cave freely; strong walls and capping or over-

burden that caves freely are required also. The method utilizes instability of the ore for caving, shear and compressive stresses for crushing, and gravity for moving the ore to drawpoints. The method is nonselective, and lean sections of ore and waste will be broken up and drawn with the ore (fig. 9).

Main haulageways are driven on a level sufficiently below the bottom of the ore so that branching transfer raises may be installed to connect with a grizzly or slusher sublevel and so that closely spaced branching draw raises can be driven to the bottom of the planned undercut block (figs. 10-11). Detailed information about the two large block-caving mines in the United States is presented in Bureau of Mines publications.[3] Figure 12 shows the block-caving system at the Braden Copper Company El Teniente mine in Chile.

Top Slicing.—This is an important modification of caving, whereby the ore is extracted by excavating a series of horizontal or inclined timbered slices alongside each other, beginning at the top of the ore body and working progressively downward. Each slice, when mined out, is caved by blasting out the supporting timbers and allowing them to crush, bringing the capping or overburden down upon the

[3] V. B. Dale. Mining, Milling, and Smelting Methods, San Manuel Copper Corp., Pinal County, Ariz. BuMines Inf. Circ. 8104, 1962, 145 pp.
C.C. Popoff. Block Caving at Kelly Mine, The Anaconda Co., Butte, Mont. BuMines Inf. Circ. 7758, 1956, 102 pp.
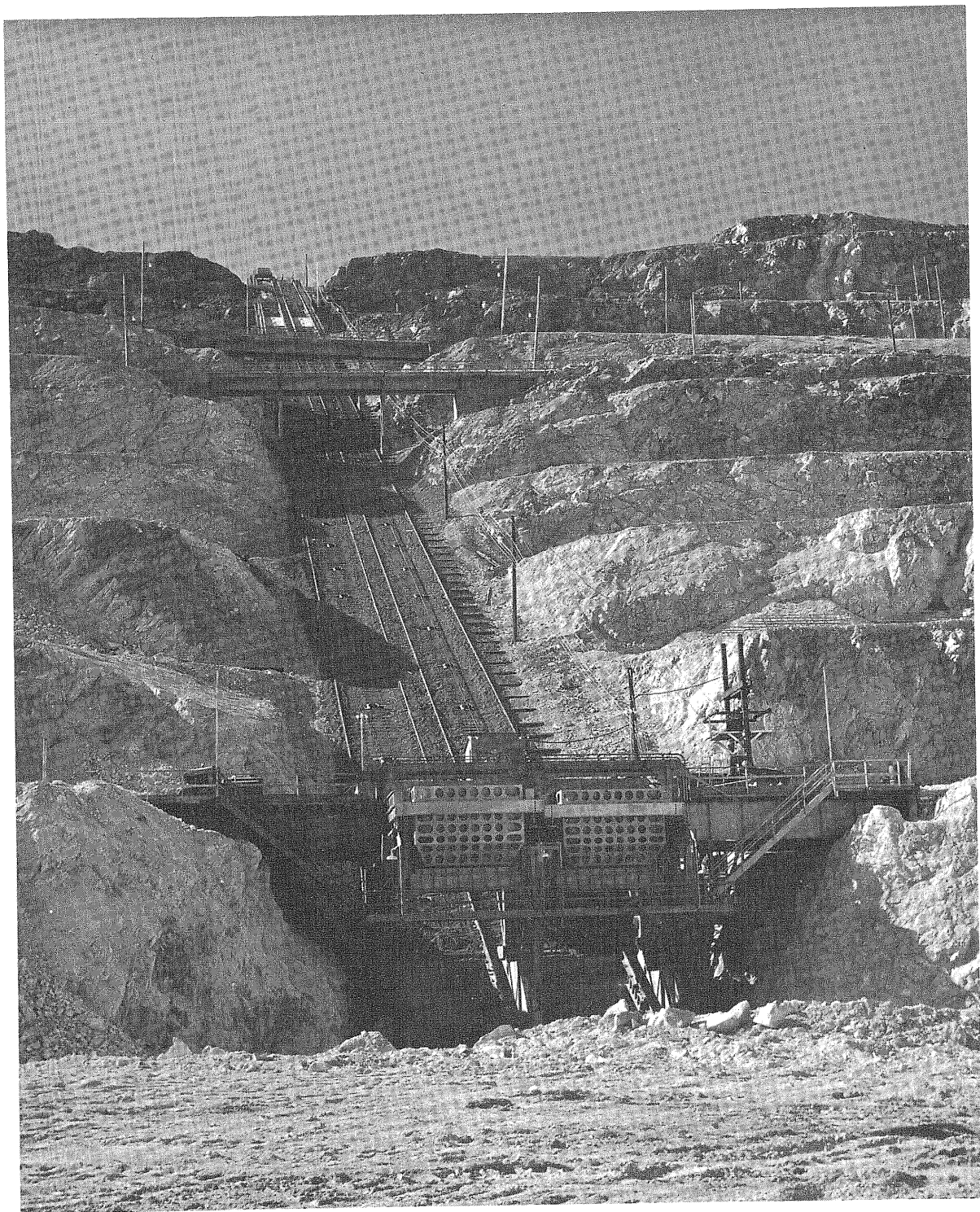
FIGURE 8.—Inclined Skip Hoist, Chino Mine, N. Mex.

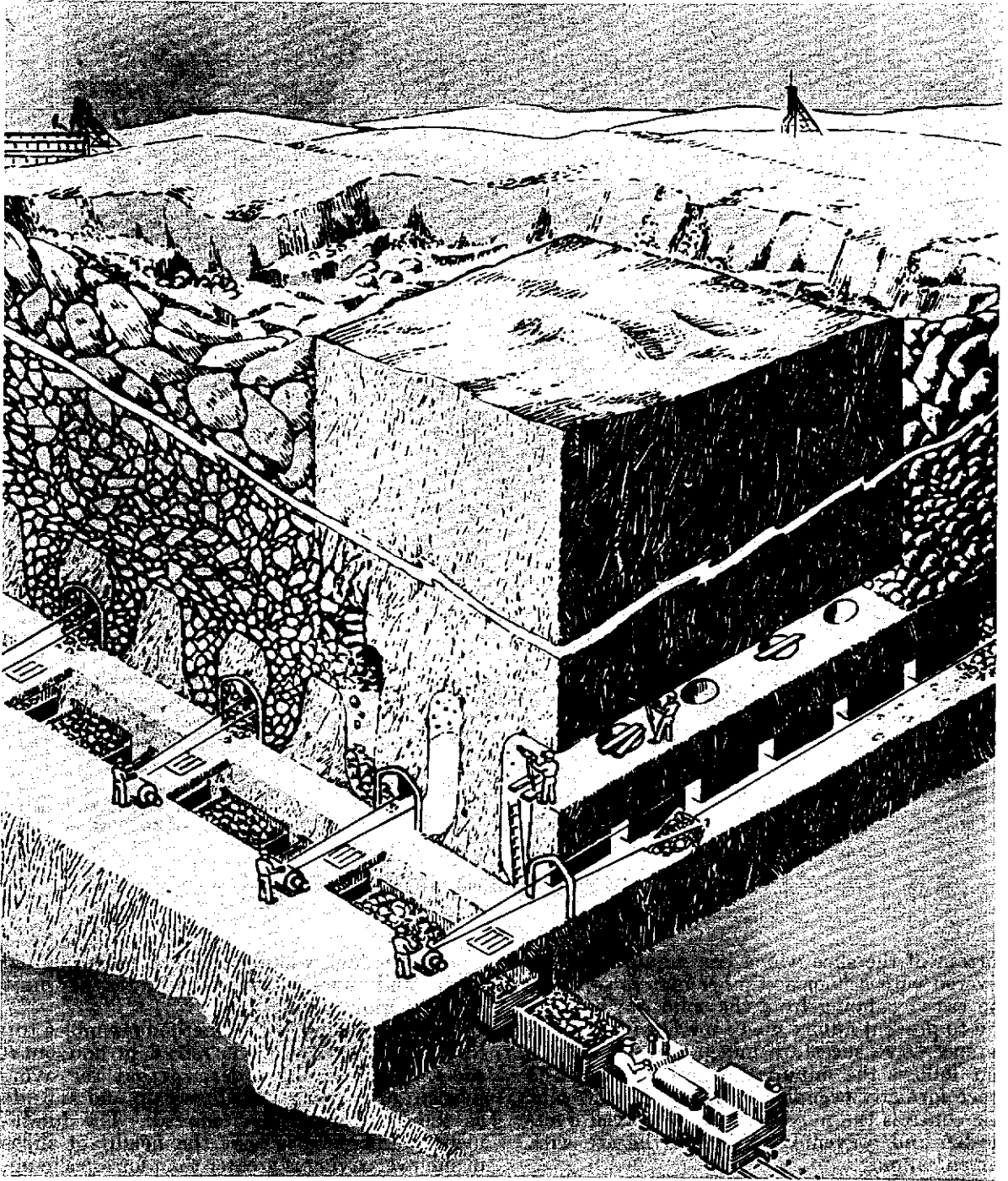(Courtesy, Kennecott Copper Corp.)

FIGURE 9.—Block-Caving Method.
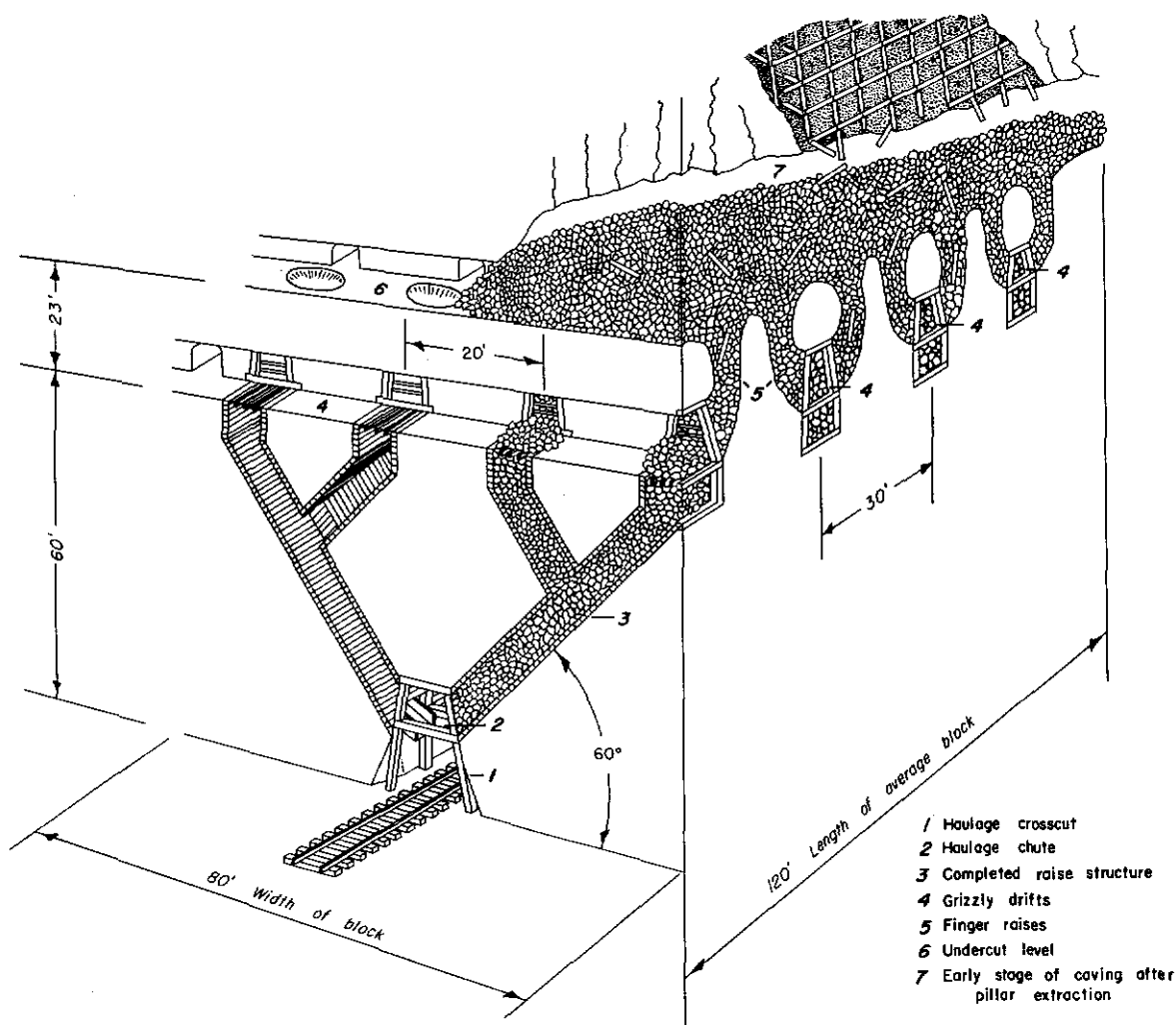
(Courtesy, The Anaconda Company)

FIGURE 10.—Block Development, All-Gravity System.

bottom of the slice which has previously been covered with a floor or timber mat to separate the caved capping from the solid ore beneath and to prevent admixture of waste with the ore. As successive slices are mined and caved, this mat follows the mining downward, filling the space formerly occupied by the ore. The mat also controls the movement of the caved overburden and prevents dilution of the ore with barren capping.

Block-caving methods result in some loss of ore and a moderate dilution of ore with waste, but top slicing, although more costly, is capable of virtually complete ore recovery without dilution, hence is favored for ore bodies of somewhat higher copper content than those to which the block-caving methods are applicable. Top slicing is also more applicable to smaller and more irregular ore bodies than is block caving.

It requires highly skilled miners, more manual labor, and more labor expenditures per unit of production.

Sublevel Caving.—This method resembles top slicing in that the ore is mined in horizontal slices in descending order, so that the overburden, or capping, will break up and subside as the ore beneath is removed. The fundamental difference is that the height of slices in sublevel caving is greater than for top slicing, allowing larger output, lower breaking costs, and use of less timber. This method, adapted principally for certain iron mines in the Lake Superior region, is rarely used in copper mining.

Supported Stopes

The term "stoping" is employed in its broader sense to mean excavating ore by a series of
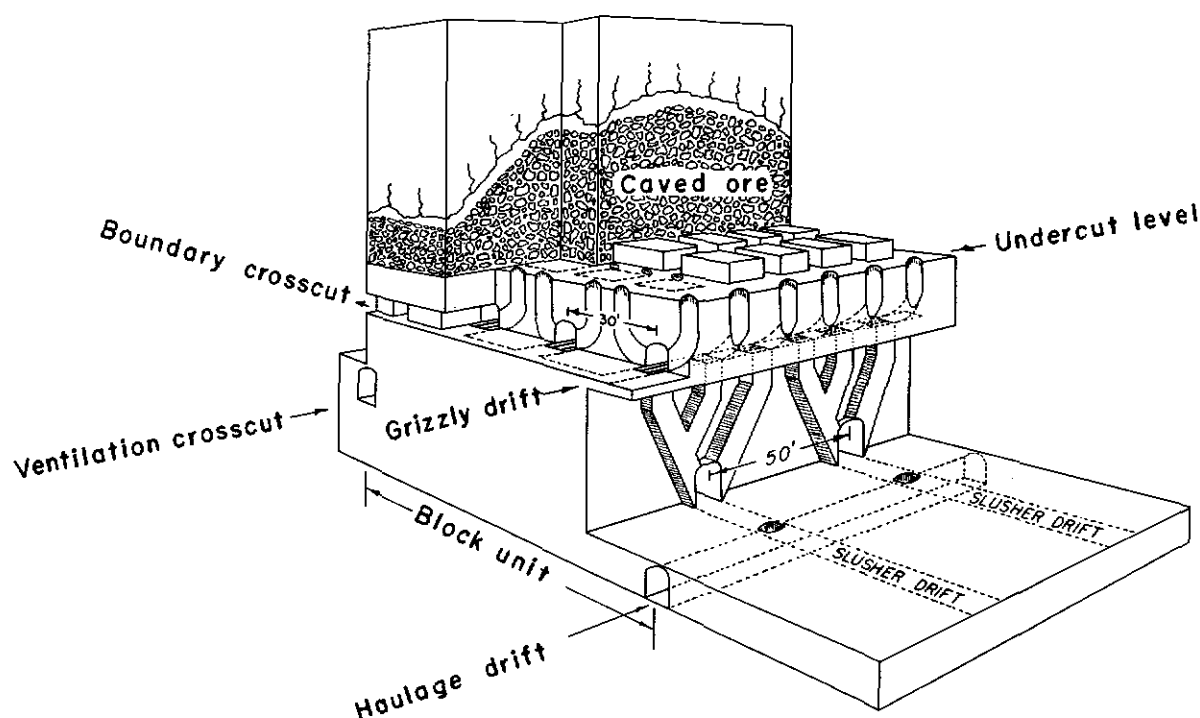
FIGURE 11.—Slusher-Gravity Block Three-Bench Raise Design.

horizontal, vertical, or inclined workings in veins or large irregular bodies of ore or by rooms in flat deposits. It covers breaking ore and removing it from underground workings (except those driven for exploration and development) and timbering, rock bolting, and filling stopes for support.

Basically, the stoping method or methods that can be applied to a given ore body depend on requirements for supporting the stope; the maximum area or span of back (roof) and walls that will be self-supporting during removal of the ore; and the nature, size, and interval between supports required to maintain the back and walls of the overlying and surrounding country rocks and overburden to prevent their movement and subsidence. Variations of the principal methods of stoping may be based on the direction or angle of workings, sequence of operations, or methods of handling broken ore.

Naturally Supported Stopes.—Naturally supported stopes are those in which no regular artificial method of support is employed, although occasional props, cribs, stulls, or rock bolts may be used to hold local patches of insecure ground. Walls and roof are self-supporting. The simplest form is the open stope, in which the entire ore body is removed from wall to wall without leaving any pillars. It is applicable to relatively small ore bodies, as there is a limit to the length of unsupported span that will stand without support, even in the strongest rocks.

In open stopes with pillar support, the length of unsupported span is controlled by leaving pillars of ore whose position and size are determined by localized ground conditions. Frequently it is possible to leave low-grade ore within the ore body as pillars, making possible more complete recovery of the higher grade ore.

Artificially Supported Stopes.—Artificially supported stopes are those in which installation of systematic temporary or permanent support of ground around mined-out areas is a part of the mining cycle.

Shrinkage Stopes.—In shrinkage stopes the ore is mined in successive flat or inclined slices, working upward from a level or the bottom of the block of ore. After each slice or cut, only enough broken ore is drawn off from below to provide working space between the top of the pile of broken ore and the back of the stope. Usually about 35 to 40 percent of the ore will be drawn during active mining in the stope. The remaining ore serves as a floor upon which to work in drilling the back for succeeding cuts and also provides temporary support to the stope walls. For this reason, shrinkage stopes are considered a form of artificially supported stope.

When active mining has been completed to the level above or to the floor pillar, the rest of

A Undercut process
B Block: Not drawn
C Block: 20% drawn
D Block: 100% drawn
E Virgin ground
F Vertical raises
G Grizzly level
H Gathering system
I Gates
J Ore pass
K Control gates
L Haulage level

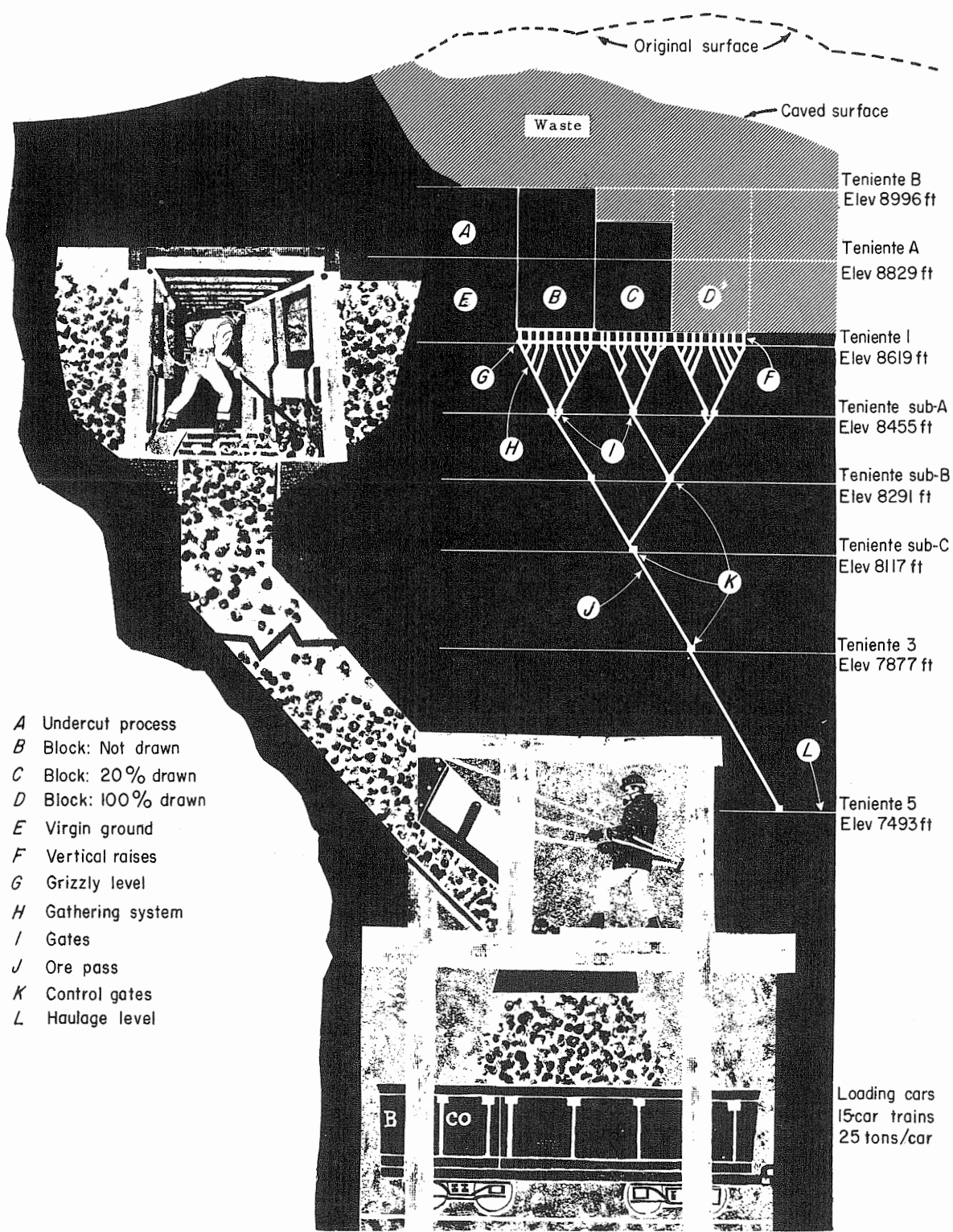FIGURE 12.—Diagramatic Sketch of Block-Caving System of the El Teniente Mine, Sewell, Chile.

(Courtesy, Kennecott Copper Corp.)

the broken ore is drawn from below, leaving the stope empty. The stope may be filled with waste later to prevent general movement and subsidence or to permit mining of pillars left between stopes during the first mining.

Shrinkage stoping is applicable to bodies of ore enclosed between firm walls that will not slab or slough off to any extent when left standing for a considerable time. The method is applied most frequently to relatively thin, tabular deposits dipping at angles greater than 50° in which few waste inclusions occur and which have fairly regular walls.

One of the disadvantages of shrinkage stoping is the delayed recovery of broken ore that cannot be drawn until the entire block has been mined. Another is that there is more oxidation of sulfide ores than in systems having immediate withdrawal of ore. Oxidation may adversely affect metallurgical recovery or, in extreme cases, result in a mine fire.

Cut-and-Fill Stopes.—In cut-and-fill stoping the ore is excavated by successive flat or in-clined cuts or slices, working upward from the level as in shrinkage stoping; but, after each cut, all the broken ore is removed and waste rock, sand, mill tailings, or other filling material is run within a few feet of the back, providing permanent support to the walls (fig. 13) and a working floor for the next cut. The term "cut-and-fill" implies a definite characteristic sequence of operations: (1) Breaking a slice of ore from the stope back, (2) removing the broken ore, and (3) introducing filling. The cycle is repetitive.

Cut-and-fill stoping is applicable to mining firm ore enclosed between two walls, one or both may be weak. In general, it is suitable for mining deposits too irregular for shrinkage stoping and deposits in which shrinkage could be employed if it were not for weak walls. Improved ore-handling and waste-spreading methods have extended the economic applicability of cut-and-fill stoping by comparison to shrinkage stoping, and the method has the
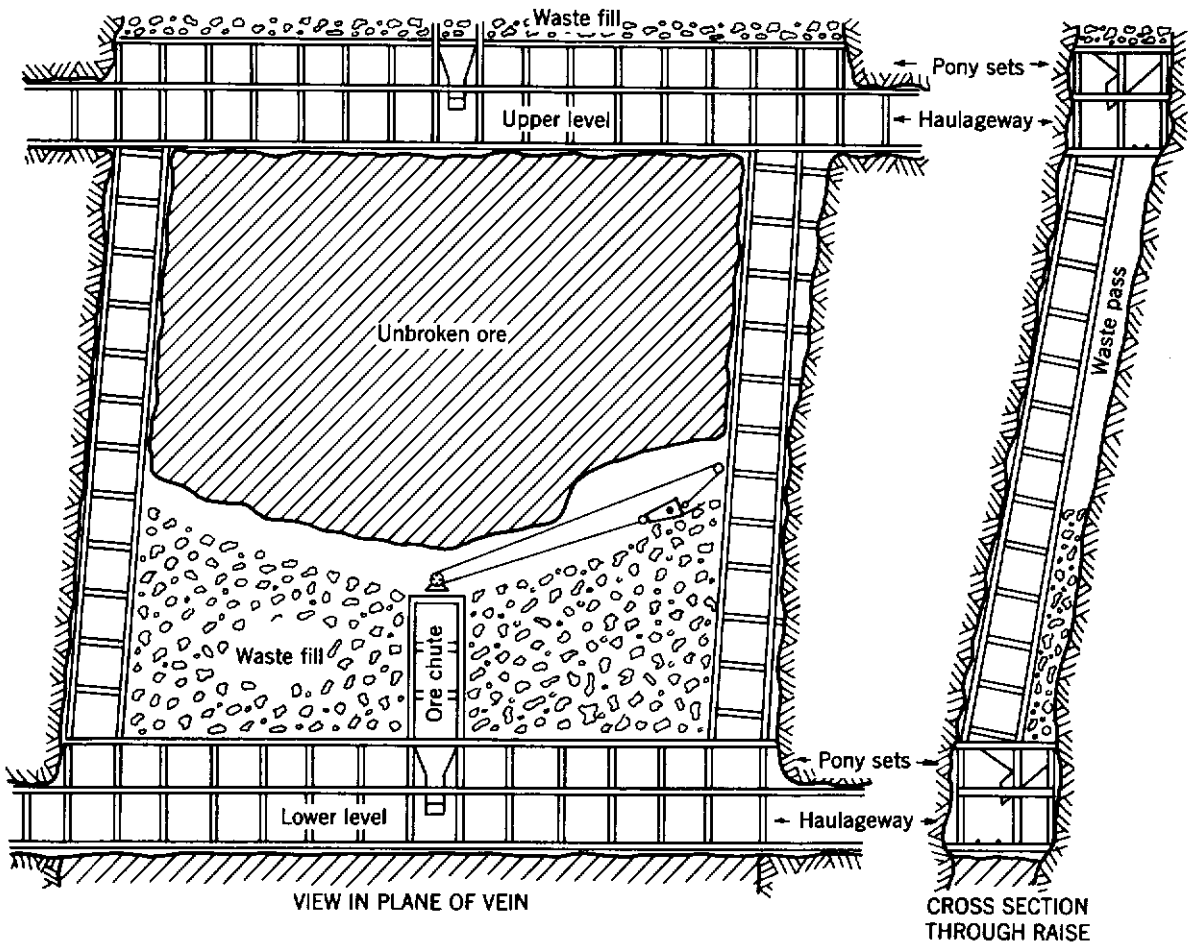

FIGURE 13.—Cut-and-Fill Stope.

advantage of greater selectivity, better working conditions, and greater safety.

Rock bolting provides a type of support that allows cut-and-fill stoping of heavy ore zones with weak and blocky hanging walls that otherwise would require square-set timbering. Rock bolting extends the time before ground failure, requires less time than timbering, permits exposure of the hanging wall for greater lengths and heights, and eliminates handling timber on the surface, in the shaft, and underground. All these factors contribute to increased production from fewer working places, thus reducing mining costs.

Timbered stopes.—Timbered stopes are those in which timber is used systematically for support. The most elaborate system of timbering is the square-set method (fig. 14), in which the walls and back of the excavation are supported by regular framed timbers forming a skeleton enclosing a series of contiguous, hollow, rectangular prisms in the space previously occupied by the ore and providing continuous lines of support in three directions at right angles to each other.

The ore is removed in small, rectangular blocks, usually just large enough to provide room for standing a set of timber. Ordinarily the stopes are mined in floors or horizontal panels one above the other, and the sets of each floor are framed into the sets of the preceding floor. Timbered stoping is usually accompanied by filling, and often in heavy ground the sets are filled with waste after they are installed, leaving only a small volume of the stope unfilled at any time. It has become accepted quite generally that, unless the ground is heavy enough to require filling for permanent support, the expense of timbering is not warranted, and other methods should be employed.

Timbered stoping is adaptable to mining regular or irregular ore bodies, where the ore and/or walls are too weak to stand—even over short spans—for more than a brief time, and where caving and subsidence of overlying rocks must be prevented. It is the most selective underground mining method, hence is particularly suitable for mining rich, irregular ore bodies.

CONCENTRATING

The metallurgical extraction processes used in producing copper metal are based on physical and chemical characteristics of the minerals in ore bodies, such as grain size, copper content, nature and content of byproducts to be recovered, and the type of impurities to be eliminated.

Low-grade, oxidized-copper ores usually are treated by leaching and precipitation (cementation) or electrolytic deposition of copper;

some sulfide and oxide ores are sufficiently high-grade for direct smelting, but the bulk of all native or sulfide copper ores are first subjected to a physical separation of minerals by an upgrading process known as concentration.

Sulfide concentrates and high-grade ores are treated in a smelter in a series of pyrometallurgical steps to produce an impure (blister) copper, which is subsequently refined by pyrometallurgical or electrolytic methods. Native copper concentrates are smelted, and the copper is fire refined. In both smelting processes the gangue minerals and other valueless components are removed as slag. Byproducts are recovered at one or more of the various steps of concentration, smelting, or refining.

Concentration is the process of effecting physical separations of two or more minerals. Mineral dressing, ore dressing, beneficiation, and particularly milling are also part of the concentration process, having slightly different connotations. The plants in which mineral-dressing operations are conducted are known as mills or concentrators.

After liberation of the valuable minerals from the gangue, the separation of two or more minerals from each other is possible if they present critical differences in certain physical or chemical properties. Most important in concentration and separation of copper minerals from gangue are the chemical form, size, density, and surface characteristics of the several minerals in the ore.

The products of concentration are concentrates, containing the bulk of the valuable mineral, and tailings, containing gangue minerals. An intermediate product known as middling may be re-treated for further recovery of valuable minerals before final rejection as tailing.

Concentration is less costly than smelting Furthermore, shipping costs to a distant smelter are reduced by producing a concentrate near the source of ore to avoid freight charges on waste associated with the ore.

Crushing and Grinding.—The first step in concentration is to crush and grind the ore to such a degree as to liberate the valuable minerals from the gangue minerals, thus the grain size of the ore minerals is an important controlling factor. Hardness, tenacity, brittleness, and structure influence the cost of grinding and the relative degree of sliming (production of extreme fines) of each mineral in an ore. Crushing and grinding are done only to the size necessary to liberate copper minerals from the gangue since sliming results in losses.

Most crushing plants for copper ores obtain size reduction in three steps. Primary crushing of ore as it comes from the open-pit or underground mining operation produces 6- to 9-inch

Note: This side of the stope is ready for filling. Waste is thrown in from the sorting chutes as long as possible.

Rill stope raise

SECTION A-A

Rill stope raise

Offset

Hanging-wall angle brace

Framed round timber

4" lagging

4" X 10" lagging

Offset

Drift

Gob

Temporary side

Sorting chutes

Waste

Ore

Drill holes

A

Center chutes

—133'4"—

A

Lower drift

EARLY STAGE OF STOPING

Upper drift

200'

Chutes

Waste bulkhead

4" X 10" lagging

Manway

Waste filling

Framed round timber

10" X 10" timber

Starting 9th floor

Drill holes

2" X 7'9" lagging

Center chutes

—133'4"—

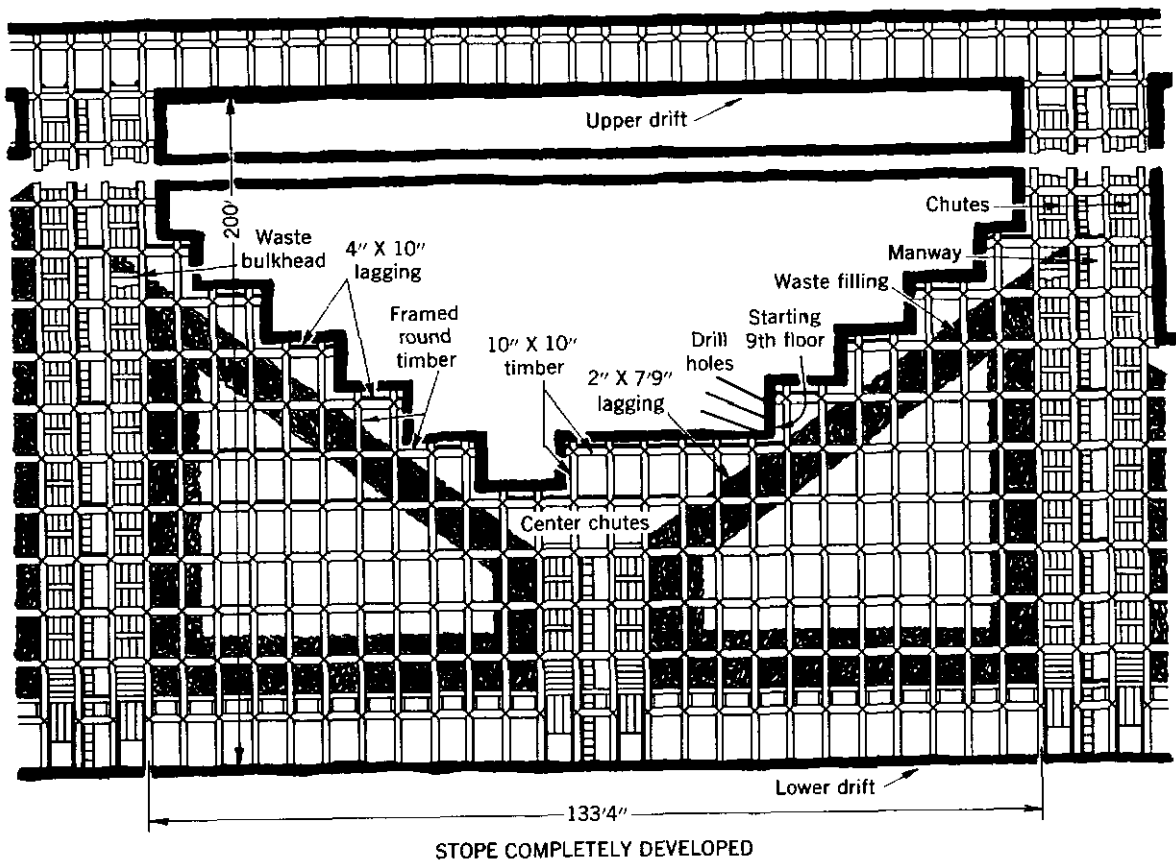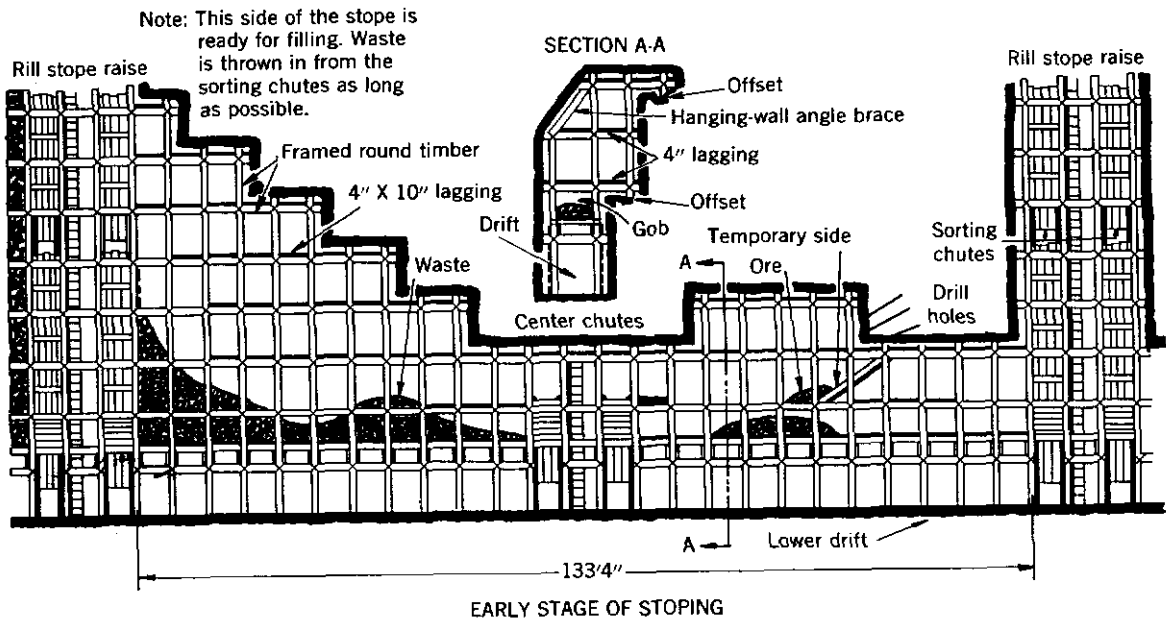Lower drift

STOPE COMPLETELY DEVELOPED

Figure 14.—Square-Set Stope.

pieces. This initial reduction is accomplished usually by one large gyratory crusher capable, at some operations, of crushing up to 2,500 tons per hour and the larger units can receive 60-inch boulders weighing 10 tons or more. The 6- to 9-inch product passes to several secondary gyratory or cone crushers where it is reduced to a maximum size of 1½ to 2 inches. Following screening for removal of material already reduced to wet-milling size, the oversize is passed through a tertiary stage of crushing in higher speed cone-type crushers. The product is screened, and the oversize is recirculated until it is reduced to about ⅝ inch.

Fine grinding is accomplished in rod and/or ball mills. Although it is possible to grind 2-inch-diameter feed to about 48-mesh (0.012 inch in diameter) in a single stage, grinding is usually cheaper for large tonnage if done in two or three stages when a high degree of comminution is required. Rodmills are used for coarse grinding, ballmills, for fine grinding. The newer concentrators built in the United States and South America use one open circuit rodmill, followed by two ballmills in closed circuit with rake classifiers, hydrocyclones or hydroscillators, and sand screws, to obtain a properly sized flotation feed.

Screens and classifiers play an important part in mineral dressing. Screens and grizzlies are used to bypass material already fine enough or to return oversize for recrushing. They are most efficient in size ranges greater than 10-mesh (0.065 inch in diameter); below this size various types of classifiers are preferred in most metallic mineral concentrators.

Classifiers are capable of separating finer sizes of materials into groups according to their settling rate in a fluid, usually water, or air. Of two particles of equal density but of unequal size, the larger will settle faster; of two particles of equal size but differing density, the higher density particle will settle faster. Chief applications of classification in modern ore-milling practice are in closed circuit with grinding mills to return oversize for regrinding. Classification also is used to prepare table feed in gravity concentration, but tabling plays a comparatively minor part in modern milling practices.

The more important accessories to ore concentrators are conveyors, bins, pumps, feeders, and filters.

Gravity Concentration.—Differences in the density of minerals form the basis of gravity concentration processes. The principal types of gravity concentration are: (1) Jigging, which depends on differences in the settling rate of minerals as they are carried horizontally by waterflow over a screen bed and subjected to a vertical pulsating action; (2) heavy-media

separation methods, which depend on the use of a fluid sufficiently high in specific gravity or a suspension of a heavy mineral such as magnetite, galena, or ferrosilicon in water, to cause the gangue minerals to float and to allow the ore minerals to settle, or vice versa; (3) tabling, which depends on the differences in speed of travel of minerals of different sizes and density, as they are caused to flow in a stream of water or air transversely across an inclined, riffled surface that is subjected to longitudinal reciprocating motion.

Jigging, heavy-media separation, and tabling are used very little in copper-ore concentration, because most ores require finer grinding for mineral liberation than the minimum economic size limit of gravity methods. Most copper sulfide minerals are recovered by flotation. A pilot table, however, is often used as an indicator or guide to help control losses in a flotation circuit.

Flotation

Flotation is a process of wet concentration in which air bubbles are used to float one kind of particle from a mixture of two or more kinds of finely divided materials suspended in water. Certain minerals may be preferentially wetted by organic reagents in the presence of water so that they will adhere to air bubbles and float to the surface of the pulp. Other minerals, usually the gangue, are unaffected and remain suspended in the water.

The reagents that selectively coat the valuable mineral particles are known as collectors. In sulfide flotation these are usually xanthates or other chain hydrocarbons. Supplementing the action of the collectors is a group of reagents known as frothers whose function (in conjunction with agitation and aeration) is to create a myriad of small bubbles in the pulp. The bubbles attach themselves to the properly conditioned valuable mineral and rise to the surface to form a froth that may overflow or be removed mechanically. Frothers in sulfide flotation are usually alcohols, pine oil, or ring-carbon compounds, such as cresylic acid.

When two or more minerals of the easier floating type are floated together to form one concentrate, the process is known as bulk flotation. Differential flotation is the term used to describe an operation in which one or more minerals are depressed during flotation of one or more other minerals, or where several different minerals are floated successively into separate concentrates. The usual differential separations in copper milling are copper sulfides from pyrite, galena from sphalerite, and sphalerite from pyrite. Any one of the sulfides may be rendered more floatable than the others by the current surface modifications, and such

modifications are obtained by using a class of reagents known as conditioning agents. Under this class are grouped the depressing, activating, and dispersing agents; the pH regulators; and the cleaning agents.

In general, sulfide-mineral particles coarser than 35-mesh (0.016 inch in diameter) cannot be effectively recovered by flotation; consequently, an ore that is to be floated must first be ground fine enough so that all, or substantially all, the desired mineral is smaller than this limiting size. This is aside from considerations of liberation, which may require even finer grinding.

To obtain a good recovery and a high-grade concentrate, several stages of flotation concentration are required. This is conveniently accomplished by employing successive agitating chambers or cells, so that the tailings from the first cell may pass progressively from one cell to the next as the original ore becomes impoverished in metal content. The concentrate from the first group of cells treating low-grade ores is seldom rich enough for a final product and is known as a rougher concentrate, which is re-treated or cleaned in one or more stages to produce a cleaner concentrate. In some instances, the rougher tailing may require regrinding for further liberation of minerals, and in other cases, it may be floated directly to produce a low-grade scavenger concentrate, which is often returned to the first rougher cell. The cleaner concentrate is dewatered in a thickener and a vacuum filter, and the dried concentrate is sent to the smelter. The plant tailing is conveyed by gravity flow or pumped to a settling pond. Where water recovery is required, the tailings are thickened, and the reclaimed water is returned to a large storage tank.

Figure 15 is the flowsheet of the Mission concentrator of the American Smelting and Refining Co., near Tucson, Ariz. This modern installation is one of the more recent worldwide. Figures 16 and 17 show the grinding and flotation sections of this mill.

With mixed ores, in which both sulfide and oxidized minerals occur, treatment depends on relative proportions of the two kinds of minerals. If sulfide minerals predominate, flotation is used, employing reagents that favor flotation of oxidized minerals. With such treatment, it is usually possible to recover more than 90 percent of the sulfide copper and 50 to 70 percent of the copper in oxides. If oxidized minerals predominate, the copper is usually recovered by leaching with sulfuric acid; ferric sulfate is an effective leaching agent when the sulfide portion of the mixed ore has significant value.

Dual Process.—When low-grade ores contain almost equal amounts of sulfide and oxidized minerals, combinations of leaching and flotation may be employed. In the Inspiration Consolidated Copper Co. Dual Process, the ore is first leached with sulfuric acid in the leaching plant to recover the oxide copper content, and the washed and drained residue is treated in the concentrator where, after fine grinding, the copper sulfide values are recovered by flotation.

LPF Process.—Along with conversion from underground to open-pit mining at the Ray Mines Division of Kennecott Copper Corp., the mill at Hayden, Ariz., was rebuilt to incorporate the leach-precipitation-flotation (LPF) process for recovering oxidized copper in the open-pit ore. This process converts copper oxide to metallic copper, which responds to recovery by flotation. The process starts by making a sand-slime separation of the rodmill discharge. The sands are leached in drums with dilute sulfuric acid and washed counter-currently. The washed sands are ground in ballmills for conventional, alkaline, copper sulfide flotation. The slimes are leached in a tank that received pregnant solution from sand leaching. Sponge iron is added to the slime pulp to precipitate copper, which is then recovered by flotation in an acid circuit. The LPF concentrate and the sulfide concentrate are joined and pumped to the smelter.

Segregation Process

There are oxidized and mixed oxide-sulfide ores that are not amenable to leaching with sulfuric acid or conventional flotation concentration. The success of acid-leaching oxidized ores depends largely on the amount of acid consuming constituents in the gangue. If the gangue contains significant quantities of calcite or other acid consuming minerals, acid leaching is infeasible. Where chrysocolla is the predominant oxide copper mineral, flotation is impractical because there is no commercial flotation process for recovering this mineral. However, research at the Federal Bureau of Mines Tucson Metallurgy Research laboratory demonstrated that the segregation process has merit for treating oxidized and mixed oxide-sulfide copper ores, regardless of the gangue constituents present.

The segregation process involves heating the oxidized or mixed oxide-sulfide copper ore with a halide salt, and a carbonaceous material, such as coke or coal, at approximately 1,400° to 1,500° F. Basically, the process proceeds in four stages: (1) Decomposition of sodium chloride and formation of hydrochloric acid; (2) the hydrochloric acid attacks the copper minerals to form volatile cuprous chloride; (3) the cuprous chloride is reduced to metallic copper on the carbon particles; and (4) the
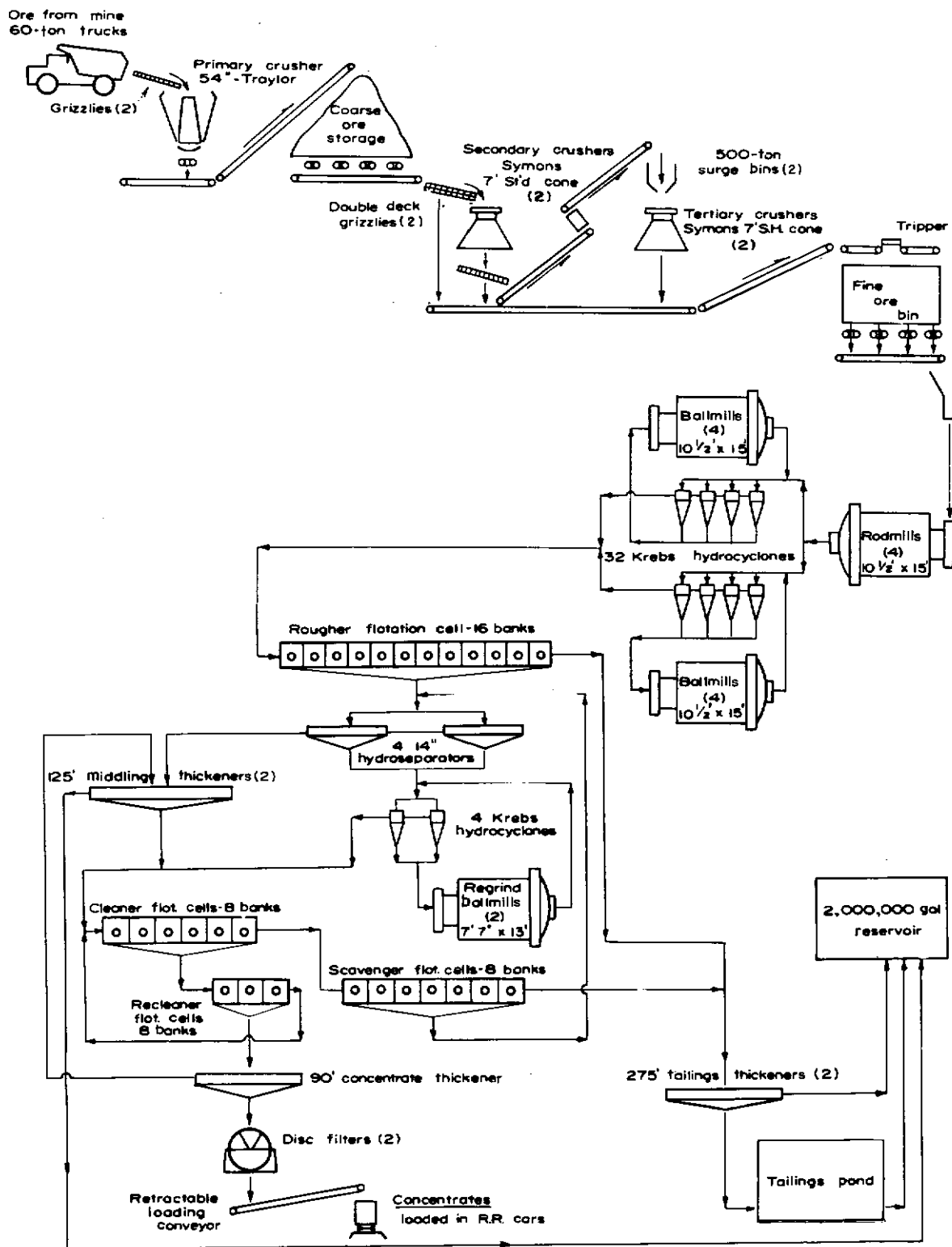
FIGURE 15.—Mission Mill Flowsheet.

(Courtesy, American Smelting and Refining Co.)

segregated copper is recovered by the conventional flotation process. Tests and results of pilot plant and commercial experience are described in a Bureau of Mines report and a journal article.[4]

HYDROMETALLURGY

Leaching is the term applied to the process of recovering the valuable metal or metals from an ore by dissolution with an aqueous solvent, leaving the gangue or waste material virtually unaffected. Subsequent recovery of the metal or metals from solution is effected through either chemical or electrolytic precipitation. For copper these processes are limited largely to copper-bearing materials that, because of grade, composition, or other considerations, are not amenable to concentration and pyrometallurgical extraction. Generally, hydrometallurgical methods are used to treat low-grade ores containing native copper, oxide, or mixed oxide-sulfide minerals because costs of other types of treatment are too high or the recoveries of copper are too low. Combinations of hydrometallurgical methods for recovery of oxide minerals coupled with flotation or sulfate roasting for recovery of the sulfide portion have been successful in treating some mixed ores.

Leaching

There are four principal methods used in leaching: (1) Leaching in place, (2) heap leaching, (3) leaching by percolation, and (4) leaching by agitation. The choice of method depends on disposition and grade of ore to be treated, as well as the character of the associated gangue.

Leaching in Place.—Ore bodies that contain sulfide minerals and that are shattered and broken so air and water have access to the ore are suitable for leaching in place. The copper sulfide minerals exposed are subjected to natural oxidation by alternate and intermittent contact with air and water to form water-soluble copper sulfate. The leach solution is accumulated in drainage tunnels driven under the ore body or in the lower workings of the mine. Leaching in place has been successfully applied in mines at Butte, Mont.; the Miami and Ray mines in Arizona; the Eureka mine, Ducktown, Tenn.; the Cronebane mine, County Wicklow, Ireland; and the Aznalcollar and Rio Tinto mines in Spain.

Heap Leaching.—Heap leaching is similar to leaching in place in that it depends on the natural oxidation of the sulfide minerals by continual contact with air and water. Some sulfide ores and mixed ores containing copper values too low grade to be processed profitably by any other method, but needing to be removed from the mine, can be treated by heap leaching. The process consists essentially of piling or dumping leaching ore, without crushing or other preparation, on an area having proper drainage to natural or prepared basins. When possible it may be advantageous to arrange the piles so that the coarser ore will be placed at the bottom and the finer ore on top, aiding ventilation and circulation of the solution. Leach solutions are directed over the piles and allowed to drain through the heap. The sulfide minerals that have been oxidized to sulfates are dissolved in the leaching solution, which collects in prepared basins. Copper is recovered from the pregnant leach solution by cementation on scrap iron. After removal of the copper, the spent solution is returned to the piles for another cycle of leaching.

This method is simple but the reaction is slow, and years may be required to obtain satisfactory extraction. Only large tonnages can be profitably treated by this method. The process has been the principal method of recovering copper at Rio Tinto, Spain, for hundreds of years. At one time there was about 20 million tons of ore, occupying 350 acres, undergoing treatment. In Arizona the method is used at the Lavender pit, Castle Dome, Morenci, Silver Bell, and Esperanza mines.

Percolation Leaching.—Percolation leaching is the method most widely used for commercial hydrometallurgical recovery of copper from large deposits of low-grade oxide ores. Ore charged to a number of leaching vats is successively treated with leaching solution, either by upward or downward percolation, or both. Usually the leach solution enters a vat at the bottom, percolates upward until the ore is flooded, is held there for a soaking period, and is then drained and advanced to the next vat where the process is repeated. The flow of the leach solution in this sequence completes a single cycle. The number of cycles necessary for optimum extraction depends on the solubility of the copper minerals in the ore. If six cycles are necessary for recovery of the copper from the ore, six charges or vats of ore would be undergoing simultaneous leaching.

Particle size and contact time, the two principal factors affecting leaching efficiency, are interdependent. Most copper ores to be treated by leaching are crushed dry because of the presence of soluble copper minerals, but dry crushing below $\frac{1}{4}$ inch becomes uneconomical. Higher grade ore may require crushing to

[4] Rampacek, Carl, W. A. McKinney, and P. T. Waddleton. Treating Oxidized and Mixed Oxide-Sulfide Copper Ores by the Segregation Process. BuMines Rept. Inv. 5501, 1959, 28 pp.
Freeman, G. A., Carl Rampacek, and L. G. Evans. Copper segregation at the Lake Shore Mine. J. Metals, May 1961, pp. 370–372.
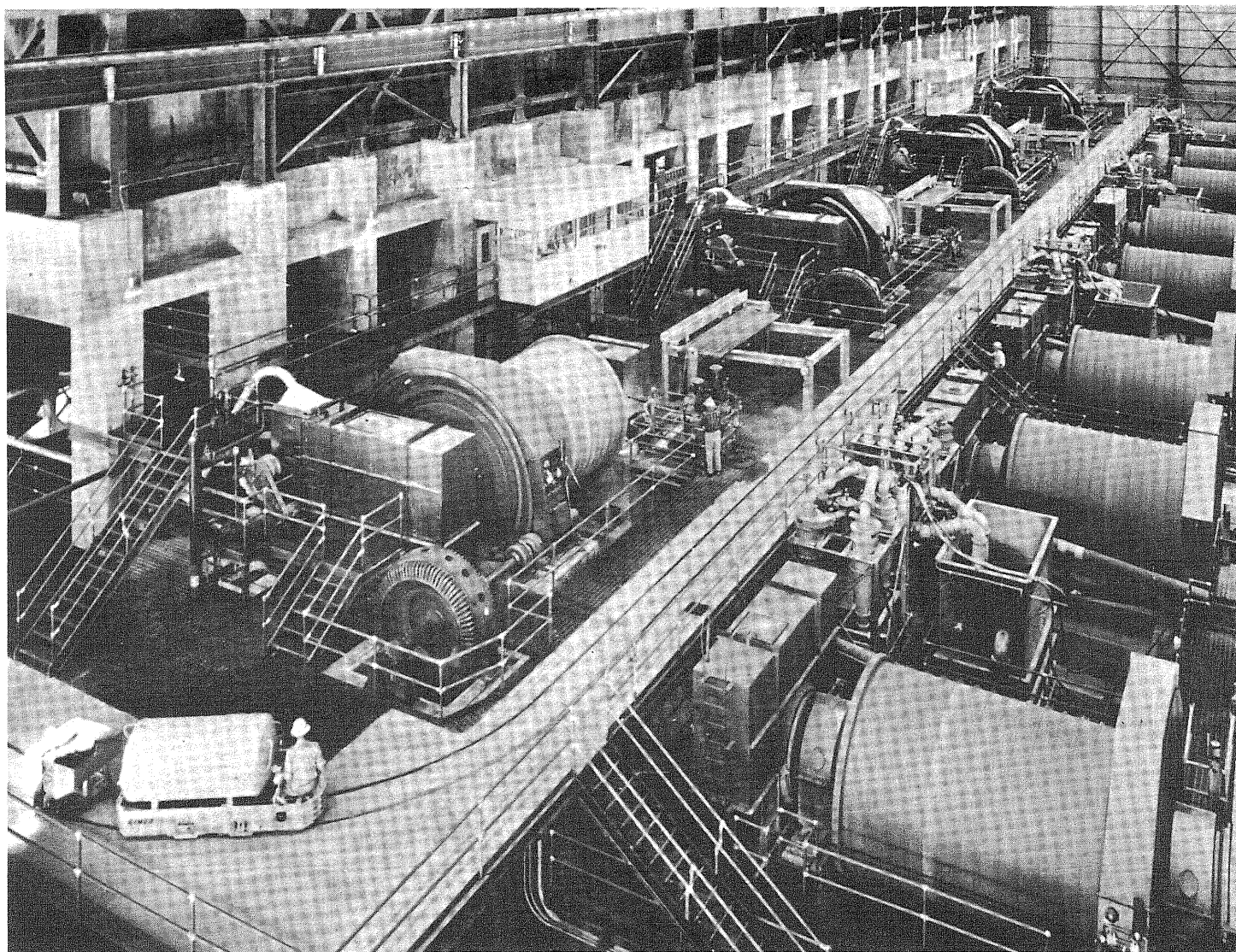
FIGURE 16.—Mission Mill Grinding Bay.
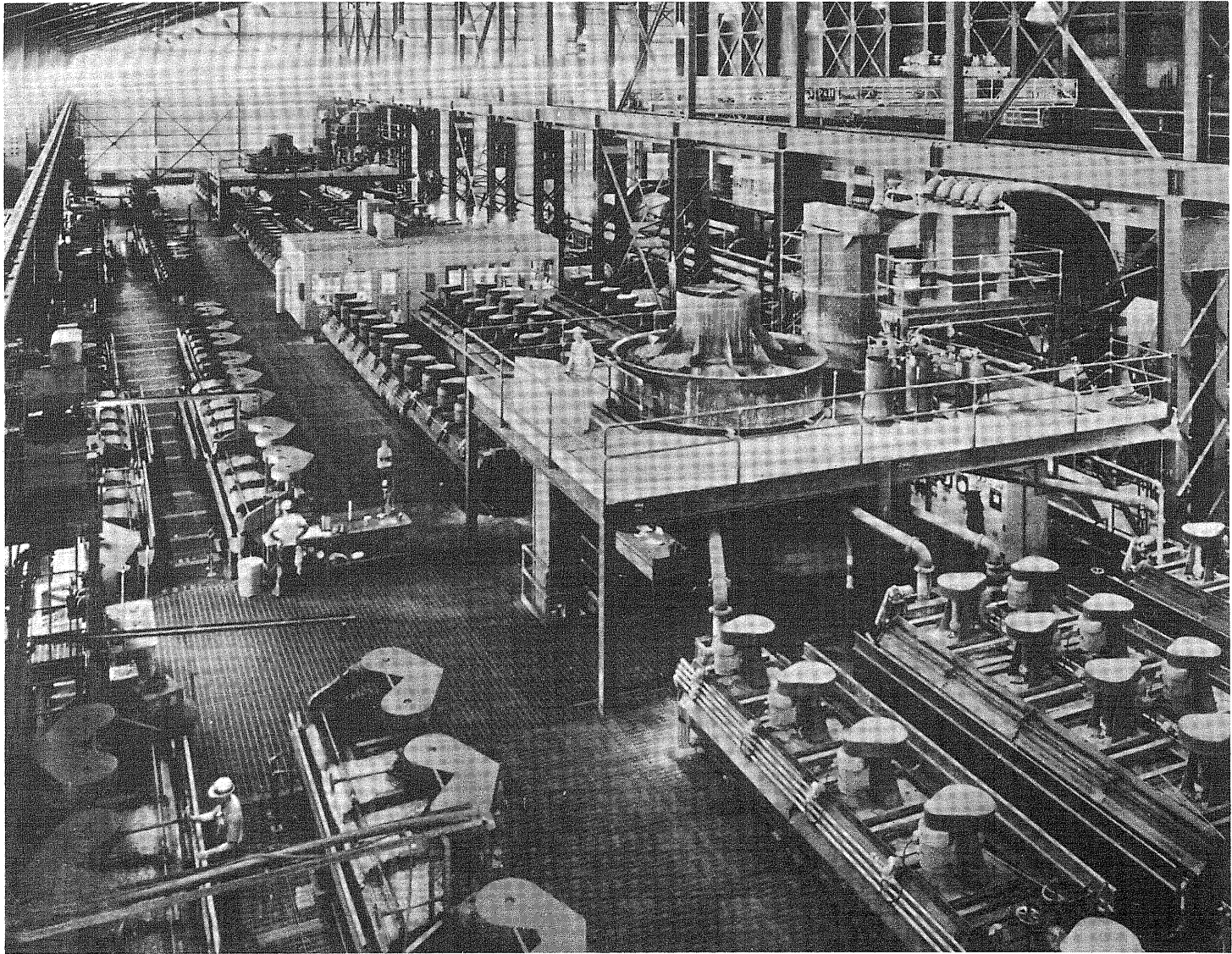(Courtesy, American Smelting and Refining Co.)

FIGURE 17.—Mission Mill Flotation Section.
(Courtesy, American Smelting and Refining Co.)

smaller sizes owing to grain size of the copper mineral and the degree of dissemination in the ore. Fine crushing or grinding is not only expensive, but the fine material produced reduces porosity in the vats and inhibits percolation of the leach solution. In fact, when fines or slimes are produced in crushing the ore, they are usually separated by classifiers and treated by suspension leaching methods.

Agitation Leaching.—Agitation leaching is applicable to those ores that cannot be effectively treated by percolation or to those operations where efficient extraction is required at a rapid rate. Dissolution of the copper minerals is effected by keeping finely divided ore particles in suspension in the solvent. Agitation is accomplished either mechanically or by forced air and must be strong enough to prevent settling in the tanks. The fine ore or concentrate may be leached on either a batch or continuous basis. For the batch process a charge of ore is agitated with the required amount of leach solution until optimum extraction is reached. In the continuous process, ore and leach solution are added simultaneously, but instead of effecting dissolution in a single tank, the ore-solution mixture passes from tank to tank, in series, until maximum extraction is obtained. Pachuca tanks with high lifts are convenient for this type of agitation. Separating residual gangue from leach solution and washing occluded solution from residual gangue are done in thickeners. Thickeners are standard equipment for separating a dilute pulp into a clear solution and a thickened pulp. When used in series, thickeners provide a convenient method for countercurrent washing.

Recovery

In current practice there are only two important methods for recovering copper from oxidized ores: (1) Direct leaching of ore followed by electrolysis of the leach solution or precipitation of copper from the leach solution with scrap iron (cementation), and (2) flotation of oxide minerals, leaching concentrates, and electrolysis of leach solution.

The choice of method depends largely on reserves and the size of operation planned. The oxide copper minerals in most ores are readily soluble in dilute sulfuric acid. Electrolysis of the leach solution produces refined copper with no further treatment required except melting for casting into shapes. The process also regenerates sulfuric acid which is recycled for leaching. The direct operating costs of producing copper by this method are said to be as low or lower than treating sulfide ores by flotation, smelting, and refining. Any departure from vat leaching-electrolysis usually is

based on compelling economic or technical considerations.

Chemical Precipitation.—Chemical methods for copper precipitation are those not using electrolysis. While there are a number of theoretically possible precipitants, the only one that has found commercial importance is iron in the forms of scrap iron and sponge iron. This process of precipitation is termed "cementation" and the copper precipitate formed is called cement copper.

Precipitation may be carried out in various kinds of containers (launders, tanks, towers), depending upon the nature of the iron used.

The precipitation reaction,

$$CuSO_4 + Fe \rightarrow Cu + FeSo_4,$$
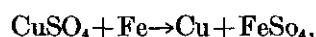
indicates that 1 pound of iron should theoretically precipitate 1.137 pounds of copper. If sponge iron is used as the precipitant, the amounts of iron required and cement copper produced approach the theoretical figures. If scrap iron is used, however, the nature of the scrap will regulate the amount needed and the grade of the cement copper produced. The type of scrap iron used at most precipitating plants is detinned and shredded tin cans, and about 1.4 pounds is used to precipitate 1 pound of copper. While the basic reaction for copper precipitation by iron is very simple, other reactions may occur during the operation that are related to the acidity of the solutions, the amount of ferrous and ferric iron and copper present, and the rate of flow. These affect iron consumption and the grade of copper produced. Free acid and ferric salts in solution can contribute to consumption of iron according to the following reactions:

$$H_2SO_4 + Fe = FeSO_4 + H_2, \text{ and}$$
$$Fe_2(SO_4)_3 + Fe = 3FeSO_4.$$
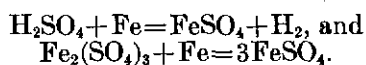
The advantages of this method are: (1) Simplicity of operation, (2) effective precipitation from all concentrations of copper in solution, and (3) precipitation not being affected by impurities in solution. Disadvantages are that (1) The process does not regenerate acid as in the electrowinning method; and (2) the cement copper assays are between 50 and 90 percent copper, requiring smelting and refining to be marketable as high-grade copper.

Precipitation From Ammoniacal Solution.—Copper dissolved in an ammoniacal solution forms complex cupric ammonium salt that hydrolyzes readily unless an excess of free ammonia is present. Simple heating to drive off excess ammonia converts the complex to the copper salt, which decomposes in solution with precipitation of copper oxide.

The only hydrometallurgical installation in operation utilizing an ammoniacal leach is that of the Lake Linden and Tamarack Divisions of Calumet & Hecla, Inc., at Hubbell, Mich. In treating ores or tailings, the coarser native copper is recovered by gravity concentration followed by classification and separation of plus 100-mesh sands for the ammoniacal leach. Minus 100-mesh material is treated by flotation for recovery of the fine native copper. The plus 100-mesh tailing, averaging less than 0.7 percent copper, represents about 10 percent of the original feed. A batch, downward percolation leaching method is used. Cupric ammonium carbonate is the active leaching agent.

Electrolytic Precipitation.—Electrolytic precipitation of copper from copper sulfate leach solutions, using insoluble anodes, is termed electrowinning and is the method used by several large producers for recovering copper. The principal advantages of the method are: (1) Electrolytically refined copper can be produced with no other treatment required; (2) sulfuric acid is regenerated, sometimes equal to or in excess of the amount needed for leaching, although usually some makeup acid is necessary; and (3) the smelting operation is eliminated.
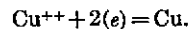
Grunenfelder explains the reactions of electrorefining and electrowinning, as follows:[5]

Electrolysis can be considered as consisting of two equivalent oxidation and reduction reactions, the oxidation taking place at the anode, the reduction taking place at the cathode. In electrorefining, the mechanics of the reactions are simplified to the extent that the oxidation of an equivalent of metal at the anode, proportional to the amount of current passing (Faraday's law), is accompanied by the reduction of the same equivalent amount of metal ion at the cathode. Disregarding behaviors of impurities, the composition of the electrolyte remains unchanged, and the net cell reaction, ideally, is anodic corrosion and cathodic deposition of equal amounts of the same metal. Because there is no decomposition potential involved, the voltage required is mainly that needed to overcome ohmic resistance of the electrolyte, or $E = IR$. The process takes advantage of factors which influence resistance, such as concentration, temperature, etc., and thus optimum electrolytes and electrolyzing conditions are chosen.

In electrowinning, the net cell reactions are also equivalent oxidation and reduction reactions, but in this instance, since insoluble anodes are used, the oxidation and reduction does not take place on equivalents of the same material. Further, then, the source of material undergoing reaction must be the electrolyte.

Specifically, for the electrowinning of copper from an electrolyte containing copper as a sulfate solution, the $CuSO_4$ must undergo the oxidation-reduction reactions. This again represents the ideal condition with no complication of impurities that could be present along with the $CuSO_4$ in solution. If $CuSO_4$

⁵ Grunenfelder, J. George. The Hydrometallurgy of Copper. Ch. from Copper, The Science and Technology of the Metal, Its Alloys and Compounds. Am. Chem. Soc. Reinhold Pub. Corp., New York, 1954, pp. 317–318.

is considered to ionize as Cu^{++} ions and SO_4^{--} ions, the reaction at the cathode is

$$Cu^{++} + 2(e) = Cu.$$

This reduction reaction is equivalent to that taking place in electrorefining. At the anode the oxidizing reaction is

$$SO_4^{--} - 2(e) = SO_4.$$
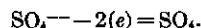
The sulfate radical is not stable and immediately reacts with water in the electrolyte to form sulfuric acid and oxygen according to the following reaction:

$$SO_4 + H_2O = H_2SO_4 + \tfrac{1}{2}O_2.$$
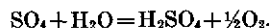
It will be noted that in electrolytic decomposition of a solution of $CuSO_4$ the solution will be depleted of one equivalent of copper for each equivalent of copper deposited at the cathode, the solution will be enriched by the formation of one equivalent of sulfuric acid, and one equivalent of oxygen will be liberated at the anode.

The voltage required for the electrolysis is not the simple IR drop as noted for electrorefining but includes, in addition to the IR requirements, enough added potential to decompose the $CuSO_4$, plus potential to liberate the O_2 formed at the insoluble anode. The latter is referred to as the gas overvoltage.

Since the net cell reaction is known, the reaction potential can be calculated.

$$CuSO_4 + H_2O = Cu + H_2SO_4 + \tfrac{1}{2}O_2.$$
$$\Delta H = +56,620 \text{ cal}$$
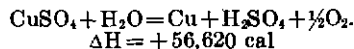
The reaction being endothermic, electrical energy equivalent to 56,620 calories of chemical energy must be supplied for the decomposition of one mole of $CuSO_4$.

The electrodes used in electrowinning are copper starting sheets for cathodes and some type of insoluble anode that will neither react chemically with the electrolyte nor be oxidized, either by the current or the oxygen liberated. Antimonial lead anodes are favored, particularly where chlorides and nitrates create no special problems. Anodes may be cast or of rolled sheet but are often of grid construction to reduce weight and lower anodic current density. Cell construction, electrical hookups, and the cathode sheets are similar to those used in electrorefining.

The power requirements for electrowinning are 8 to 10 times greater than for electrorefining but the power costs are largely offset by the lower labor costs. Anode handling and scrap remelting, which are major cost items in electrorefining, are at a minimum in electrowinning since the insoluble anodes may have a service life of 10 years.

The electrowinning process is used to recover copper from leach solutions by the Inspiration Consolidated Copper Co. in Arizona; Chile Exploration Co. at Chuquicamata, Chile; Andes Copper Mining Co., Potrerillos, Chile; Union Minière du Haut Katanga at Jadotville, Republic of the Congo; and Nchanga Consolidated Copper Mines, Ltd., Northern Rhodesia. Operating details for four of these plants are shown in table 12.

SMELTING

There are three main types of copper ores—native copper, oxidized, and sulfide. Since most of the world copper production is extracted from low-grade sulfide ores that require concentration, the dominant metallurgical processes for recovering copper metal are adapted to treating fine-grained sulfide concentrate. The usual concentrate consists mainly of copper sulfide, iron sulfide, and gangue, and the purpose of smelting is to separate copper from the iron, sulfur, and gangue. The strong affinity of copper for sulfur and its weak affinity for oxygen, compared with iron, sulfur, and other base metals of the ore, form the basis for the three major steps in smelting—roasting, reverberatory furnacing, and converting.

Direct smelting of oxidized ores in blast furnaces began losing importance after introduction of flotation, which produced a fine material not adaptable to treatment in blast furnaces. Since concentration of low-grade oxidized ores results in high tailings losses, direct leaching of such ores followed by cementation and reverberatory smelting of the precipitates is common practice.

The treatment of native copper ores of the Lake Superior district in Michigan departs from the general pattern in that there is no formation of matte and no converting step.

Roasting.—Roasting is a pyrometallurgical process of heating an ore or concentrate in an oxidizing atmosphere to effect oxidation reactions. The object of roasting sulfide ores or concentrates of copper is to oxidize sulfur and iron and to move volatile impurities such as antimony, arsenic, and bismuth. Copper and iron sulfides are among the easiest of the common metallic sulfides to convert to oxides, with minimum production of sulfates and other intermediates; furthermore, mixtures of copper and iron sulfides relatively free from inert compounds are readily ignited and will roast autogenously to 6 to 8 percent sulfur without further application of heat.

The amount of sulfur in the reverberatory furnace charge determines the grade of matte formed. Ores high in iron and sulfur may produce mattes too low in copper. The proportion of sulfur in such ores is reduced before smelting by roasting them under oxidizing conditions.

Roasting practice has developed along two principal lines—hearth roasting and blast roasting. Various types of multiple-hearth roasters are preferred in copper smelters to roast sulfide feed for reverberatory smelting.

Roasting for copper blast furnaces is usually accomplished in moving-bed sintering equipment to agglomerate and partially roast fine-grained ores and concentrates.

The concentrate produced from some sulfide ores may be controlled at the concentrator to avoid roasting before reverberatory smelting. In some of the newer smelters roasters were not installed, and at some of the older ones the roasters are used as dryers. It is apparent that roasters are gradually being abandoned in copper pyrometallurgy.

Fluid-bed roasting, used at some plants for roasting copper ores is a new roasting technique adapted from the suspended-solids technique of the petroleum-refining industry. The method consists of partial suspension of solid particles in a gaseous stream. The gas velocities are such that size segregation of properly prepared feed will not occur. The entire fluidized bed, which is in turbulent motion like a boiling liquid, is substantially uniform throughout. Copper sulfides can be sulfatized by fluosolid roasting, making the copper soluble in water or weak acid. Calcines have consistently been obtained in which as much as 90 percent of the copper is water soluble, with an additional 5 to 9 percent soluble in 5-percent sulfuric acid. At the same time, close temperature control allows only 1 to 1.5 percent of the iron to go into solution.

Reverberatory Furnace Smelting.—Reverberatory furnaces are long roomlike structures from 80 to 130 feet long and 10 to 35 feet wide. Capacities range from 50 to 1,500 tons of charge per day. The fuel and charge are kept separate. Oil, gas, or pulverized coal is burned in a separate compartment from which the flame and hot gases pass over the ore. Heating of the charge is accomplished by radiation from the roof and side walls rather than by direct contact with the hot gases. There is little or no reaction between the gases in the furnace atmosphere and the material on the hearth; it is possible to get some oxidation of the charge by using a large excess of air for combustion, but this wastes heat and is seldom practiced. The principal chemical reactions that take place in the charge of a reverberatory furnace are between the various constituents to form a matte and a slag. Reverberatory slags are essentially mixtures of silicates and alumino-silicates of iron and lime. In many cases, the charge derived from copper-bearing materials is self-fluxing; but usually fluxes, such as limestone, are added to obtain a suitable slag. Analyses of typical concentrates, mattes, and

TABLE 12.—*Electrowinning of copper*

	Inspiration Consolidated Copper Co., Inspiration, Ariz.	Andes Copper Mining Co., Potrerillos, Chile	Union Minière du Haut Katanga, Katanga, Republic of the Congo
Method of leaching	Upward percolation	5 to 6 days counter-current	Continuous agitation in Pachuca tanks.
Leaching vats:			
Material of construction	Concrete, leadlined	Reinforced concrete, mastic and acid-proof brick lined.	7 rows of 4 Pachuca tanks, steel, lead lined.
Length by width by depth	175'0"X67'6"X18'0"	105'0"X115'0"X19'6"	11' diamX27'5½" high.
Average charge per tanks___tons	9,000	8,603	5 tons/hr (dry feed).
Circulation___gpm	1,800	4 air lifts circulation 2,200, advance 800.	
Ore, percent:			
Copper	1.087	0.831	Flotation concentrates (oxides). 28.
Copper, acid soluble		0.711	Practically 100.
Oxide Cu	0.527		28 approximately.
Sulfide Cu	0.560	0.120	0.2.
Tailings, percent:			
Total Cu	0.148	0.249	0.5-1.
Water-soluble Cu	Trace	0.024	
Oxide Cu, acid-soluble	0.014	0.126	0.5.
Sulfide Cu	0.134	0.099	0.2.
Extraction, percent:			
Total Cu	86.385	71.07	97.5.
Oxide Cu	97.343	79.62	
Sulfide Cu	76.071	25.8	

	Starting sheets	Deposition tanks	Starting sheets	Deposition tanks	Starting sheets	Deposition tanks
Electrolyte to cells:						
Specific gravity		1.209	1.238	1.133		1.270.
Copper___grams per liter	30-40	31.1	46.8	29.3	35-40	66.
Total H₂SO₄___do	120-150		166.6	30.0		
Free H₂SO₄___do		20.1			105	6.7.
Cl₂___do			0.003	0.07		
Total Fe___do		20.5	3.18	2.24		2.5.
Ferric Fe___do		5.7	0.10	0.22		1.5.
Ferrous Fe___do			3.08	2.02		1.0.
Temperature___°C	55	35.8	56.2	29.5		
Electrolyte from cells:						
Specific gravity			1.248	1.106		
Copper___grams per liter		26.4	46.6	10.65		30.
Total H₂SO₄___do			167.6	60.4		
Free H₂SO₄___do		28.9				62.1.
Cl₂___do			0.006	0.08		
Total Fe___do			3.30	2.10		2.5.
Ferric Fe___do		12.0	0.05	1.16		1.3.
Ferrous Fe___do			3.25	0.94		1.2.
Temperature___°C		41.8	48.7	38.5		55.
Current:						
Total___amperes					¹ 20,000	¹ 20,000.
Current density___amp/sq ft	14-16	11.4	15.93	9.78	16	16.
Current efficiency___percent		71.65	92.60	83.95	90-92	90-92.
Voltage per tank		2.16	0.359	2.066	0.6X3=1.8	2.05-2.2X3=6.1-6.6.
Copper___kwh/lb		1.347	0.168	1.062	0.275	0.93.
Copper___lb/kw-day			161.92	25.229		25.8.
Anodes:						
Material	Blister Cu	8 percent-Sb lead.	Blister Cu	83.5 Pb, 15.0 Sb, 1.3 As, 0.15 Cu	Cast copper	6 percent-Sb hard lead.
Length, width, thickness		3'2"X3'4"X½"	3'3¼"X2'2½"	3'3¼"X2'2½"	3'9½"X2'9¼"X1¾".	3'9¹⁵/₁₆"X2'5⅛"X⁷/₁₆".
Spacing		4.0"	4½"	3½"	4¾"=120 mm	3½"=90 mm.
Weight___pounds	1,100	340	550-600	200	780	250.
Mode of suspension			Cast lugs	Cast lugs	Cast lugs	Copper bar, cast-in.
Life in days			35	Corrosion equals 2 lb of anode/ton of Cu deposited.	15-20	
Scrap___percent	22.0		18		20-25	
Cathodes:						
Material	Rolled Cu		Rolled Cu sheet	Starting sheets	Cu blanks	Cu starting sheets.
Length by width by thickness	3'8"X3'6"	3'8"X3'6"	2'4¾"X3'7½"	2'4"X3'1½"	3'11½"X2'10¼"X³/₁₆".	3'5¼"X2'9¼".
Weight___pounds		100	80	68.48		110-120.
Mode of suspension		Punched loops	Riveted bars	2 loops	Riveted copper bars	Punched loops.
Replaced after(?) days	1	7	1	8-12	12	6.
Weight starting sheet___pounds	12-13	13-14	6.91	7.66	6-6½	
Percent starting sheets used for loops, scrap, etc.	15		20.27	Loops, 10.85	15	
Deposition tanks:						
Number	17-18	120	54	576	15	145.
Length by width by depth		33'0"X4'0"X4'3"	10'3"X2'10"X3'9".	10'3"X2'10"X3'9".	62'6"X3'2¼"X4'2½".	62'6"X3'2½"X4'2½".
Number of anodes, cathodes	96, 95	98, 96	25, 24	34, 33	50X3	65X3.
Electric connection			Cast-rolled Cu Bus bars.	Cast and rolled Cu busbars.	3 groups series-electrodes in multiple.	(²).
Material of construction			Reinforced concrete lined with 7 percent-Sb lead.	Reinforced concrete lined with 7 percent-Sb lead.	Reinforced concrete, mastic-lined asphalt.	(²).
Circulation___gpm		100	10	29-44		7.92.

See footnotes at end of table.

TABLE 12.—*Electrowinning of Copper*—Continued

Chile Exploration Company, Chuquicamata, Chile

Method of leaching	Batch percolation
Leaching vats:	
Material of construction	Reinforced concrete lined with 4" of 27% asphalt sand mixture
Length by width by depth	150'×110'×(18.5' and 19.5')
Average charge per tank...tons	13,133
Circulation...gpm	3,000
Ore, percent:	
Copper	1.809
Copper, acid soluble	1.477
Oxide Cu	1.477
Sulfide Cu	0.332
Tailings, percent:	
Total Cu	0.343
Water-soluble Cu	0.030
Oxide Cu, acid-soluble	0.056
Sulfide Cu	0.287
Extraction, percent:	
Total Cu	81,963
Oxide Cu	96,357
Sulfide Cu	17,880

	Starting sheets		Deposition tanks			
	Commercial	Refined	1st stage, 70,093	2d stage, 19,491	Plating down, 3,638	Whole house, 93,222
Electrolyte to cells:						
Specific gravity	1.130	1.225	1.115	1.110	1.110	1.114.
Copper...grams per liter	38.921	52.10	24.96	20.28	17.36	23.69.
Total H_2SO_4...do	93.76	201.54	94.74	96.19	85.22	94.67.
Free H_2SO_4...do	35.38	123.39	57.30	65.77	59.18	59.14.
Cl_2...do	0.049	0.04	0.09	0.09	0.12	0.09.
Total Fe...do	2.81	3.44	2.87	2.82	2.69	2.85.
Ferric Fe...do			1.16	1.69	1.05	1.26.
Ferrous Fe...do	2.81	3.44	1.71	1.13	1.64	1.59.
Temperature...°C	28.3	47.6	32.6	34.2	33.4	33.4.
Electrolyte from cells:						
Specific gravity	1.127	1.225	1.110	1.105	1.095	1.108.
Copper...grams per liter	35.766	52.10	19.71	15.96	5.78	18.38.
Total H_2SO_4...do	93.75	201.54	95.51	94.98	85.22	95.07.
Free H_2SO_4...do	40.10	123.39	65.95	71.04	76.55	67.50.
Cl_2...do	0.049	0.04	0.09	0.10	0.12	0.09.
Total Fe...do	2.88	3.44	2.83	2.83	2.69	2.82.
Ferric Fe...do	0.73		1.88	1.95	1.70	1.88.
Ferrous Fe...do	2.15	3.44	0.95	0.88	0.99	0.94.
Temperature...°C	29.5	47.6	36.0	36.4	39.3	36.2.
Current:						
Total...amperes	25,319	25,060	24,538	23,155	26,957	24,538.
Current density...amp/sq ft	16.22	21.2	14.20	13.40	15.60	14.20.
Current efficiency...percent	86.89	84.79	85.53	84.05	81.13	84.87.
Voltage per tank	2.12	0.46	2.00	1.96	2.02	2.00.
Copper...kwh/lb	0.932	0.208	0.891	0.892	0.951	0.900.
Copper...lb/kw-day	25.76	115.42	26.80	26.91	25.24	26.67.
Anodes:						
Material	Pb-Sb-Ag	Soluble anode	Pb-Sb-Ag	Pb-Sb-Ag	Pb-Sb-Ag	Pb-Sb-Ag.
Length, width, thickness	4'2¼"×3'1¼"×⅛".	4'¾"×3'1¾"	4'1"×2'9"×⅛".	4'1"×2'9"×⅛".	4'1"×2'9"×⅛".	4'1"×2'9"×⅛".
Spacing...inch	3	4½	3	3	3	3.
Weight...pound	310	789	210	210	210	210.
Mode of suspension	Cu insert	Cast Cu lugs	Cu insert	Cu insert	Cu insert	Cu insert.
Life in days		19				
Scrap...percent		20				
Cathodes:						
Material	Cu blank	Cu blank	Cu sheet	Cu sheet	Cu sheet	Cu sheet.
Length by width by thickness	4'1"×3'1"×¼"	4'1"×3'1"×¼"	3'0"×4'0"	3'0"×4'0"	3'0"×4'0"	3'0"×4'0".
Weight...pound	150	150	150	150	150	150.
Mode of suspension	Riveted to bar	Riveted to bar	2 loops punched	2 loops punched	2 loops punched	2 loops punched.
Replaced after (?) days	Stripped daily	Stripped daily	5-8	5-8	5-8	5-8.
Weight starting sheet...pound	11	16	11	11	11	11.
Percent starting sheets used for loops, scrap, etc.	19	19	19	19	19	19.
Deposition tanks:						
Number, average	72.48	30.73	616	152	67.67	835.67.
Length by width by depth	19'2"×3'1"×4'10".	19'2"×3'11"×4'10".	19'2"×3'11"×4'10".	19'2"×3'11"×4'10".	19'2"×3'11"×4'10".	19'2"×3'11"×4'10".
Number of anodes, cathodes	63, 62	48, 47	73, 72	73, 72	73, 72	73, 72.
Electric connection	Whitehead	Walker	Whitehead	Whitehead	Whitehead	Whitehead.
Material of construction	Mastic-lined concrete	Mastic-lined concrete	Mastic-lined concrete	Mastic-lined concrete	Mastic-lined concrete	Mastic-lined concrete.
Circulation...gpm	36.41	3-5	334	376	333.7	342.

[1] Two electrical circuits each of 80 tanks in series. Six motor-generator sets: 2×4,000 amp 500 volts (1 spare). Five generators of 4,000 amp, 20,000 amp for each circuit.

Source: Mantell, C. L. Electrochem. Eng., McGraw-Hill Book Co., Inc., New York, 1960, pp. 204–207.

slags at smelters in Montana, Utah, and Arizona are shown below:

	Cu, percent	Fe, percent	S, percent	SiO₂, percent	CaO, percent	Al₂O₃, percent
Concentrate__	24. 8	23	33	12. 9	---	2. 5
Matte_____	45. 0	23	24	----	---	----
Slag_____	. 56	42. 5	----	34. 5	5. 8	6. 3
Concentrate__	31. 3	22. 7	29. 8	14. 5	---	. 09
Matte_____	43. 4	22. 0	25. 0	----	---	----
Slag_____	. 43	34. 5	----	39. 2	6. 8	4. 8
Concentrate__	23. 3	24. 2	31. 0	9. 4	2. 4	2. 4
Matte_____	31. 6	35. 0	25. 0	----	---	----
Slag_____	. 40	34. 2	----	37. 0	5	7. 2
Concentrate__	15. 4	31. 0	36. 9	11. 0	---	4. 3
Matte_____	27. 2	40. 7	24. 0	0. 8	---	1. 3
Slag_____	. 34	36. 1	1. 3	37. 6	3. 5	9. 6

The reverberatory furnace is constructed on a massive and heavily reinforced concrete foundation that supports the sides and end walls and encloses the furnace bottom. The hearth is 24 to 30 inches thick and is lined with a refractory material such as silica or magnesite brick. The sidewalls are usually of silica brick, but if a magnesite hearth is used a layer of chrome brick is laid to separate the acid sidewalls from the basic bottom. The more recent reverberatory furnaces have the Detrick-type suspended arch, which permits easy fettling and arch maintenance. Charge hoppers are arranged in a row above and along each side of the furnace. Slag is tapped from the upper end of the furnace into launders leading to slag cars; matte is tapped from near the burner end into ladles for transfer to the converters.

Blast-Furnace Smelting.—The copper blast furnace is used mainly for ores more than 1 inch in diameter, although fines may be treated if sintered. A few plants still use blast furnaces because they are useful under certain conditions, but they account for very little of the world copper supply.

The blast furnace is essentially a long, narrow, water-jacketed upright shaft. The length of the furnace varies considerably, being proportioned according to capacity. The size of a furnace is expressed in terms of the dimensions at the tuyère level. The width, being limited by ability of the blast to penetrate the charge, is usually 44 to 48 inches, though some furnaces are as narrow as 30 inches and others as wide as 56 inches. The molten products, slag and matte, collect and are drawn off at the bottom, and new material is charged at the top to keep the charge-level relatively constant.

Blast-furnace fuel is almost invariably coke, charged with ore and flux and ranging from 13 to 17 percent of the charge. The coke burden may be adjusted to control the degree of sulfur elimination, hence it is seldom necessary, unless sintering is required, to roast ores in advance of matte blast-furnace smelting.

Air under low pressure enters the furnace through tuyères near the bottom of the furnace above the hearth. The tuyères are pipes 4 to 6 inches in diameter, spaced 10 to 18 inches apart along both sides, and connected with a bustle pipe that carries the main air supply.

The lower part of the furnace walls usually consists of steel water jackets, while the upper part of the walls may be constructed of either refractory brick or a second tier of water jackets. In copper blast furnaces, the crucible is used as a collecting trough from which the matte and slag flow to a forehearth or settler in which the slag rises to the top and is removed almost continually.

Blast furnaces are still used at the following plants: Union Minière du Haut Katanga, Elizabethville, Republic of the Congo; The Rio Tinto Co., Spain; Mount Lyell M. & R. Co., Ltd., Tasmania; Tennessee Corp., Copperhill, Tenn.; Falconbridge, Ontario, Canada; Phelps Dodge Corp., Laurel Hill, N.Y.; and International Nickel Co., Coniston, Ontario, Canada.

Electric Furnace Smelting.—Electric smelting of copper concentrates to produce matte is used when the cost of electric power is less than coal or other fuels. The chemistry of the electrothermic reduction is essentially the same as in reverberatory smelting; interaction of the components of the charge is the same, with formation of matte and slag and elimination of some sulfur as sulfur-dioxide gas.

The Bolidens Copper Co. in Sweden operates an electric matte smelting furnace that is similar in design to the reverberatory furnace. It is 77 feet long by 20 feet wide with six Söderberg electrodes installed through the centerline of the roof. Each electrode is 3.9 feet in diameter and is connected to a 2,000-kva transformer for a total furnace energy of 12,000 kva.

Calcines from the roasters are charged directly into the furnace through charging pipes in the roof. Three tapholes for drawing off slag are at different levels at one end of the furnace. The slag, which is tapped almost continuously, flows through covered launders to a granulating plant where it is sprayed by jets of water; the mixture of granulated slag and water is pumped to the slag dumps. Matte is tapped at a lower level through three holes at the opposite end of the furnace. The matte averages approximately 35 percent copper; the slag, 0.3 to 0.5 percent.

Some advantages of electric smelting have been noted as follows:

1. Thermal efficiency is greater than that of a reverberatory furnace because there is no volume of high-temperature combustion gases carrying away much of the generated heat.

2. There are no combustion gases to dilute the sulfur dioxide (SO_2) gas, and the smaller amount of gas is much richer in SO_2.

3. Outstanding over the reverberatory furnace is the ability to reduce magnetite in hot and cold converter slags returned to the furnace. In the electric furnace the electrodes can be lowered to increase the heat of the bath; by adding fine pyrite and siliceous material, the magnetite (Fe_3O_4) is reduced to FeO which combines with the slag.

Converting.—Converting is the final stage in smelting copper sulfide ores or concentrates and is accomplished by blowing thin streams of air through the molten matte (product of the reverberatory or blast furnace) in a refractory-lined converter.

The first or white-metal stage of converting is rapid oxidation of the iron sulfide in the matte to iron oxide and sulfur dioxide. Enough silica is supplied to form an iron silicate slag. The copper remains as copper sulfide until most of the iron has been oxidized. When slagging is complete, the slag is poured off, leaving molten copper sulfide—known as white metal at this stage. Blowing is then continued in the second or blister stage to oxidize the sulfur of the white metal, leaving metallic copper. The first stage is strongly exothermic and raises the temperature of the bath high enough to form slag and to provide enough superheating for the second stage, which does not liberate as much heat.

A low-grade matte produces a larger amount of slag to be retreated for copper recovery and requires more silica than a high-grade matte; but if the matte is too high in copper and too low in iron, too little heat may be liberated to maintain the process. Heat can be supplied with *pulverized coal or fuel oil to convert* copper mattes that contain as much as 60 percent copper. Table 13 shows analyses of matte treated and converter slag produced at forty worldwide copper smelters.

There are two principal types of copper converters, the Pierce-Smith and the Great Falls. Both types are mounted on trunnions for tilting to receive the charge and pour off slag and metal, and both have a row of tuyères in the rear connected with a wind box to introduce air into the charge. Of the 40 plants shown in Table 13, 29 use Pierce-Smith (horizontal) converters, 7 use Great Falls converters (upright). Most of the larger plants use 13-by 30-foot, Pierce-Smith converters, but 14 plants report having converters about 10 to 12 feet in diameter and about 20 feet or less in length. Figures 18 and 19 show the pouring of blister copper at United States and U.S.S.R. smelters.

Tuyères range in diameter from 1¼ to 2 inches; in the newer plants the larger sizes are preferred. They are placed 8 to 12 inches apart and high enough above the bottom to clear the level of the metal at the finish of the blow. Frequent punching of the tuyères is necessary, especially during the blister stage. Each tuyère is equipped with a special valve for punching the tuyère without dropping the air pressure and allowing the molten bath to run out through the tuyère. Mechanical tuyère punching is employed at some plants.

Air is supplied at 10 to 20 pounds per square inch, and the amount required ranges from 160,000 to 200,000 cubic feet per ton of blister produced from 40-percent matte.

Two types of converter linings are used in modern practice—basic (magnesite) and acid (silica). Basic linings are almost universally preferred, chiefly because of lower operating and maintenance costs. Basic linings have a much longer life than acid linings, converting 2,000 to 2,500 tons of copper per ton of lining, compared to 10 to 100 tons with the acid type. Other advantages of the basic lining are use of larger vessels, ability to convert low-grade mattes with relatively little lining consumption, and less handling of the shells. The disadvantages of the basic lining are excessive punching of the tuyères necessitated by formation of magnetic iron oxide about their mouths, greater time required for lining and repairing, and excessive blowing-out of fine, siliceous ore during charging of the flux.

Converter slag contains 2 to 5 percent copper and is returned to smelting furnaces for re-treatment, where its high-iron content is generally an aid to fluxing.

When the converter cycle is finished the converter is tilted to discharge the copper metal into ladles in which it is transferred to the anode furnace and casting machines. At small plants casting molds are assembled on trucks, which are passed in front of a launder, receiving the copper directly from the converter.

Due to the rough upper surface when solidified, the product of the converter is known as blister copper. The rough surface is a result of the expulsion of gases, largely air and sulfur dioxide, as the molten metal cools.

Blister copper contains most of the precious metals of the charge and minor quantities of certain other elements; hence it is invariably refined before being marketed.

A 13- by 30-foot, Pierce-Smith converter holds 200 tons of matte and at 1.2 cycles for 24 hours has a capacity of nearly 100 tons of copper per day on 37-percent matte. Within moderate limits, the capacity varies 5 tons with each 1-percent variation in matte tenor. A 12-foot-outside-diameter, Great Falls converter may take an initial charge of 10 tons of 40-percent matte and by four successive blows and charges may yield 20 tons of blister copper per 6 to 12 hours or 40 to 80 tons a day.