FINAL REPORT

on

ENERGY USE PATTERNS IN METALLURGICAL AND NONMETALLIC MINERAL PROCESSING

(PHASE 9 - AREAS WHERE ALTERNATIVE TECHNOLOGIES SHOULD BE DEVELOPED TO LOWER ENERGY USE IN PRODUCTION OF HIGH-PRIORITY COMMODITIES)

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UNITED STATES BUREAU OF MINES

August 25, 1976

(Contract No. S0144093)

BATTELLE
Columbus Laboratories
505 King Avenue
Columbus, Ohio 43201

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ENERGY USE PATTERNS IN METALLURGICAL AND NONMETALLIC MINERAL PROCESSING

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INTRODUCTION

This report is for Phase 9 of the study on "Energy Use Patterns in Metallurgical and Nonmetallic Mineral Processing". The report covers the 14 high-priority commodities for which process flowsheets were presented in the Phase 4 Interim Report.

A major objective of this study is to identify operations where alternative technologies could yield lower energy consumption than the conventional processes covered in the Phase 4 Interim Report. Phase 9 of this program is directed toward the identification of those operations which would most benefit by the introduction of feasible new energy-conserving technology.

The approach used to fulfill the Phase 9 objective is as follows:

(1) Concentrating attention on those materials whose production requires large amounts of energy on a unit or on a total annual production basis (as defined in previous tasks of this study). The purpose is to define those operations

in which the introduction of new energy-conserving technology could have a major energy reduction impact.

(2) Identifying and examining those operations which could be replaced by previously recognized high-efficiency operations used in the processing of other materials.

(3) Making a direct comparison of the thermodynamically-defined minimum energy for production of a material and the energy currently used for that production. A comparison of the calculated free energy for a given reaction and the actual energy consumed in practice may provide a means of identifying those manufacturing steps which are relatively inefficient and where alternative processing steps and/or new technology might yield major energy savings.

METHODOLOGY

In the Phase 9 study the production processes for each of the high-priority commodities were studied by the technical investigator(s) who prepared the flowsheets and characterizations previously reported in the Interim Reports on Phases 4 and 8. In addition to the original investigator, the Task Leader for Phase 9 reviewed the processing steps for each commodity. During this effort the steps in each process were examined to ascertain how energy savings might be achieved via the utilization of alternative processes.

For each of the 14 high-priority commodities reviewed in the Phase 4 and Phase 8 Interim Reports, a section has been included in this report which evaluates the need for and possibility of (based on technical feasibility) introducing new energy savings processes. Where practical, estimates of the energy saving which might be gained through the introduction of the major process changes are included. In some instances, it was not possible to assign a quantitative value to the energy saving and, in those cases where the energy saving potential for a given process could be estimated, it was frequently not possible to assign a total potential saving for the entire industry.

In general, the need for the introduction of alternative production processes was judged on the basis of (1) the energy required per ton of product, (2) the efficiency of the unit processes involved, and (3) the complexity of the production route, in terms of number of possible alternative chemical reactions, to the same end product.

If the energy consumed in the production of a net ton of primary product was large (relative to the other high-priority commodities) and the unit operations used were thermally inefficient then the need for developing alternative energy-saving unit operations and/or production processes was considered great. On the other hand, if the energy requirement for a net ton of product was low and the production of the commodity involved basically one primary reaction, such as in the calcining of limestone to produce lime, then the opportunity for the development of alternative processes was taken to be small. These judgments are not absolute. For instance, there is no intention to overlook the potential gains in energy consumption which might occur by the use of alternative processes in the lower-energy consumption materials and in those operations currently judged to be relatively efficient from an energy usage point of view.

In preparing the Phase 9 report not only were the Phase 4 and 8 reports major input sources, but also a recent report prepared by Battelle's Columbus Laboratories for the U.S. Federal Energy Administration (FEA) entitled "Evaluation of the Theoretical Potential for Energy Conservation in Seven Basic Industries"* was reviewed and used in considering the efficiency of operations

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*This report, prepared under Contract No. 14-01-0001-1880 and dated July, 1975, was concerned with the following industries: steel, copper, aluminum, glass, synthetic rubber, selected plastics, and paper. Only the first four of these are included within the high-priority materials considered herein.

for four major energy-consuming commodities, aluminum, copper, glass, and steel. As part of the above cited-FEA report, the theoretical minimum energy required to produce the commodity and the efficiency (i.e., useful energy output/total energy input x 100) based on the First Law of Thermodynamics, was analyzed for unit processes in seven major industries.

Three other measures of process performance which incorporate the concept of availability based on the Second Law of Thermodynamics were also evaluated for selected unit processes and the seven industries as part of the FEA report. A summary of the numerical results of the thermodynamic analyses for aluminum, copper, glass, and steel industries and specific unit operations within those industries is given in Table 1.

The efficiency values given in Table 1 are the ratios of useful output energy to total input energy (as based totally on the First Law energy analysis). The useful output of the industries was taken as the energy of all principal products and by-products which were not consumed in the industry.

The effectiveness values were based on the calculated "availability" for each material stream in the process being considered. The thermodynamic property, availability, is defined as the maximum work which can be obtained from a material or material stream by bringing the material to complete equilibrium with the environment by reversible processes. Thus availability is due not only to pressure and temperature differences, but also due to differences in chemical potential and electrical potential between the material and the environment. Availability, unlike energy, can be destroyed, and in any real process availability is destroyed. Availability is destroyed when work is not obtained from a process in which there is a decrease of some potential differences, e.g., pressure, temperature, etc. The performance index used in summarizing Second Law or availability analyses is called effectiveness and at least three different definitions for effectiveness have been used in the literature and were calculated in the FEA study.

Effectiveness 1, which is believed to be the most useful definition, is defined as the ratio of the increase of availability to the decrease of availability for the process. The increase in availability for a process is the sum of the availability increases for those material streams flowing through the process which leave the process with a greater availability than they had when they entered the process. The decrease in availability for a

TABLE 1. EFFICIENCY AND EFFECTIVENESS SUMMARY FOR SELECTED INDUSTRIES AND UNIT OPERATIONS*

	PERCENT							
	DIRECT EQUIVALENT (3413 Btu/kilowatt-hour)				FUEL EQUIVALENT (10,400 Btu/kilowatt-hour)			
Industry/Unit Operation	Efficiency	Effectiveness 1	Effectiveness 2	Effectiveness 3	Efficiency	Effectiveness 1	Effectiveness 2	Effectiveness 3
Aluminum Ingot Production	29.6	31.5	32.6	30.4	14.2	14.4	15.1	14.6
Bayer Process (including calciner)	12.3	23.2	36.0	25.3	12.3	23.2	36.0	25.3
Hall Process	44.0	46.0	52.0	53.0	16.8	16.2	19.9	20.0
Copper Ingot Production	3.5	--**	9.2	9.1	2.7	--	7.1	7.1
Reverberatory Furnace	43.0	--	48.0	47.0	43.0	--	48.0	47.0
Copper Converter	45.0	--	47.7	50.4	45.0	--	47.7	50.4
Copper Refining	46.0	--	53.0	47.9	35.0	--	39.0	36.0
Glass (Container) Production	23.9	8.2	22.1	20.6	21.3	7.1	19.5	18.3
Melting Furnace Plus Regenerator	33.5	11.8	25.7	23.4	31.7	11.1	24.2	22.2
Steel Ingot Production	42.0	29.0	40.1	41.5	41.0	28.0	39.0	40.0
By-Product Coke Making	95.6	47.8	92.0	94.9	95.6	47.8	92.0	94.9
Blast Furnace	70.9	64.6	75.9	76.2	67.3	60.0	72.2	72.5
Basic Oxygen Furnace	89.4	--	86.6	84.1	87.3	--	85.0	82.0
Electric-Arc Furnace	95.2	26.7	92.3	90.4	83.0	11.0	80.0	79.0
Open Hearth Furnace	85.3	--	81.1	78.2	84.3	--	80.0	77.3

* Final report on "Evaluation of the Theoretical Potential for Energy Conservation in Seven Basic Industries", prepared under Contract No. 14-01-0001-1880, for Federal Energy Administration, by Battelle's Columbus Laboratories, July 11, 1975.

** A dash indicates that Effectiveness 1 values cannot be calculated as there is no increase in availability in the particular operation.

process is, similarly, the sum of all the availability decreases for those material streams flowing through the process which leave with less availability than when they entered. One of the important advantages of the Effectiveness 1 definition over other definitions of effectiveness or efficiency arises from the fact that Effectiveness 1 is based on increases and decreases in availability, rather than on absolute values of input and output availability as are the other definitions. This means that Effectiveness 1 values are not affected by the energy or availability of process feedstocks. In cases where the energy and availability of the feedstocks are large compared with that of purchased fuels, the efficiency and the effectiveness give high values because of the large energy and availability input and output of the feedstock and product. The only disadvantage to the use of Effectiveness 1 is that in some processes there is no increase in availability and the calculations cannot be made. In these cases the use of the Effectiveness 2 definition is recommended with the warning that, in processes having significant availability input with the feedstock, a high value of Effectiveness 2 is to be expected. This high value should not be interpreted to necessarily mean that little room for process improvement exists. The fact that Effectiveness 1 cannot be calculated indicates that the process could, theoretically, be self-sufficient.

Effectiveness 2 is defined as the ratio of the total output availability of the process to the total input availability. It is similar to the definition of efficiency in that total input is used, but significantly different in that the total output is used in the numerator, not just the useful output. The total availability output is used in the numerator because this indicates the maximum work which theoretically can still be done by the materials leaving the process.

Effectiveness 3 is a mixed First and Second-Law index of process performance; it is defined as the ratio of total availability output to total energy input and thus is similar in some respects to efficiency and Effectiveness 2. Effectiveness 3 involves the comparison of availability and energy, two similar but not identical thermodynamic properties having the same units.

In many analyses the values of Effectivenesses 2 and 3 will be quite close, primarily because for hydrocarbon fuels, the energy and availability are nearly the same. However, for many materials the energy and availability are quite different and when these materials are significantly involved in a process the use of Effectiveness 3 value can be quite misleading. One common material in this category is steam which is used extensively as a heat transfer medium and working fluid in many industrial processes.

The data presented in Table 1 can be used to rate and compare the extent to which energy is utilized in the four industries and in various unit operations within these industries.

As indicated in Table 1 two sets of efficiency and effectiveness values were calculated; one set (Column 1) was based upon an electrical to thermal energy conversion factor of 3413 Btu per kilowatt-hour (or the theoretical conversion factor); the other (Column 2) was based on 10,400 Btu per kilowatt-hour. The latter value reflects the generation and transmission losses associated with the industry-wide production of electrical energy and is comparable to the 10,500 Btu per kilowatt-hour value which has been used throughout previous reports prepared on this Bureau of Mines contract.

It should be noted that the efficiency and effectiveness values presented in Table 1 are based on industry "average" values and may not reflect operating conditions and/or the efficiency of operations within specific plants or facilities. These data do, however, provide a basis for identifying unit processes and technologies which are less efficient than other unit operations and might be replaced with alternative technologies in order to conserve energy.

The efficiency and effectiveness data presented in Table 1 were used in evaluating the need for more efficient unit operations in the production of these four basic commodities.

SUMMARY OF RESULTS

In general, it was found that the greatest need for considering replacement and/or alternative unit operations or total processes was associated with the production of the metallic commodities (e.g., aluminum, copper, steel, zinc, lead, etc) since these materials usually require large amounts of energy per net ton of product and their production involves multiple step-multiple reaction processes. Also, in some cases, their production is currently based on relatively inefficient unit operations. This should not be taken to imply that the current processes for yielding these commodities are not the most economical and/or the preferred practices in terms of available raw materials, capital intensiveness and/or materials consumption.

Table 2 summarizes replacement or alternative unit processes which are envisioned as potential means for conserving energy relative to current practices for each of the 14 high-priority industrial commodities. References associated with the particular commodity write-up are presented at the end of each of the commodity sections.

TABLE 2. NEED AND PRINCIPAL OPPORTUNITIES FOR ENERGY SAVINGS VIA PROCESS ALTERNATIVES IN THE PRODUCTION OF HIGH-PRIORITY MATERIALS

Commodity	Need for Replacement Technology	Possible Energy Saving Alternative Processes	Estimated Energy Saving via Replacement of Phase 4 Conventional Processes with Alternative Processes
Aluminum	Highly Desirable	Alcoa Smelting Process to replace Hall Process	25%(i.e. 182×10^6 relative to 244×10^6 Btu/ton)
Calcium (lime)	Low	None Envisioned	—
Cement(Portland)	Low	None Envisioned	—
Ceramics (common brick)	Low	None Envisioned	—
Chlorine (gaseous)	Desirable	None Envisioned	—
Copper	Highly Desirable	(1)Flash Smelting to replace Reverberatory Furnace	10%(i.e., 100×10^6 relative to 112×10^6 Btu/ton)
		(2)Continuous Smelting(Combining Reverberatory-Converter Practices)	
		(3)Blast Furnace to replace Reverberatory Smelting	Each estimated to save about 10%.
		(4)Bureau of Mines Autogenous Smelting	
Glass (containers)	Desirable	None Envisioned	—
Iron and Steel (slabs)	Desireable	(1)Continuous Steelmaking	~5%
Lead	Desirable	(1)Single Step Smelting of Galena	30%(i.e., 19×10^6 relative to 27×10^6 Btu/ton)
		(2)Imperial Smelting Furnace Process	Not Known
Nitrogen	Low	None Envisioned	—
Phosphorus (elemental)	Desirable	None Envisioned	—
Refractories (basic brick)	Low	None Envisioned	—
Sulfur	Low	None Envisioned	—
Zinc	Desirable	(1)Imperial Smelting Furnace Process	~7%(i.e., 60.5×10^6 relative to 65×10^6 Btu/ton)
		(2)Jarosite Processing	5-8%(i.e., $61.5\text{-}59.5\times10^6$ relative to 65×10^6Btu/ton

ALUMINUM

As presented in the Phase 4 Interim Report, the energy used by the U.S. aluminum industry to produce one net ton of aluminum ingot is 244×10^6 Btu. The apparent consumption of aluminum ingot in the U.S. in 1973 was 5.77×10^6 net tons, which implies a total annual energy requirement of about $1,408 \times 10^{12}$ Btu. Aluminum has the highest energy requirement per net ton of primary product and the second highest total annual energy requirement among the 14 commodities considered in the Phase 4 Interim Report.

Based on an FEA report[(1)] the theoretical minimum energy required to convert bauxite to aluminum ingot is 28.3×10^6 Btu per net ton. As indicated in Table 1 in the Introduction, the efficiency (i.e., useful output energy/total input energy x 100) associated with U.S. aluminum ingot production is slightly less than 30 percent on the basis of 3413 Btu per kilowatt-hour or 14.2 percent if the electrical input is taken at 10,400 Btu per kilowatt-hour. The effectiveness values associated with aluminum ingot production are essentially equal to the efficiency values.

As also indicated in Table 1, the efficiencies of the major energy-consuming unit processes for producing aluminum (i.e., the Bayer Process and the Hall Process) are less than those of many other major unit operations in the steel, copper, and glass industries.* Based on the energy requirement and the efficiencies of the major unit operations for aluminum production, it is considered highly desirable to develop energy-saving, replacement technologies (where feasible) for the manufacture of aluminum.

As presented in the Phase 4 Interim Report, the major energy-consuming steps in the production of aluminum are (1) the Bayer Process, wherein alumina is extracted from bauxite and (2) the Hall Process, wherein a purified alumina is electrolytically reduced to yield aluminum. Numerous attempts have been made to develop alternative processes to the Bayer-Hall Processes.

Alternative processes for the production of aluminum have been reviewed and described in a survey by Ing and Zeerleder[(2)] and in a recent paper by Peacey and Davenport[(3)]. These alternative processes typically fall within three categories: (1) replacement processes for the Bayer Process

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*The effectiveness values of the Bayer and Hall Processes are comparable to those of the major unit operations in the copper and glass industries and generally less than those of the major unit operations in the steel industry.

(i.e., alternative processes for producing an adequate-purity alumina for input material to the Hall Process), (2) replacement processes for the Hall Process which use Bayer-process alumina as the primary input material, and (3) processes which replace both the Bayer and Hall Processes. Alternative processes of these three types are discussed below.

Alternative Processes to the Bayer Process

Of the aluminum-bearing minerals, bauxite, which is the major input to the Bayer Process, is the richest in aluminum. Untreated bauxite typically contains 55 to 65 percent alumina (Al_2O_3) and, in turn, about 4 pounds of bauxite are processed per pound of aluminum produced. Typically 8 to 9 pounds of clay or kaolin are required per pound of aluminum. The use of a leaner ore than bauxite implies that more ore must be treated and, in turn, frequently more energy is required to produce a pure alumina. However, because bauxite suitable for the Bayer Process is not as accessible as kaolin or clays, numerous attempts have been made to develop alternative processes.

As described in Reference 2, replacement processes to the Bayer Process are of two general types: (1) wet methods and (2) electrothermal methods. A major problem with both types of processes relative to the Bayer Process is purity of the end-product alumina. Many of the methods yield a less-pure alumina than the Bayer Process and are, therefore, currently unacceptable alternatives. Detailed analyses of the energy requirements of the Bayer replacement wet methods are not available. However, the wet methods do not appear to offer any major energy saving relative to the Bayer Process.

Several electrothermal processes have been developed for producing alumina. Probably the best-known electrothermal process is the Pedersen Process, wherein bauxite, coke, limestone, and iron ore are melted in an electric-arc furnace to produce pig iron and a calcium aluminate slag. The slag, typically containing 30 to 50 percent Al_2O_3 and 5 to 10 percent SiO_2, is treated with caustic soda to produce sodium aluminate which, in turn, is treated to produce alumina. The energy assigned the Pedersen Process alumina will depend on the credits assigned the pig iron and furnace gases produced in the electric-arc furnace as the calcium aluminate slag is essentially a coproduct. The overall energy requirements for alumina produced via the Pedersen Process, or the other electrothermal methods are generally greater than the Bayer Process energy requirement if the electrical energy input to the furnace is taken at 10,400 Btu per kilowatt-hour.

Thus, while the Bayer Process appears to be a relatively inefficient process (i.e., about 12 percent efficient on a useful energy-out energy-in basis)* alternative processes for producing relatively pure alumina do not appear to provide significant energy saving potential relative to the conventional Bayer Process.

Replacement Processes for the Hall Process

The Hall Process is the principal energy consuming unit operation in the production of aluminum ingot. The Hall Process uses 80.5 percent of the total energy requirement for aluminum production and is characterized by an efficiency (i.e., useful energy output/total energy input x 100) of 44 percent on the basis of 3413 Btu per kilowatt-hour or slightly less than 17 percent on the basis of 10,400 Btu per kilowatt-hour. Because of these characteristics, the aluminum industry has attempted to conceive and develop energy conserving processes to replace the Hall Process.

One method of producing aluminum from Bayer alumina which has an energy saving potential relative to the Hall Process is the Alcoa Smelting Process. In the Alcoa Process, Bayer alumina is chlorinated to produce a gaseous mixture of aluminum trichloride ($AlCl_3$), carbon dioxide, and carbon monoxide. The aluminum trichloride is separated from the gases by condensation in a fluidized bed of solid $AlCl_3$ particles. The solid $AlCl_3$ is continuously introduced into a DC electrolytic cell containing a fused chloride electrolyte at about 1290 F (700 C). Liquid aluminum metal is produced at the cathode and gaseous chlorine, which is recycled to the chlorination plant, is formed at the anode.

A simplified flowsheet for the Alcoa Smelting Process is shown in Figure 1. Alcoa has utilized 2 electrolytic cell designs, (1) a monopolar cell similar to the conventional Hall cell and (2) a bipolar cell containing 4 bipolar electrodes. Both cell arrangements are stated to offer reduced power consumption

* The Bayer Process has an associated Effectiveness 1 value of about 23 percent

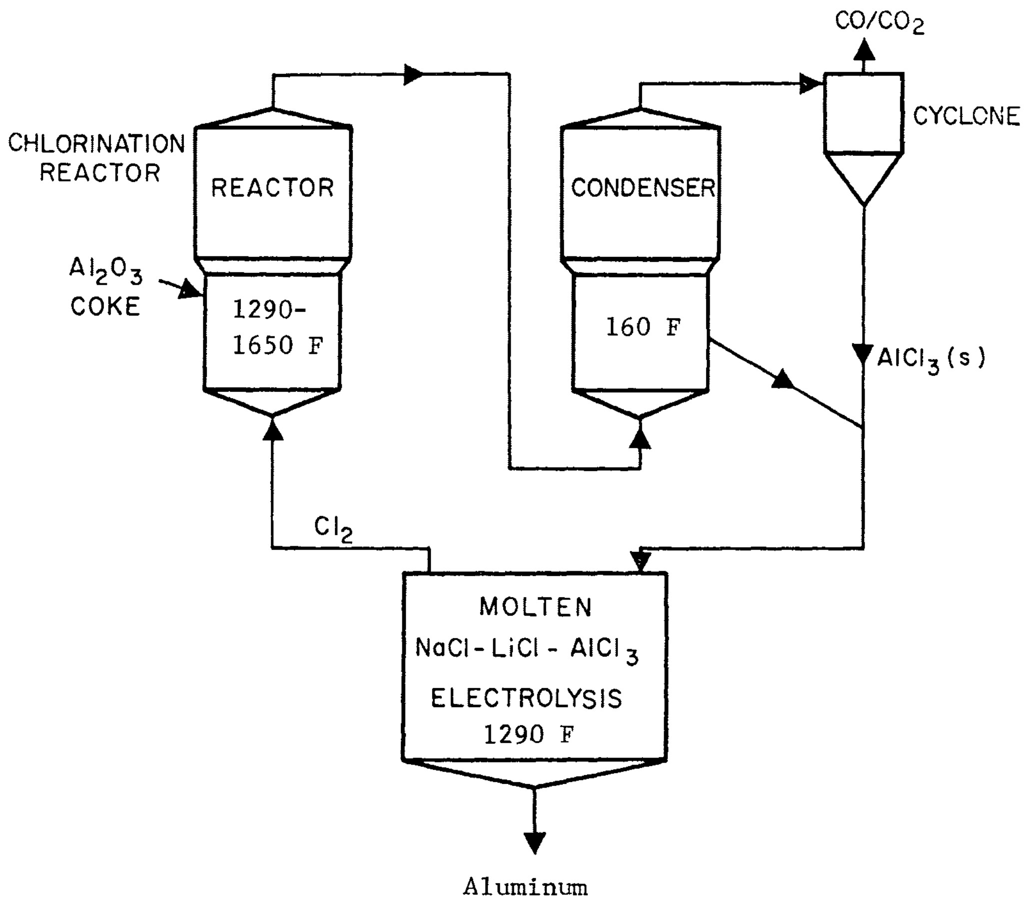

FIGURE 1. SCHEMATIC FLOWSHEET OF THE ALCOA SMELTING PROCESS(3)

relative to the standard Hall cells because of (1) greater electrical conductivity of the chloride electroyte (i.e., 4 ohm^{-1} cm^{-1}) compared to that of cryolite (i.e., 2.8 ohm^{-1} cm^{-1}) and (2) smaller interpolar separations used in the Alcoa cells relative to the conventional Hall cell. Peacey and Davenport(3) estimate a 20 percent reduction in electrical energy requirement for the Alcoa Smelting Process relative to the best Bayer-Hall system.* As the more efficient U.S. aluminum producing plants use about 6.5 kilowatt-hours per pound of aluminum in the Hall cell,** a 20 percent reduction (i.e., a reduction of 1.3 kilowatt-hours per pound) would result in a specific energy consumption of 5.2 kilowatt-hours per pound of aluminum. Based on the above value of 5.2 kilowatt-hours per pound of aluminum cited for the Alcoa Process, the energy required per net ton of aluminum would be about 185×10^6 Btu per ton. Peacey and Davenport indicate that the Alcoa Smelting Process would consume about 20 percent less carbon than the best Hall cell operation. Peacey-Davenport also estimate that a 100,000-metric-ton-per-year Alcoa Smelting Process plant requires a 5 percent less fixed capital investment cost (dollars per ton of aluminum) and a 10 percent greater direct operating cost than a comparable size Bayer-Hall system.

The development of the Alcoa Smelting Process is currently underway and it appears that this alternative process will provide a significant energy savings relative to the Hall Process.

Alternative Processes to Bayer-Hall Processes

Several processes to produce aluminum have been proposed which would replace both the Bayer and Hall Processes. Of the methods considered, two processes may provide an energy saving relative to the traditional aluminum production route. These are (1) electrolysis of aluminum sulfide and (2) the reduction of aluminum trichloride with manganese (i.e., the Toth Process). Electrolysis of aluminum sulfide rather than alumina would provide several advantages, from an energy point of view, over the Bayer-Hall Processes.

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*Alcoa has stated that the Alcoa Smelting Process could reduce energy requirements by 30 percent relative to the best Hall cell operation.

**The U.S. industry average is about 8 kilowatt-hours per pound of aluminum which is the value used in arriving at a production energy requirement in the Phase 4 report.

These advantages include (1) electrolysis at a lower bath temperature (i.e., 1560 F rather than 1740 F in a Hall cell), (2) a greater solubility of the aluminum sulfide in the electrolyte than is possible with alumina in cryolite, and (3) a lower consumption of anode material because depolarization of oxygen and its subsequent reaction with the anode would be eliminated. No estimate is available for the potential energy saving with this process. There are, however, some technical problems with the preparation of pure aluminum sulfide and the purity of the resulting aluminum.

The Toth Process, which has received much publicity recently, is based on the exchange reaction between manganese metal and aluminum trichloride to give manganese dichloride and aluminum metal, as represented by the equation:

$$3Mn + 4Al_2Cl_6 = 3(MnCl_2 \cdot Al_2Cl_6) + 2Al. \qquad (1)$$

This reaction is carried out at 572 F and 15 atmospheres pressure. A simplified flowsheet for the Toth Process is shown in Figure 2. As shown, calcined clay and coke are chlorinated at 1700 F with a mixture of chlorine and silicon tetrachloride. The volatile chlorides are condensed from the carbon monoxide to yield a liquid Al_2Cl_6 product. The liquid aluminum chloride (Al_2Cl_6) reacts with manganese metal in the aluminum generator to produce a metallic aluminum and a manganese aluminum chloride salt mixture. Aluminum chloride is separated from the fused salt by evaporation and subsequent condensation. The remaining solid manganese chloride is oxidized to produce manganese sesquioxide (Mn_2O_3) which is then reduced to manganese metal (ferromanganese) in a conventional manganese blast furnace.

Numerous researchers have questioned the technical and economic feasibility of the Toth Process. The use of manganese as the reductant does not appear economically feasible, and regeneration of the manganese in a blast furnace would significantly increase the coke requirements per ton of aluminum product*. A major question exists concerning the purity of the blast furnace regenerated manganese and its overall effectiveness in reducing aluminum chloride.

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*Three tons of metallic manganese are required per ton of aluminum. It is estimated that 5 tons of coke would be required per ton of aluminum using blast furnace regeneration of manganese.

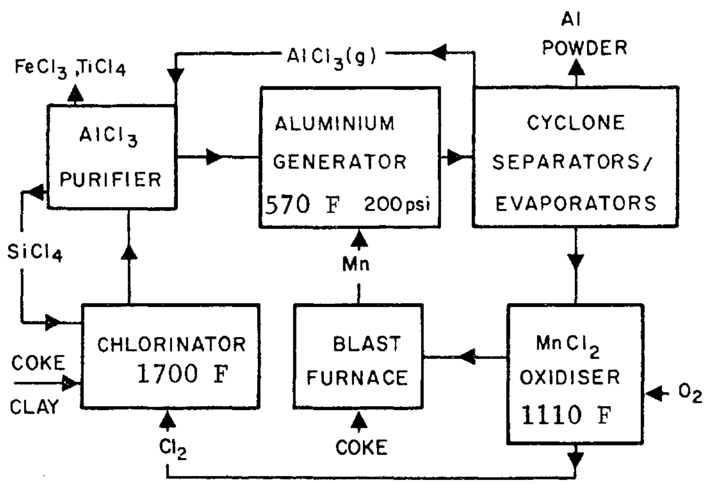

FIGURE 2. SCHEMATIC FLOWSHEET OF TOTH PROCESS(3)

While the Toth process would significantly reduce the electrical energy input for producing aluminum (Peacey and Davenport estimate the Toth process would require only about 1 kilowatt-hour per pound of aluminum), the associated operation of a manganese blast furnace and, in turn, the increased carbon requirement per ton of aluminum will probably make the energy requirements associated with the Toth Process greater than those of the conventional Bayer-Hall Processes. A comparison of the electrical energy and carbon energy equivalents for the Toth Process and other aluminum production processes is given in Table 3.

Other processes which are currently being considered to replace the Bayer-Hall Processes, such as the Alcan Process and the Monochloride Process, are not considered energy-conserving relative to the best Bayer-Hall technology available. The Alcan and Monochloride Processes are described in Reference 3. An evaluation and comparison of probable capital and operating costs, electric power, and carbon requirements of the Alcoa, Alcan, Monochloride, and Toth Processes with the best available Bayer-Hall technology as reported by Peacey and Davenport(3) is given in Table 3.

Conclusions - Aluminum

Because of the large energy requirement to produce a ton of aluminum and the relatively low efficiency of the principal unit operations (i.e., the Bayer Process and Hall Process), it is highly desirable to develop alternative energy-saving methods for the production of aluminum. Of the various alternative processes to the conventional Bayer-Hall Processes, the Alcoa Smelting Process appears particularly promising in terms of technical feasibility, and possible energy savings. In general, other processes which may allow an evergy savings are not currently acceptable to the industry because of reduced purity of the aluminum product and/or economic considerations.

TABLE 3. EVALUATION AND COMPARISON OF VARIOUS ALTERNATIVE PROCESSES WITH BAYER-HALL TECHNOLOGY[a] (3)

	Processes					
	Average Bayer-Hall[b]	Best Bayer-Hall	Alcoa[c]	Alcan	Monochloride[d]	Toth[d]
Fixed Capital Investment, $ Per net ton Al	-	1870	1775	1030	1120	1120
Direct Operating Costs, $ per net ton Al	-	385	424	385	327	577
Electrical Energy, kwhr per lb Al	8.0	6.35	5.08	11.58	6.86	0.98
Carbon Requirement (lb C per net ton Al)	890	1090	870	2395	1850	10,090
Sum of Electrical Energy and Carbon Energy Equivalent[e] (million Btu/ton Al)	190	160	128	308.5	193	270

(a) Based on a plant producing about 110,000 net tons of aluminum per year.
(b) Data taken from Phase 4 Report
(c) With bipolar cell and includes Bayer plant.
(d) Bauxite is taken as starting material.
(e) Carbon energy is taken as 22 x 10^6 Btu/890 lbs. carbon required/ton Al produced, consistent with Phase 4 report.

References - Aluminum

(1) Final Report on "Evaluation of the Theoretical Potential for Energy Conservation in Seven Basic Industries", Prepared by Battelle's Columbus Laboratories for the U.S. Federal Energy Administration (Contract No. 14-01-0001-1880) July, 1975.

(2) Ing and A. Zeerleder, "Attempts to Improve Aluminum Reduction Since Heroult and Hall", Journal of Institute of Metals, Vol. 83, 1954-55, pp 321-328.

(3) Peacey, J. and Davenport, W., "Evaluation of Alternative Methods of Aluminum Production", Journal of Metals, July, 1974, pp 24-28.

CALCIUM (LIME)

The average amount of energy used to produce one net ton of quicklime is 8.5×10^6 Btu. Because the 1973 U.S. consumption of lime was 21.4×10^6 net tons, the total required annual energy was about 182×10^{12} Btu. Calcining (i.e., the thermal decomposition of calcium carbonate, limestone) accounts for about 95 percent of the total energy requirement, about 8.1×10^6 Btu per net ton. The theoretical minimum energy required for decomposition of calcium carbonate to produce one net ton of lime is about 2.6×10^6 Btu.

The need for developing new replacement technology or processes as a means of saving energy in the production of lime is considered to be low for several reasons. First, the energy requirement to produce lime via the conventional route is relatively low in comparison to that for many other commodities. Second, it is anticipated that high-calcium limestone will remain the primary raw material for lime production. And third, the production of lime is essentially a one-step reaction involving only the heating of the limestone. Thus the need for developing energy-saving alternative processes is low and no new replacement processes are envisioned.

CEMENT

The average energy requirement per net ton of portland cement is 7.6×10^6 Btu. As about 90.5×10^6 net tons of cement were consumed in the U.S. in 1973, the total energy associated with the U.S. consumption of cement annually is about 688×10^{12} Btu. The major energy consuming operation in the production of cement is the kiln or clinker burning step, in which the raw materials (limestone, clay, shale, and sand) are heated to produce a clinker. The kiln or burning operation typically requires 6.1×10^6 Btu per ton of cement or about 80 percent of the total requirement.

Improvements in kiln design and operation are referred to as "new processes" in the Phase 8 Interim Report. While these improvements to cement production should result in significant energy saving relative to the older conventional practice, these are not new processes in the sense of the Phase 9 study. The improvements reported in Phase 8 do not significantly alter the basic flow diagram for cement production.

The need for developing energy conserving alternative processes for producing cement is judged to be low. This judgment is based upon the following data. First, the energy requirement per ton of cement is relatively low in comparison to that of the other high-priority commodities. Second, the current method for producing cement is a relatively simple, straightforward operation involving essentially a single heating of the raw materials mix. Also, no energy saving alternative processes are currently known to the BCL reviewers.

CERAMICS - COMMON BRICK

The production of common brick requires an average energy of 3.5×10^6 Btu per net ton. The 1973 U.S. consumption of common brick was 17.6×10^6 net tons with an associated total energy of 62×10^{12} Btu. The production of common brick is a relatively simple and straightforward operation wherein appropriate clays are mixed with water, extruded and fired. Firing and drying of the brick is the major energy consuming operation, requiring about 95 percent of the total energy consumed in making the brick. Because the energy requirement (Btu per ton) to make bricks is low relative to the other high-priority materials and because the conventional process is essentially a one-step heating operation, there does not appear to be a great need and/or potential for developing alternative energy-conserving processes for producing common brick. No alternative energy saving processes for producing common brick are envisioned.

CHLORINE

About 18 x 10^6 and 20.7 x 10^6 Btu are required to produce one net ton of gaseous and liquid chlorine, respectively. As the 1973 U.S. consumption of chlorine (both gas and liquid) was 10.3 x 10^6 net tons, the total annual energy required for the U.S. production was about 199 x 10^{12} Btu.

The major energy consuming step in the production of chlorine is the electrolytic diaphragm cell for electrolysis of the salt solution.* About 88 percent of the total energy required to produce chlorine is associated with the cell energy requirements. The efficiency of the electrolytic cell with metal anodes is estimated as 55 to 65 percent.

Because the energy requirement per ton of chlorine is relatively low and because it is difficult to envision alternative energy conserving processes to replace the electrolytic cells, the need for and likelihood of developing energy-saving alternative processes is judged to be low. There are, however, energy saving improvements to the cell operation which may be introduced to the U.S. industry.

As reported in the Phase 4 and Phase 8 Interim Reports, the major recent improvement in the production of chlorine, in terms of energy conservation, was the replacement of carbon by metal anodes in the electrolytic cells.

The introduction of metal anodes typically increased cell efficiency by 5 to 10 percent from a value of 50 to 60 percent with carbon anodes. The cathode in diaphragm cells has been and remains an iron wire screen, which does not influence cell efficiency significantly. The efficiency improvement with metal anodes results from the ability to maintain a constant gap between anode and cathode, stabilizing cell resistance at a minimum level.[(1)]

Other energy saving improvements to cell operation include the use of (1) bipolar diaphragm cells and (2) new improved ion-exchange membranes. Bipolar diaphragm cells, such as developed by PPG Industries, may reduce the cell electrical input to about 2575 kilowatt-hour (versus 3100 kwh in 1973) per net ton of chlorine produced.[(1)] Such cells use metal anodes and synthetic

*The electrolytic cell is also the major energy consuming operation in the mercury amalgam cell process for producing chlorine. Mercury cells require more energy (i.e., 3200-3600 kwhr per net ton Cl) than the diaphragm, cell systems (i.e., 2700-2900 kwhr per net ton Cl).

membranes because of the design of the cells. Improved ion-exchange membranes are primarily of interest because of their longer service life (estimated as two years) compared to the asbestos membranes which are normally renewed or replaced at intervals of about 100 days. At some slight penalty in electrical efficiency, these improved ion-exchange membranes, such as the DuPont Nafion-family membranes, are expected to produce a better quality (i.e., lower salt content) caustic soda at significantly higher concentrations (e.g., over 25 percent as compared to the 10-15 percent from asbestos diaphragms). Although this has minimal impact on electrical input to the cells, less subsequent energy will be needed to concentrate the cell liquor to commercial (50 percent) caustic soda.[3] Metal anodes and ion-exchange membranes should reduce diaphragm cell electrical requirements to an average of 2550-2650 kwh AC versus the 3100 kwh per ton of chlorine given in the Phase 4 report.

Conclusions - Chlorine

The need to develop energy-conserving alternative processes for producing chlorine is judged to be low and no energy saving replacement technologies are known for the large-scale production of chlorine. Continuing improvements to the existing diaphram-cell operation should lower the energy required to produce chlorine and caustic.

References - Chlorine

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COPPER - CEMENT COPPER

The energy required to produce one net ton of cement copper by the dump leach process is 87×10^6 Btu. As about 0.17×10^6 net tons of cement copper (about 10 percent of total mine output of copper) was processed in the U. S. in 1973; thus, the total energy associated with the 1973 U. S. consumption was 14.9×10^{12} Btu.

As indicated in the Phase 8 Interim Report, little can be done to change the current dump leaching process to save energy. This fact, together with the observation that this particular commodity does not consume a large amount of energy on a total annual basis relative to the other high-priority commodities is taken to imply that the need for developing energy-saving alternative processes for yielding cement copper is low.

COPPER - REFINED COPPER

The energy required to produce one net ton of refined copper is about 112×10^6 Btu. The 1973 U. S. consumption of refined copper was slightly less than 2×10^6 net tons, which implies that the total annual energy associated with this commodity was 221×10^{12} Btu. Of the high-priority materials considered in the report, refined copper has the third largest energy requirement per net ton of product and the sixth largest energy on a total annual production basis.

The energy of copper sulfide ore is greater than the energy of copper; thus, theoretically the production of copper should not require any energy (i.e., fuel) input.[(1)] This fact, together with the observation that today's copper ores are relatively lean with respect to copper content (typically containing 0.7 percent copper), accounts for the relatively low efficiency (i.e. 3.7 percent) and effectiveness values (i.e., less than 10 percent) indicated in Table 1* for the production of copper ingots. As indicated in Table 1, the principal unit operations associated with the production of copper (i.e., reverberatory furnace, converter and electrolytic refining) appear relatively efficient in comparison with unit operations in other industries. The total energy required to produce a net ton of refined copper is distributed as follows: 57 percent (i.e., about 64×10^6 Btu per ton of Cu) for mining, transportation and concentration of the ore; about 34 percent for smelting (i.e., reverberatory furnace plus converter) the concentrate, and 9 percent for electrolytically refining the copper. The reverberatory furnace uses the largest quantity of energy in the smelting operation, requiring about 20 percent of the total energy used for the production of refined copper.

Based on (1) the energy requirements to produce one net ton of refined copper and the total annual energy associated with the U. S. consumption of refined copper (2) the complexity of the conventional production processes (i.e., the number of required unit operations and/or chemical reactions associated with refined copper production) and (3) the availability of known alternative energy-saving production processes for producing refined copper, it is considered highly desirable to develop energy saving replacement technologies (where feasible) for the manufacture of copper.

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* Table 1 appears on page 5.

Methods and/or procedures which may allow energy conservation in the production of copper were described in the Phase 8 Interim Report. While mining and concentration of the copper ores accounts for the major portion of the energy required for production of copper, no new alternative energy-saving practices are envisioned to those currently used and/or described in the Phase 4 and 8 Interim Reports.

The envisioned energy-saving replacement and/or alternative processes are generally directed toward replacement of the smelting steps. Several operations which offer an opportunity to save energy relative to the conventional smelting procedures include (1) Outokumpu-type Flash Smelting, (2) Continuous Smelting, such as the Noranda Process, the WORCRA Process, the Q-S Process and the Mitsubishi Process, (3) Autogenous Smelting and (4) Blast Furnace Smelting. Each of these replacement methods is discussed below.

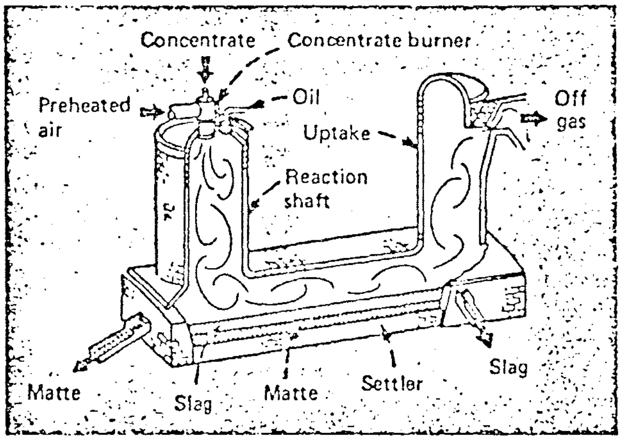
Flash Smelting

The type of flash smelting developed by the Outukumpu smelter in Finland and a variation developed by International Nickel Company in Canada provide a significant fuel saving in comparison with reverberatory smelting. A schematic illustration of the Outokumpu Flash Smelter is shown in Figure 3. Flotation concentrates, with flux and preheated ore, are injected into a hot chamber where the flash burning of sulfides in suspension furnishes the energy needed for smelting. This reduces the need for auxiliary fuel input and also reduces the volume of the combination products. A 8 to 14 percent SO_2 offgas is produced with the Outokumpu Flash Smelter which is used to produce sulfuric acid.

The International Nickel variation uses 95 percent oxygen in place of air and a more concentrated (i.e., 70 to 80 percent) SO_2 gas is produced.

Kellogg has indicated that the Outokumpu-type flash smelter requires only about 40 percent of the energy used in a typical green charge reverberatory furnace[(5)]. The Inco-oxygen flash smelter is anticipated to provide even greater energy saving relative to the conventional reverberatory smelter.

Besides improved utilization of energy and diminished gas volume, the flash smelting practices are considered to yield a higher grade matte than reverberatory smelting. However, high copper content slags are produced in flash



OUTOKUMPU FURNACE produces 14% SO_2 offgas, an ideal gas grade for sulfuric acid plants

FIGURE 3. SCHEMATIC ILLUSTRATION OF OUTOKUMPU FLASH-SMELTING UNIT

smelters and additional energy would be required to reclaim the copper from the high-copper slag.

Continuous Smelting

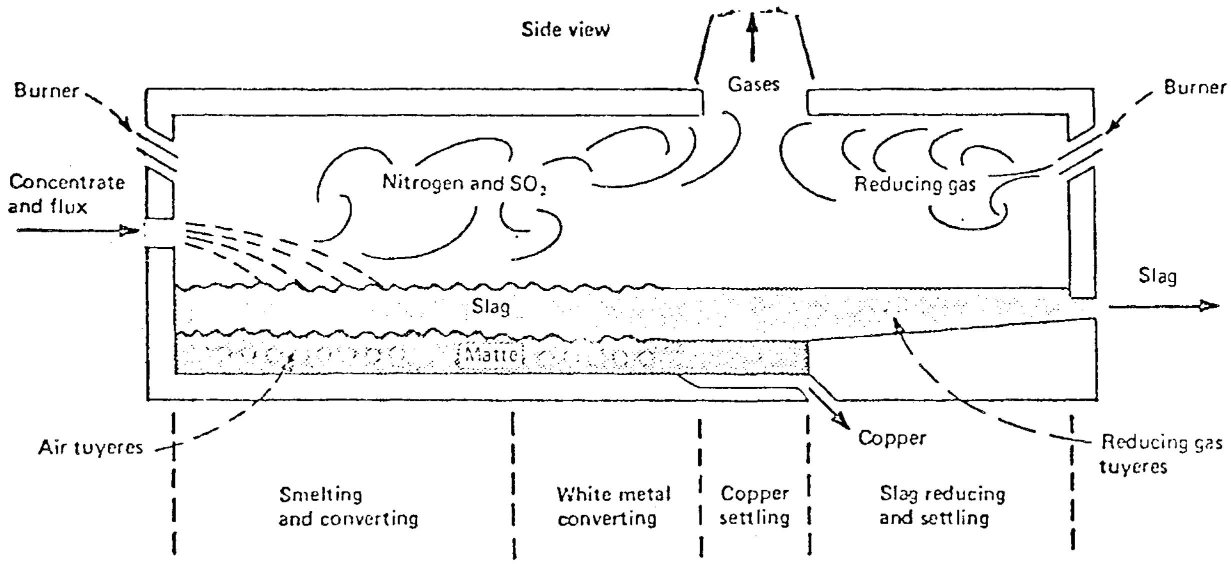
Combining the conventional reverberatory-converter operation into a single unit should provide an energy saving and has received reasonable attention from a development point of view, over the last few years. A number of processes, which are known as continuous smelting processes have been developed to the pilot plant stage and a few to the semicommercial stage. Several of the continuous smelting processes are described below.

The Noranda Process[6,7]

Developed by Noranda Mines, Ltd., in Canada, the process uses a single long (about 70 feet) combined smelting-converting unit. A schematic illustration of the Noranda process is shown in Figure 4. The unit consists of a horizontal, cylindrical furnace having a central depressed area for copper collection and a round hearth at one end for slag removal. A burner heats the smelting end where concentrates and flux are charged. Air or an air-oxygen mix is introduced through tuyeres along the base of the furnace to oxidize the matte that is formed. Thus, there is only one furnace to control and transfer of molten material from one furnace to another is eliminated. The exit gas is of sufficient SO_2 content for the subsequent production of sulfuric acid. After testing a 100-ton-per-day pilot operation, Noranda has built a 800 ton-per-day commercial operation. Kennecott has also announced plans to install a Noranda-type continuous smelter. While detailed energy requirements are not known for the Noranda-type continuous smelter, this process should allow an energy saving over the conventional smelting process.

The WORCRA Process[6,8]

The WORCRA continuous smelting process has undergone pilot testing over the last few years. This process combines smelting, converting, and slag cleaning in one operation within a long stationary furnace. Molten copper is removed from one end of the unit, slag is removed at the other end, and offgases are removed as a single stream for subsequent processing. In place of the tuyeres



NORANDA PROCESS produces rich off-gases to facilitate SO_2 recovery.

FIGURE 4. SCHEMATIC ILLUSTRATION OF NORANDA PROCESS

used in the Noranda process, air lances introduced from the top of the furnace are used for converting in the WORCRA process.

Through utilization of the exothermic reaction on burning the sulfides, the energy requirement for the WORCRA-type unit would be expected to be less than with the conventional processes; however, an analysis by Kellogg of the operating data for the WORCRA furnace at the Port Kembla pilot plant did not indicate a significant energy saving relative to conventional reverberatory smelting.(5)

The low value of the Port Kembla unit specific capacity (i.e., tons smelted per hour per unit of furnace volume) and a fuel consumption comparable to conventional reverberatory smelting were mainly attributed to a larger-than-necessary slag settling basin, wherein the entrained matte is separated from the slag for return to the smelting zone of the furnace. Because of the relatively large size of the slag cleaning branch, it was suggested that more fuel was consumed relative to that required in a properly proportioned and/or designed furnace system. Modification to the design and operation of the WORCRA-type continuous smelter could lead to improved energy utilization and then could make this process favorable (from an energy point of view) relative to the conventional smelting practices.

The Q-S Process

Named after the inventors Paul Queneau and Reinhardt Schumann, the Q-S process is also a multistage progressive converter operation that combines continuous smelting and converting in one furnace. Sulfide concentrates are flash smelted with oxygen and oxygen is also introduced through submerged tuyeres to effect production of copper. A slag scavenging operation is also part of the process. Pilot plant investigations of the Q-S process are currently in the planning stage. No estimate of the energy requirements of the Q-S process are currently available.

The Mitsubishi Process[6,10-12]

Developed by Mitsubishi Metals Corporation of Japan, this process differs from the Noranda and WORCRA continuous processes in that all of the processing is not done in a single unit. Rather, the concentrates are smelted in one furnace and the slag and matte flow continuously through a slag cleaning furnace to a converting furnace equipped with overhead air lances. A schematic illustration of the Mitsubishi Process is shown in Figure 5. The continuous process uses countercurrent flow as matte flows from the smelter furnace to the converter, while the converter slag is returned to the smelter. The continuous stream of SO_2 from both furnaces should be suitable for acid or sulfur recovery.

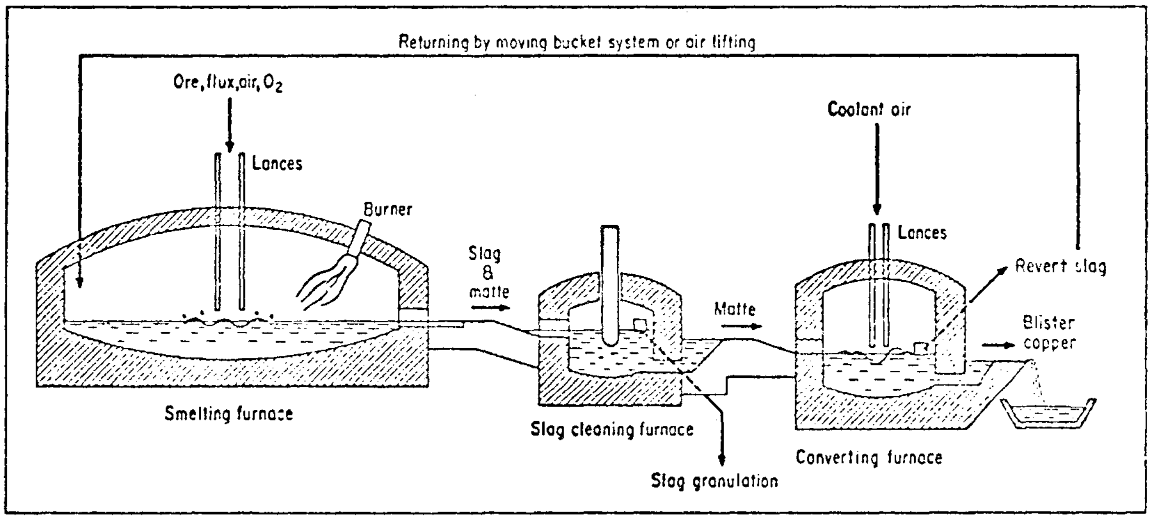


FIGURE 5. SCHEMATIC ILLUSTRATION OF MITSUBISHI PROCESS

The Mitsubishi Process has been scaled to a 1,500-ton blister copper per month semicommercial plant which started operation in November, 1971. Steady-state operation and control of the Mitsubishi Process is achieved by on-line, feed-forward computer control of the inputs of air and flux to the converter furnace. Assuming that long-term, steady-state operating conditions can be maintained, the Mitsubishi Process may make possible significant reductions in energy consumption relative to the conventional smelting practices.

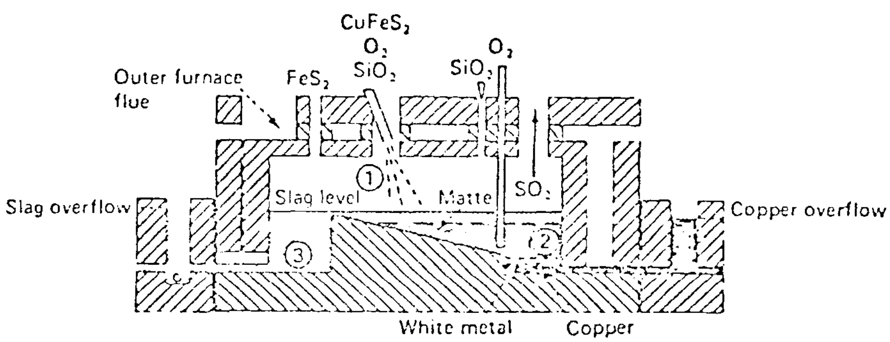
Autogenous Smelting

Autogenous smelting work has been conducted on a laboratory scale by the United States Bureau of Mines[13]. Like the Noranda and WORCRA processes, the Bureau of Mines autogenous smelting is a continuous smelting method in a single unit to produce copper directly from concentrate. The furnace combines flash smelting with converting by means of an oxygen lance immersed through the slag into the matte. A schematic illustration of the Bureau of Mines Autogenous Smelting Unit is shown in Figure 6. While this process appears to offer potential for energy saving relative to the conventional smelting processes, no further development of the process appears to be underway.

The Momoda Blast Furnace[2]

The Momoda Blast Furnace which was developed by Sumitomo Metal Mining Company, is currently used by two copper smelters in Japan. In this process concentrates are charged to the blast furnace with other copper-bearing materials as a stiff plasticized mass containing 10 to 15 percent water. The energy requirement for smelting a ton of charge in a Momoda blast furnace is given as only about 28 percent of the energy required for reverberatory smelting with a wet charge; thus there appears to be a substantial energy saving. While the blast furnace smelting of wet concentrate may provide a lower energy route to copper production than the conventional smelting processes, a detailed analysis of the blast-furnace operation is required before the magnitude of the energy saving is known.

Copper may also be produced by hydrometallurgical processes, such as Anaconda's "Arbiter" Ammonia Leach Process[14], the Sherritt-Gordon Process, the



Verticle elevation section through center

FIGURE 6. SCHEMATIC ILLUSTRATION OF U. S. BUREAU OF MINES AUTOGENOUS SMELTING UNIT

Cymet Process[15] and the Duval Corporation Process[16]. While there are few precise data available on the fuel and power requirements for these hydrometallurgical processes, they do not appear to offer any potential for energy conservation relative to conventional smelting processes. Kellogg has estimated that the hydrometallurgical processes for producing copper require at least twice the energy associated with the conventional pyrometallurgical smelting processes[5]

Conclusions - Refined Copper

Based on the energy required per ton of refined copper and the total annual energy associated with this commodity, it is highly desirable to develop alternative energy-saving methods for the production of copper. In general, various pyrometallurgical smelting processes, including flash smelting, continuous smelting and blast furnace smelting, may provide an energy saving relative to the conventional melting processes.

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(16) "Duval Claims Development of Pollution Free Hydrometallurgical Copper Refining Process", Engineering and Mining Journal, September, 1970, p. 171.

GLASS

The production of one net ton of glass containers requires about 17.4 x 10^6 Btu. Combining this figure with the 1973 U. S. consumption of about 12.4 x 10^6 net tons of glass containers results in a total annual energy associated with this commodity of 216 x 10^{12} Btu.

The theoretical minimum energy required to produce glass for containers is 2.15 x 10^6 Btu per net ton of glass melted[(1)]. The overall efficiency (i.e., the energy of the product glass divided by the total energy input x 100) is approximately 23.9 percent, as indicated in Table 1 in the Introduction*. The Effectiveness 1 value for the production of glass is 8.2 percent based on a 3413 Btu per kilowatt-hour conversion factor and 7.1 percent based on a 10,400 Btu per kilowatt-hour conversion factor.

The glass-melting operation is the most energy intensive step in production of the glass containers. About 47 percent (i.e., 8.1 x 10^6 Btu) of the 17.4 x 10^6 Btu per ton of glass is associated with the melting operation. With current levels of furnace insulation, regeneration and combustion air preheating, the melting furnaces have an estimated efficiency of 33.5 percent**.

As indicated in Table 1 in the Introduction, the efficiency and effectiveness values for the overall glass production processes and the major unit process (i.e., the melting step) are generally less than prevail with the other industries and unit operations considered.

While the overall production of glass containers may be judged to be relatively inefficient from an energy usage point of view, the energy required to produce one net ton of glass containers and the total annual energy associated with the U. S. consumption of this commodity are intermediate values within the high-priority group of materials. Also, the current procedures for producing glass containers are a relatively direct sequence of steps which appear to be the most simplistic route considering raw materials, economics, etc.

In light of the above factors and considering that no energy-conserving alternative processes for producing glass containers were known to the BCL reviewers, the need for developing energy-saving alternative production methods for this commodity was taken to be between low and desirable.

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* See Page 5.

** See Table 1, page 5, for other efficiency and effectiveness statements for glass melting furnaces.

While no new energy-saving alternative and/or replacement procedures are envisioned at this time, various improvements (in terms of energy usage) to the conventional glass-making technology are described in the Phase 8 Interim Report[2]. Improvements to the glass-making operations, particularly the melting operation, include submerged combustion and oxygen-enrichment. These variations to the conventional operation may significantly reduce the energy requirements for this commodity.

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IRON AND STEEL - CARBON STEEL CASTINGS

The energy required to produce one net ton of carbon steel castings is 42.6×10^6 Btu. It is estimated that the production of alloy steel castings requires about 20 percent more energy (i.e., a total of 50×10^6 Btu) per net ton than carbon steel castings because of increased inputs of alloying elements, longer refining times, and more critical specifications. The 1973 U. S. consumption of carbon steel and alloy steel castings was about 1.28×10^6 and 0.62×10^6 net tons, respectively. Thus, the total energy required that year for U. S. consumption of carbon steel castings and alloy steel castings was 54 and 31×10^{12} Btu respectively, or a total energy of 85×10^{12} Btu for all steel castings.

As indicated in the Phase 4 and 8 Interim Reports, the distribution of energy to produce carbon and alloy steel castings is about 55 percent for electric-arc furnace melting, including input materials, 15 percent for heat treatment, 10 percent for molding and coremaking, 10 percent for space heating, 4 percent for pollution control and 6 percent for other requirements. The potential for energy conservation in the manufacture of steel castings is lower than for gray iron castings (see succeeding section) because (1) there is a significantly smaller production of steel castings than iron castings and (2) there are fewer opportunities for modification of the process flowsheet for energy conservation. As with gray iron castings, the basic processes for producing carbon steel castings consist of melting and casting carbon (or alloy) steel in prepared molds.

Because (1) the conventional method for preparing carbon and alloy steel castings is direct and relatively efficient and (2) no alternative energy-saving processes are envisioned, the need for and probability of developing alternative energy-conserving processes for this commodity is judged to be low.

IRON AND STEEL - GRAY IRON CASTINGS

The production of one net ton of gray iron (including ductile or nodular iron) castings requires about 34×10^6 Btu. In 1973, slightly less than 10.8×10^6 net tons of iron castings were consumed implying an annual energy requirement of 366×10^{12} Btu for this commodity.

As indicated in the Phase 4 and Phase 8 Interim Reports, the major portion of the energy associated with iron castings is consumed in the melting operation. Approximately 61 percent (i.e., 20.7×10^6 Btu) of the energy required to produce gray iron castings is assigned to the cupola furnace operation*, including input materials. Of the remaining energy requirements, 13 percent is assigned to molding and cokemaking; 13 percent to space heating; 6 percent to pollution control; 6 percent to casting, shakeout and cleaning; and 1 percent to other operations.

The production of gray iron castings is considered to be a relatively direct and straightforward processing route, consisting primarily of the melting and casting of the iron in prepared molds. This fact, coupled with the lack of any envisioned alternative process which could conserve energy relative to the conventional operation, was taken to indicate that the need for and probability of developing energy-saving replacement processes for producing rion castings was low.

* The energy associated with electric furnace melting of gray cast iron is considered comparable to that of cupola furnace melting.

IRON AND STEEL - STEEL SLABS

The average energy required to produce one net ton of steel slabs in the U. S. is about 24×10^6 Btu. Since the 1973 U. S. consumption of steel slab equivalent was about 140×10^6 net tons; the total energy associated with the 1973 consumption of this material was $3,350 \times 10^{12}$ Btu.

The theoretical minimum energy required to produce one net ton of steel slabs is 7.5×10^6 Btu. As indicated in Table 1 in the Introduction, the overall efficiency (i.e., useful output energy/total input energy x 100) is about 41 percent for both electrical to thermal conversion factors. In general, the major unit operations associated with the steelmaking industry are relatively efficient. Such operations as the by-product coke ovens, where coal is converted to metallurgical coke, and the blast furnace, where iron ore is reduced with coke to yield liquid hot metal, are characterized by very high efficiency and effectiveness values and are often cited as model operations from an energy utilization point of view.

While the efficiencies of the overall processes and the individual unit operations are relatively high for the steelmaking industry, the desirability of developing energy-saving alternative processes to the conventional route(s) is considered to be reasonably great. This judgment is based on the facts that (1) the production of steel consumes the greatest amount of energy on a total annual basis of any of the high-priority commodities, and (2) potentially energy-conserving alternative routes to steelmaking have been envisioned and have been, in part, examined in bench-scale experimental studies.

A group of alternative processes to produce steel, many of which appear to provide an energy-savings relative to the conventional route, are the continuous iron- and steelmaking processes. Approaches to continuous steelmaking are described in the following section.

Continuous Iron and Steelmaking

Present day processes for converting iron ore into steel consist essentially of two major stages. The first is production of an impure iron-carbon alloy (e.g., pig iron) from the ore (primarily iron oxides) and is called ironmaking. The second, referred to as steelmaking, involves refining and alloying of the impure iron to produce steel.

Each of the two stages can be conducted in a variety of ways that depend on the grade of the ore, energy forms available, size of the operation, capital and operating costs, etc. For example, methods for making iron from ores include:

(1) Reduction of the ore with hot gases and coke to produce molten pig iron, as in a blast furnace;
(2) Reduction of the ore with low-temperature gases to produce sponge iron;
(3) Smelting reduction, in which the ore is melted and then reduced by injecting powdered coal or other suitable reductants; this method is still in the developmental stage.

Similarly, the iron (i.e., iron-carbon alloy) produced in the ironmaking stage can be refined to produce steel by several methods. When the iron alloy is molten as produced, it is commonly refined in either an open-hearth furnace or a basic oxygen furnace, where the impurities (e.g., carbon, silicon, etc.) are removed by oxidation and/or reactions with a slag. Sponge iron must be melted prior to refining. Thus, it is commonly refined in an electric-arc furnace, though some may be mixed with molten pig iron and refined in the open-hearth or basic oxygen processes.

There is an interesting difference between the ironmaking processes on the one hand and the steelmaking processes on the other. The former are basically continuous operations (though the blast furnace, for example, is usually tapped only intermittently), while the latter are batch processes. The fact that the iron refining operation is done in batches precludes feeding iron ore continuously into one end of a plant and having steel products emerge continuously from the other end. However, if such a continuous iron- and steelmaking plant could be developed, it would have obvious advantages, not the least of which would be reduced energy consumption. This could come about from several sources, including (a) avoidance of heat losses during tapping and transferring of batches of pig iron, (b) decreased consumption of refractory materials because of fewer instances of thermal shock, and (c) potentially simpler procedures and devices for controlling environmental pollution.

Visionary steelmakers see numerous advantages to continuous iron- and steelmaking and are pursuing possibilities along two fronts. Some take the approach that the currently most important method for making iron, the blast furnace, is a perfectly acceptable and basically continuous source of impure iron and that only the steelmaking process must be revamped to make it continuous. Others contend that both the ironmaking and the steelmaking operations need to be reexamined if an optimum continuous operation is to be realized.

With respect to the first-mentioned view, numerous continuous steelmaking processes are currently undergoing development. Some of the more advanced of these are described briefly below, along with their potential for conserving energy based on thermodynamic considerations.

- The IRSID Tank-Type Process (France)[(1)]. The principle of the IRSID continuous steelmaking process is shown in Figure 7. Molten pig iron, oxygen, lime, and cooling agents (scrap, for example) flow continuously into the reactor, where impurities are removed by oxidation and by reactions with the slag. The mixture pours continuously into the decanting vessel, where the slag and purified iron separate. The iron is continuously siphoned away to an adjustment furnace where appropriate alloying additions are made to ensure the correct grade of steel.

  Glinkov has examined the thermodynamics of various continuous steelmaking processes[(2)]. He concludes that the IRSID process, by virtue of efficient utilization of oxygen consumes only about 93.5 percent of the energy required in basic-oxygen steelmaking. This is accompanied by lesser formation of brown smoke, which means that yields are improved and exhaust gases contain less particulate matter.

- Spray Steelmaking at BISRA (United Kingdom)[(3)]. The principle of this process is to accelerate steelmaking by increasing the exposure area of the iron to oxygen. As molten iron from the blast furnace pours into the spray unit, a ring of oxygen jets atomizes the stream and rapidly oxidizes the impurities. A second ring of nozzles adds lime and fluxes to the spray to form a slag of oxidized impurities. By the time the stream

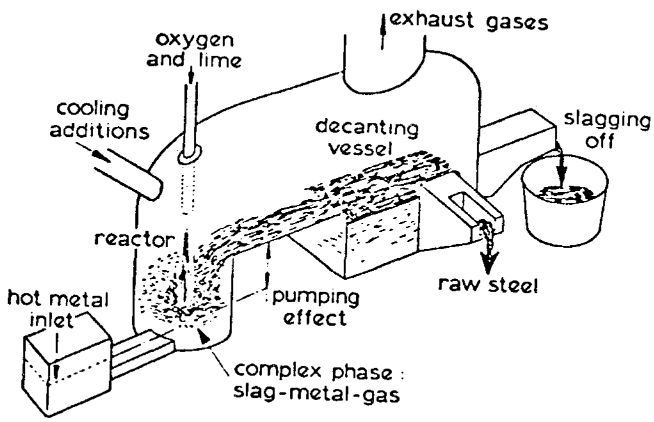

FIGURE 7. SCHEMATIC ILLUSTRATION OF IRSID STEELMAKING PROCESS(1)

reaches the bottom of the reaction unit, the steelmaking reactions have been completed. The products drop into another vessel, where slag is continuously drawn off and where composition is adjusted. According to Glinkov's thermodynamic analysis,[2] the spray steelmaking process requires approximately 94.5 percent of the energy consumed in the basic-oxygen process.

- The WORCRA Single-Stage Trough-Type Process (Australia)[4]. The WORCRA continuous steelmaking furnace has three zones in communication in a horizontal plane, as depicted in Figure 8:
  (1) A central bowl-shaped feed zone in which hot metal from the blast furnace and slag flow co-currently.
  (2) An elongated refining zone in which oxygen is jetted sequentially into the slowly flowing metal stream to oxidize impurities and in which slag, generated by lime additions near the outflow end, flows countercurrent to the metal.
  (3) A slag clean-up zone in which metal shot can settle out of the slag and gravitate back along a sloping hearth to the bowl.

  The refined metal flows continuously to another unit for compositional adjustments prior to continuous casting.

There is some lack of agreement over the energy-consumption characteristics of this process. Glinkov's thermodynamic analysis[2] indicates that it requires about 149 percent of the energy required by the basic oxygen process, despite Baker and Wormer's claims of high yields, economy of oxygen and lime usage, little fume generation and low refractory consumption. Apparently, Glinkov's figures were obtained from tests on small units in which scrap usage was unusually small and heat losses were high.

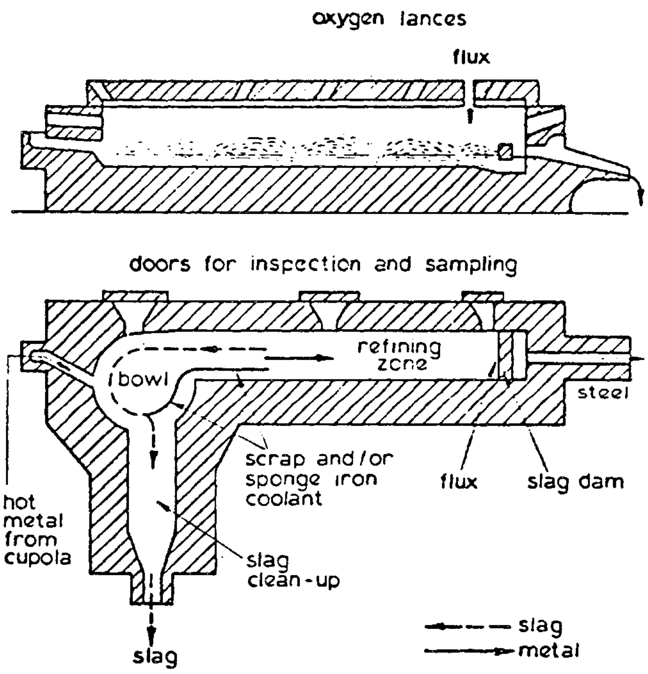

FIGURE 8. SCHEMATIC ILLUSTRATION OF WORCRA CONTINUOUS STEELMAKING PROCESS(4)

Other continuous steelmaking processes that are undergoing development include the Moscow Institute of Steel and Alloys (MISiS) single-stage multi-chamber hearth-type process[2] and the NRIM (Japan) multi-stage trough-type process.[5] The MISiS process is a system of open baths in which large quantities of scrap (~40 percent) can be melted. According to Glinkov, this process consumes about 5 percent more energy than the basic oxygen process. In the NRIM process, steelmaking reactions are separated into three groups--desiliconization and desphosphorization, decarburization, and final control of steel grade. No energy consumption figures are available for this process.

If any of the above-described continuous methods for steelmaking were to be combined with existing methods of ironmaking -- the blast furnace, for example, which as already noted is basically a continuous process -- and with continuous casting, the goal of achieving continuous output of steel from continuous input of raw materials would be realized. There is, however, another approach to continuous operation that utilizes a different type of ironmaking process. For example, smelting reduction (described earlier) is a truly continuous process and one which, according to Eketorp and Brabie,[6] consumes less energy than the blast furnace process.

Eketorp[7] has proposed a continuous steel plant that would utilize smelting reduction to produce molten impure iron. As shown in Figure 9, the plant would consist of several units. In the ironmaking unit, a reducing agent such as oil, gas, or coal powder is blown into a bath along with an iron oxide concentrate. The oxide melts and is reduced to iron. Apparently, when the iron oxide is reduced in the molten state, the molten iron will not contain silicon, manganese, or phosphorus[8]. The major impurity will be carbon (3-4 percent). The CO formed in the oxide reduction is burned with oxygen above the bath to provide heat to maintain the endothermic reduction reaction. Most of the sulfur contained in the reducing agent is absorbed by the molten iron, to be removed in a subsequent desulfurization unit with CaO.

As the molten iron from the reduction reactor travels toward the steelmaking unit, it is admixed with molten steel scrap melted in a continuous high frequency induction furnace. In the steelmaking unit, carbon is oxidized from above with oxygen. Fine scrap and iron oxide are added to cool the bath. The CO in waste gases is burned in a heat exchanger. There is little need for slag-forming additions since the iron is low in S, Si, Mn, and P.

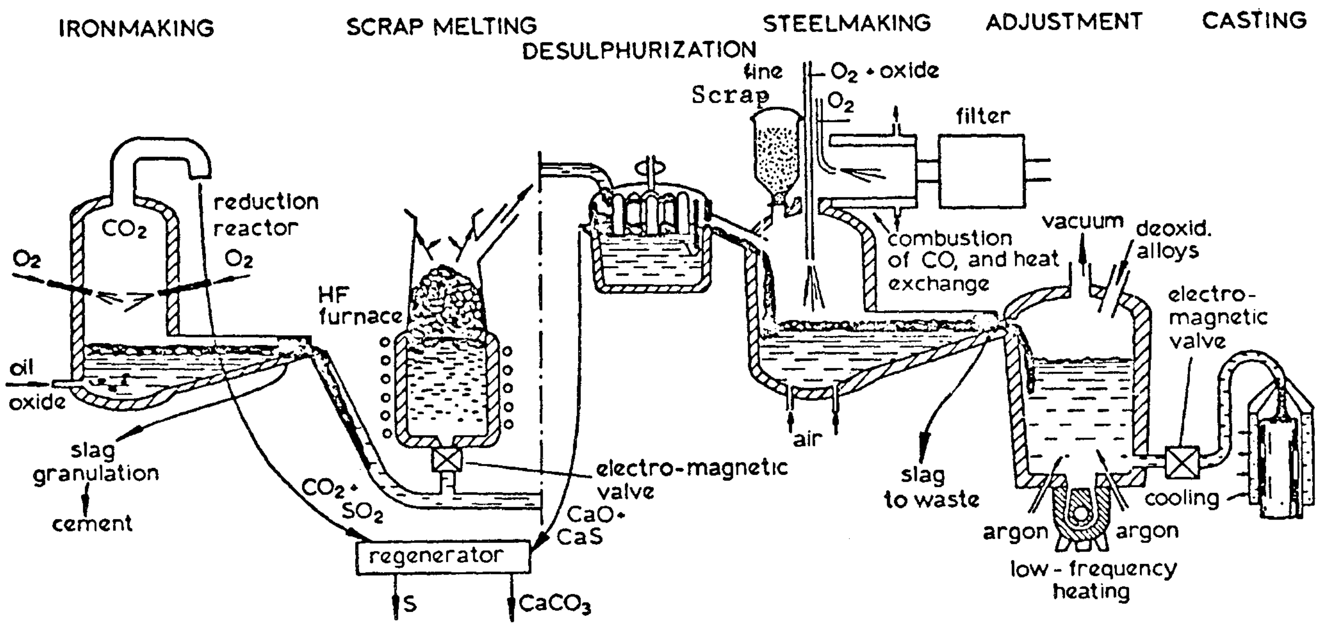

FIGURE 9. PROPOSAL FOR A CONTINUOUS STEELPLANT (7)

The steel then passes into an adjustment vessel, where alloying and deoxidation are accomplished. From this vessel, it passes through a flow valve to a continuous casting machine.

The smelting reduction reactor proposed by Eketorp is similar to a continuous ironmaking process patented by Agarwal and Davis.[9,10] As shown in Figure 10 this process also reduces the ore entirely in the liquid state. Melting is accomplished by preheating the ore and limestone and passing them through an oxy-fuel burner. Molten material collects in a bath from which it drains into a second chamber, where powdered coal (or oil or natural gas) is injected beneath the surface to reduce the molten oxide to iron. Melting the ore prior to reduction permits the use of the stoichiometric amount of reducing agent and makes unnecessary the production of high-carbon pig iron. The impurities, which according to Eketorp[8] should be primarily sulfur from the reducing agent, collect in the lime-bearing slag.

To keep the endothermic reaction going in the reducing chamber, much of the CO formed is burned with oxygen above the bath. The remaining CO is piped to the melting chamber where it is used to melt the ore. The spent gases, completely burned, are used to preheat the ore and are discharged with no remaining heating value.

The molten iron from the reducing chamber can then be made into steel by addition of appropriate alloying elements in an adjustment vessel similar to that shown in Figure 9.

Another process that bears resemblance to the smelting reduction method for ironmaking is one being pursued by the British Iron and Steel Research Association (BISRA).[11] Low grade coal is mixed with fine ore concentrate and added at one end of a high speed horizontal centrifuge. The ore is heated, presumably to melting, by means of an oxy-oil burner, and the coal reduces the oxide to iron and causes CO gas to be generated. The centrifuge segregates the liquid iron to a layer in contact with the refractory walls and protects the refractory from attack by the inner liquid layer of iron bearing slags. Oxygen is added to burn the CO to CO_2, hence releasing sufficient heat in the central core to maintain the strongly endothermic iron-ore reduction reaction. Liquid iron and depleted slag are discharged at the exhaust end of the centrifuge, from which point the iron can be sent to a refining unit.

The energy requirements for ironmaking are less for smelting reduction than for the blast furnace process, according to a thermal analysis carried out by

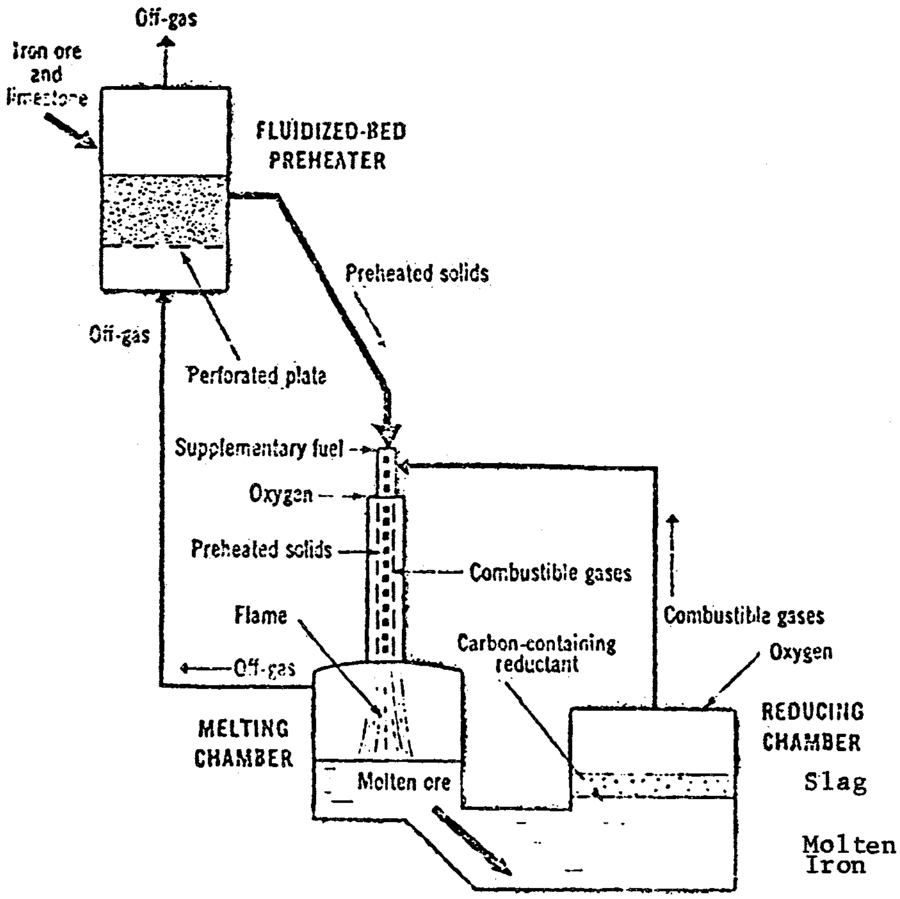

FIGURE 10. CONTINUOUS IRON MAKING PROCESS PATENTED BY AGARWAL AND DAVIS[10]

Eketorp and Brabie[6]. Assuming that all energy is generated from fossil fuels, the analysis shows that smelting reduction consumes about 14 percent less energy than does the blast furnace. Part of this energy advantage is lost, however, when the impure molten iron is refined, along with 25 percent scrap, in an oxygen converter, presumably because the chemical heat contained in the smelting-reduced iron is less than that in conventional hot metal (less impurities to oxidize). Nonetheless, the smelting reduction route to steelmaking still retains an energy advantage of about 8 percent.

At present, the technology for smelting reduction and for continuous steelmaking are still undergoing development. Thus a continuous steelplant is still some years away. Nonetheless, it is almost certain to come and when it does, it will almost certainly be more thermally efficient than the batch processes currently in use.

Conclusions - Iron and Steel Slabs

While the specific energy required to produce a net ton of steel slabs is intermediate in a ranking of the high-priority commodities and the unit operations associated with steelmaking are relatively efficient, the need to consider and develop alternative energy-saving processes for steelmaking is judged to be moderate to high. This judgment is in part based on the fact that the annual production of steel slabs accounts for the largest energy consumption of any of the high-priority commodities. This position is also based on the existence of ongoing development efforts to secure energy-conserving alternative processes for steelmaking, such as the continuous iron and steelmaking processes. Essentially all the continuous steelmaking processes reviewed in this section are anticipated to provide an energy saving of about 10 percent over the conventional production route(s). While these continuous steelmaking methods are envisioned as energy-saving, their development is a long-range undertaking and their utilization on a large scale is not anticipated within the next 10 years.

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LEAD

The production of refined lead requires 27 x 10^6 Btu per net ton. As 1973 U.S. consumption of refined lead was 887,000 net tons, the total annual energy associated with the U.S. refined lead consumption was slightly less than 24 x 10^{12} Btu. While the production of lead is not as energy intensive as many other of the high-priority primary products, there does appear to be both some need and potential means for replacing the current production processes with energy-saving alternatives.

Various proposals have been made from time to time to recover lead entirely by hydrometallurgical means, or to electolyze lead sulfide in a molten chloride electrolyte. A practical hydrometallurgical treatment may be possible, but cannot yet be considered a demonstrated technology. Recovery of lead by electrolysis of a molten salt may take surprisingly little electrical energy for electrolysis, but other sources of energy are needed to maintain the electrolytic cells at 500 C, heat leaching solutions, treat residues, and operate the plant. Some overall energy saving may be possible relative to the present blast-furnace-refining procedure; however, the energy saving potential must be regarded as only fair. The possibility of recovering elemental sulfur is a decided asset. However, disposal of waste products and possible air contamination from chlorine and chlorides present adverse problems.[1,2,3,4]

Several companies have been doing active research on direct reduction of galena concentrates to utilize more completely the fuel energy in the sulfur.[2,3,4] Theoretically, lead sulfide plus oxygen should be converted to lead and sulfur dioxide with little need for extra fuel. This is the principle of the ore or Scotch hearth practice. A single step smelting or conversion of galena concentrates appears to offer potential for energy savings. Such a process would avoid sintering and the conventional blast furnace. So far insufficient pilot plant work has been done under conditions simulating plant practice to make a practical comprehensive calculation of energy savings. As an indication, however, the heat balance in a small converter showed the need for 140,000 Btu per ton of concentrates or 200,000 Btu per ton of lead produced under conditions of no external heat losses.

The Imperial Smelting Furnace Process, which was originally developed as a means of producing zinc in a blast furnace, has been found to be applicable to smelting mixed medium-grade lead-zinc concentrates.[8] This offers some advantage in handling bulk lead-zinc concentrates by giving better concentration efficiencies. The addition of lead to the charge is not considered to increase the carbon required in the smelting process. Lead compounds in the sinter are reduced to metal by carbon monoxide in the upper part of the furnace or with waste gases produced from reduction of zinc in the main part of the furnace. However, slag constituents accompanying the lead do require some fuel for separation unless they are chargeable to the zinc in mixed concentrates. Although there is a strong possibility for making considerable energy savings by using this process for mixed concentrates, or even by purposely adding lead concentrates to the charge, its adoption in the United States is doubtful. Metallurgical coke is scarce and expensive, hence the trend in the United States is to use electrolytic zinc plants. Also, payment of royalties for a process developed abroad may be an adverse factor. So far the Imperial Process has not been used in the United States and no such plant is planned, although 12 or 13 such furnaces have been constructed in other countries.

Conclusions - Lead

Engineering improvements in concentration, preparation of charge, and blast furnace operation appear possible with known technology, but the energy savings per ton of lead probably can be reduced in the near future on the order of only 10 percent. However, substantial energy savings appear possible by either of two new developments. Use of the Imperial Smelting Process, as now used commercially and successfully under some conditions for zinc recovery, has possibilities for handling considerable lead with little or no extra coke required. Although there may be practical objections to using this process for zinc and lead, its potential energy savings cannot be ignored in this discussion. The second process, direct smelting of galena concentrates, has not yet been developed to the commercial stage, but experimental work has shown sufficient savings in energy to warrant encouragement of further development.

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NITROGEN

The energy required per net ton of gaseous and liquid nitrogen is 2.9 and 8.1 x 10^6 Btu, respectively. Because the 1973 U.S. consumption of gaseous and liquid nitrogen was 5.6 and 2.7 x 10^6 net tons, respectively (i.e., 8.3 x 10^6 net tons total), the total annual energy associated with nitrogen usage was 38 x 10^{12} Btu.

Gaseous nitrogen (and the co-products, oxygen and argon) are produced from atmospheric air by fractional distillation at cryogenic temperatures. Liquid nitrogen is, in turn, produced by condensation (i.e., liquefaction) of the gaseous nitrogen.

Because the processes for producing gaseous and liquid nitrogen are essentially direct one-step operations and because the energy requirements for the production of these two forms of nitrogen are relatively low in comparison to those of the other high-priority materials, the need for developing energy-saving alternative processes is taken to be low.

NITROGEN - AMMONIA

The production of ammonia from hydrogen and nitrogen requires 39×10^6 Btu per net ton of product. The 1973 U. S. consumption of ammonia was 15×10^6 net tons indicating a total annual energy of 586×10^{12} Btu associated with ammonia consumption.

While the energy requirement to produce ammonia is an intermediate value compared to that for the other high-priority materials, the need for developing energy-saving alternative processes for ammonia production is considered to be relatively low.

In general, the conventional process for producing ammonia from hydrogen and nitrogen is relatively straightforward with the energy content of the feed, natural gas, accounting for about 58 percent (i.e., 22.3×10^6 Btu) of the total energy required to produce one net ton of product.

An energy-saving alternative method for producing ammonia, which is currently in the early research stages of development at Stanford University, is the acid solution treatment of nitrogen with certain molybdenum compounds in a similar manner to the enzyme nitrogenation processes.[(1)] While these processes are not sufficiently developed to define their energy requirements, they offer the potential for energy conservation if the yield of ammonia per unit of feed compound can be increased to a significant level.

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PHOSPHORUS - PHOSPHORIC ACID

The production of phosphoric acid requires about 10.8×10^6 Btu per net ton of P_2O_5. Because the 1975 U. S. consumption was 5.6×10^6 net tons of P_2O_5 as phosphoric acid, the total energy required for the annual production of this commodity was 61×10^{12} Btu.

Production of phosphoric acid in the U. S. is predominantly by the wet-acid route with sulfuric acid. The wet-acid process is relatively direct with the acid digestion and recovery operation requiring 38 percent of the total energy for producing the phosphoric acid. Mining and preparation of the phosphate ore require 33 percent of the total energy, and the product concentration step requires 29 percent of the total energy.

Because of the relatively low energy associated with a net ton of this material and because the current production process is relatively simple, the need for developing energy-saving replacement technology is assessed to be low.

PHOSPHORUS - ELEMENTAL PHOSPHORUS

The average energy required to produce one net ton of elemental phosphorus is estimated to be about 172 x 10^6 Btu. Coupled with a 1973 U. S. consumption of about 0.5 million net tons, the total energy required for the U. S. consumption amounts to about 86 x 10^{12} Btu.

The electric arc furnace is the major energy consuming operation in the production of elemental phosphorus. The arc furnace is estimated to use about 130 x 10^6 Btu per ton of elemental phosphorus as electrical energy (i.e., 12,500 kilowatt-hours per net ton of P) and about 34 x 10^6 Btu per ton of product as coke breeze, which is a consumed feed material. Thus, the electric furnace operation accounts for about 96 percent of the total energy associated with a ton of elemental phosphorus. Robiette estimates that typical electric-arc phosphorus smelting furnaces are characterized by a thermal efficiency (i.e. useful energy output/total energy input x 100) of about 84 percent(1).

Elemental phosphorus can be produced in a blast furnace of similar design and operation to those used in the production of pig iron(2,3). The input materials and basic reaction for phosphorus production are the same for both electric-arc furnace and blast furnace operations. The primary difference between the two operations is that electrical energy is used to heat the reactants in the electric-arc furnace, while the burning of coke provides the necessary thermal input in the blast furnace operation (i.e., the blast furnace coke serves as both fuel and reductant). The production of phosphorus in a blast furnace requires 5.7 to 6.9 tons of coke per ton of elemental phosphorus*(3). Since the assigned energy value of coke is 31.5 x 10^6 Btu per net ton, the energy required for producing phosphorus in the blast furnace is greater than 180 x 10^6 Btu per ton. Thus, the blast furnace production of elemental phosphorus is more energy intensive than the electric furnace process. While improvements (i.e., reduction of energy requirements) in the blast-furnace production of phosphorus should result from utilization of higher blast temperatures, oxygen enrichment of the blast, etc., it is anticipated that the blast

* Equivalently, about 2.5 to 3 tons of coke per ton of P_2O_5 reduced.

furnace operation will continue to require a greater amount of energy to produce phosphorus than the electric-arc furnace.

Another method suggested for the production of elemental phosphorus is the fluidized bed-plasma arc reactor[(4)]. While the energy required to produce a unit of phosphorus in an experimental plasma bed reactor is not sufficiently documented to allow a definitive statement of potential for conservation relative to conventional practice, the fluidized bed-plasma arc system should be reasonably comparable on an energy basis to the electric-arc furnace operation.

While the energy required to produce a net ton of elemental phosphorus is among the greatest of the high-priority group of materials, the conventional electric-arc furnace process is relatively direct and thermally efficient. The production of phosphorus is expressed by the reaction:

$$Ca_3P_2O_8 + 3\ SiO_2 + 5C = 3\ CaSiO_3 + 5CO + P_2$$

which essentially involves the application of heat to the phosphate rock-silica sand mixture to yield P_2O_5, which is in turn reduced by the carbon, to yield phosphorus.

Thus, the reaction(s) involved are relatively direct and do not involve numerous multiple-step complex reactions. Also, no energy-saving alternative processes to the electric-arc furnace process are presently known for producing phosphorus on a large-scale production basis and the total annual energy consumption by the industry is relatively small.

Because of the above factors we consider the need for developing alternative energy-saving processes to produce elemental phosphorus to be between low and desirable.

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REFRACTORIES - BASIC BRICK

The average energy required per net ton of basic refractory brick is 27×10^6 Btu. Because the 1973 U.S. consumption of basic refractory brick was about 675,000 net tons, the total annual energy requirement was about 18×10^{12} Btu.

The major energy consuming step in the production of basic brick refractory is the mining and processing of dead-burned magnesia (MgO), which accounts for about 71 percent (i.e., about 19×10^6 Btu) of the total per ton energy requirement. Curing, drying, and firing of the pressed bricks is the second largest energy consuming operation, accounting for about 14 percent of the total energy requirement.

The need for developing a replacement technology or processes as a means of saving energy in the production of basic refractory brick is considered to be low. This position is taken because the production of basic refractory brick should continue to require dead-burned magnesia as the major input material and the conventional production route is relatively straightforward involving basically heating operations. No new energy-conserving replacement procedures are envisioned.

REFRACTORIES - FIRECLAY BRICK

The energy required per net ton of fireclay brick is 4.2×10^6 Btu. The 1973 U.S. consumption of this commodity was about 1.6×10^6 net tons, which implies a total energy requirement of 6.9×10^{12} Btu for fireclay brick consumption. Drying and firing the fireclay bricks require about 95 percent of the total energy requirement (i.e., 4.04×10^6 Btu per net ton).

Because the production of fireclay brick is essentially a one-step heating operation and the energy requirement per ton of product is low relative to that of the other high-priority materials, the need for developing energy conserving alternative processes for producing fireclay brick is judged to be low. No new energy-saving replacement processes are known to the BCL reviewers.

SULFUR - SULFURIC ACID

The production of one net ton of sulfuric acid from Frasch-mined sulfur requires 0.83×10^6 Btu. The U. S. 1973 consumption of sulfuric acid was 30.5×10^6 net tons implying a total annual energy of 1.2×10^{12} Btu associated with the primary product.

Because (1) the energy required to produce a net ton of sulfuric acid is the lowest of the high-priority materials and (2) the production is a relatively simple and direct sequence of operations (i.e., burning of elemental sulfur to yield SO_2; converting the SO_2 to SO_3 via oxidation, and absorbing the SO_3 in recirculating sulfuric acid containing water) the need for and probability of developing energy-saving processes alternative to the conventional method is considered to be very low.

ZINC

The production of elemental zinc requires about 65 x 10^6 Btu per net ton. As the U.S. consumption of elemental zinc in 1973 was about 1.4 million net tons, the total annual energy consumed for zinc production was 92 x 10^{12} Btu. The production of zinc is reasonably energy intensive relative to the other high-priority primary products. There also appear to be technically feasible alternative processes for zinc production with energy-saving potentials. Thus, the introduction of energy-conserving replacement technology for the current processes is judged to be desirable.

While the electrolytic method of producing refined zinc requires much less energy than the electrothermic and the vertical retort methods, its recovery ratio of zinc from concentrates is typically about 85 to 88 percent, which is less than the recovery ratio associated with these other processes. Depending on the iron content of the concentrate, the recovery of zinc from concentrates by the electrothermic and the vertical retort methods is 95.6 and 90 percent, respectively. A significant breakthrough in electrolytic zinc technology is the development of a low cost method of recovering additional zinc from leach residues. With an appropriate residue treatment plant, recoveries of 95 percent can be achieved. A process which is receiving much attention at present, involves a jarosite method of precipitating iron in easily filterable form out of solutions from a hot acid digestion of the residues. Energy expenditure for this additional step is low enough to bring the total energy requirements for producing zinc considerably under that required for the competing pyrometallurgical methods. The additional energy associated with the residue treatment involves some pumping and agitation, heating the leach solution to 185-200 F, and the energy equivalent of aqueous ammonia used to precipitate iron as ammonium jarosite. The electrolytic plus leach residue method is used in Japan, Australia, and Norway. An installation is being planned in Canada but, so far, no firm plans for such an installation have been announced in the United States.

Another development of recent years, the Imperial Smelting Furnace (ISF) Process(1), is applicable to smelting lead-zinc concentrates. There is a possibility of energy savings to be realized by using the process for unseparated concentrates. It is especially applicable to zinc-lead ores which are difficult to separate by beneficiation. A recent article(2) describing the Imperial Smelting Furnace operation indicates that the ISF route could save about 4.5 x 10^6 Btu per net ton of zinc produced relative to the conventional U.S. zinc electrolytic plant/

lead blast furnace/slag furnace systems.* However, the combination of the type of ores encountered in the United States, the trend in the U. S. toward electrolytic processing, and the fact that royalities are required, are factors which may retard its adoption in the United States. There are currently about a dozen Imperial Smelting furnace installations throughout the world.

Thought has been given to the possibility of applying flash smelting (plus zinc vaporization) to the production of zinc because of the high potential for energy savings. Fundamental difficulties in getting reasonably good reduction of the zinc sulfide without excess oxygen and, in turn, condensing metallic zinc in an atmosphere of sulfur oxides are such that this is not considered promising. However, with continued development of the Imperial Smelting process some measure of utilizing the heat from combustion of zinc sulfide may result eventually.

Conclusions - Zinc

The new jarosite method of precipitating iron from leach residues provides much better yields in electrolytic refining with low additional energy requirements, but has not yet been adopted by the U. S. primary zinc industry.

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* This energy saving is for 1 net ton of zinc plus 0.5 ton of lead bullion produced in the ISF process.

BIBLIOGRAPHIC DATA SHEET	1. Report No. BuMines OFR 117(4)-76	2.	PB 261 153

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16. Abstracts This report analyzes areas where alternative technologies should be developed to reduce energy required in the production of 14 high-volume commodities. Primary attention is given to aluminum, lime, cement, common brick, copper, glass, iron and steel slabs and castings, lead, nitrogen-ammonia, phosphorus-phosphoric acid, basic and fireclay refractories, sulfuric acid, and zinc.

17. Key Words and Document Analysis. 17a. Descriptors

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Materials
Minerals
Metals
Nonmetallic mineral products
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Energy saving
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17b. Identifiers/Open-Ended Terms

17c. COSATI Field/Group 05C, 10, 11F

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