ENERGY

CONFIDENTIAL

DR. 2085

CONS-5088-1

SLAG WASTE HEAT RECOVERY AND UTILIZATION IN
THE ELEMENTAL PHOSPHORUS INDUSTRY

Final Report, October 28, 1977–April 30, 1978

By
Tom E. Ctvrtnicek
Robert J. McCormick
Robert W. Serth
Anatol Wojtowicz
Donald L. Zanders

May 15, 1978

Work Performed Under Contract No. EM-77-C-01-5088

Monsanto Research Corporation
Dayton Laboratory
Dayton, Ohio

U. S. DEPARTMENT OF ENERGY

MASTER

DISTRIBUTION OF THIS DOCUMENT IS UNLIMITED

DISCLAIMER

This report was prepared as an account of work sponsored by an agency of the United States Government. Neither the United States Government nor any agency Thereof, nor any of their employees, makes any warranty, express or implied, or assumes any legal liability or responsibility for the accuracy, completeness, or usefulness of any information, apparatus, product, or process disclosed, or represents that its use would not infringe privately owned rights. Reference herein to any specific commercial product, process, or service by trade name, trademark, manufacturer, or otherwise does not necessarily constitute or imply its endorsement, recommendation, or favoring by the United States Government or any agency thereof. The views and opinions of authors expressed herein do not necessarily state or reflect those of the United States Government or any agency thereof.

DISCLAIMER

Portions of this document may be illegible in electronic image products. Images are produced from the best available original document.

NOTICE

This report was prepared as an account of work sponsored by the United States Government. Neither the United States nor the United States Department of Energy, nor any of their employees, nor any of their contractors, subcontractors, or their employees, makes any warranty, express or implied, or assumes any legal liability or responsibility for the accuracy, completeness or usefulness of any information, apparatus, product or process disclosed, or represents that its use would not infringe privately owned rights.

This report has been reproduced directly from the best available copy.

Available from the National Technical Information Service, U. S. Department of Commerce, Springfield, Virginia 22161.

Price: Paper Copy \$10.75
Microfiche \$3.00

SLAG WASTE HEAT RECOVERY AND
UTILIZATION IN THE ELEMENTAL
PHOSPHORUS INDUSTRY

Final Report

Period of 28 October 1977 to 30 April 1978

15 May 1978

MONSANTO RESEARCH CORPORATION
DAYTON LABORATORY
Dayton, Ohio 45407

Authors

Tom E. Ctvrtnicek
Robert J. McCormick
Robert W. Serth
Anatol Wojtowicz
Donald L. Zanders

Work Performed Under Contract EC-77-C-01-5088

Prepared for

U.S. Department of Energy
Office of Industrial Energy Conservation
Washington, D.C. 20545

NOTICE

This report was prepared as an account of work sponsored by the United States Government. Neither the United States nor the United States Department of Energy, nor any of their employees, nor any of their contractors, subcontractors, or their employees, makes any warranty, express or implied, or assumes any legal liability or responsibility for the accuracy, completeness or usefulness of any information, apparatus, product or process disclosed, or represents that its use would not infringe privately owned rights.

THIS PAGE
WAS INTENTIONALLY
LEFT BLANK

ABSTRACT

Approximately 80×10^{12} Btu/yr of thermal energy are contained in molten slags produced by the elemental phosphorus industry, the iron and steel industry, the copper industry, and wet-bottom coal-fired boilers. This study evaluates the technical, economic and environmental feasibility of recovering this wasted energy; the impact of slag waste heat recovery on the industries in question; and the steps necessary to commercialize applicable heat recovery technology.

The study considered two approaches to recovering thermal energy from phosphorus slag: the float chamber and the contact tower. Based on these approaches, nine energy recovery options for converting the energy in slag into other usable forms of energy were conceptualized and economically evaluated. All nine options are considered technically feasible and environmentally sound. The economics of the nine options are based on 33.9 kg/s (269,000 lb/hr) of slag throughput and vary with both the energy from produced and the realizable total credits for different energy forms. Slag byproduct credit is generally needed to make heat recovery economically attractive.

Slag waste heat recovery offers considerable potential for energy savings in the elemental phosphorus, iron and steel, and copper industries. Additional studies are recommended to determine if sufficient byproduct credits can be obtained to justify this technology economically.

THIS PAGE
WAS INTENTIONALLY
LEFT BLANK

CONTENTS

<u>Section</u>	<u>Page</u>
Abstract	iii
Summary and Conclusions	ix
1 Introduction	1
2 Slag Waste Heat Recovery in the Elemental Phosphorus Industry	3
2.1 Current Use of Slag Heat	3
2.2 Technical, Environmental, and Economic Feasibilities	3
2.2.1 Waste Heat Recovery Options	4
2.2.2 Technical and Environmental Feasibilities of the Slag Waste Heat Recovery Options	5
2.2.3 Systems Economics	11
2.2.3.1 Energy Recovery Credits, Costs, and Profitability Estimates	11
2.2.3.2 Potential Byproduct Credits	19
2.3 Expected Impact in the Elemental Phosphorus Industry	25
2.4 Commercialization	26
2.4.1 Elemental Phosphorus Industry	26
2.4.2 Iron and Steel Industry	29
2.4.3 Copper Industry	32
2.4.4 Wet-Bottom Coal-Fired Utility Boilers	35
References	37
Appendices	39
A Elemental Phosphorus Production, Process Description and Energy Utilization Alternatives	46
B Slag Properties, Summary	52
C Slag Handling and Transportation, Technical Discussion and Economics	54
D Surge Vessel, Technical Discussion and Economics	61
E Slag Waste Heat Recovery - Float Chamber, Technical Discussion	66
F Slag Waste Heat Recovery - Float Chamber, Economics	146

CONTENTS (continued)

	<u>Page</u>
G Slag Waste Heat Recovery - Contact Tower, Technical Discussion	156
H Slag Waste Heat Recovery - Contact Tower, Economics.	215
I Heat Recovery Boilers, Technical Discussion and Economics	223
J Condensing Turbines/Electricity Generator, Technical Discussion and Economics	229
K Noncondensing Turbines/Electric Motor Displacement, Technical Discussion and Economics	234
L Energy Losses During Slag Transportation, A Crude Estimate	239
M Surge Vessel Heat Loss, A Crude Estimate	241
N Conversion Factors and Metric Prefixes	242

FIGURES

<u>Number</u>		<u>Page</u>
1	Slag waste heat recovery options and systems can be classified by the energy forms produced (circled).	6
2	Steam price has a significant influence on the ROI for steam generation options.	16
3	Option 3 is the most economically attractive electricity generation or displacement option . . .	16
4	Price of natural gas has a significant influence on the ROI for hot air generation	17

TABLES

<u>Number</u>		<u>Page</u>
1	Energy Recovery Credits	12
2	Slag Waste Heat Recovery Systems Summary of Costs and Profitability	13
3	Byproduct Credits and Market Penetration Needed to Realize a 10% ROI From Heat Recovery Options 3 and 8.	22
4	Pricing Data for Materials Representative of Potential Slag Applications	23
5	Efficiencies of Slag Waste Heat Recovery Options.	25
6	Estimates of Potential Electricity Savings for the Elemental Phosphorus Industry, Electricity Generation Efficiency of 22%.	27
7	Relative Power Requirements by Producer and Power Suppliers for the Elemental Phosphorus Industry.	28
8	1976 Pig Iron Slag Production and Pricing Data. .	29
9	Typical Slag Compositions	32

SUMMARY AND CONCLUSIONS

Some 80×10^{12} Btu/yr (~ 3.0 GW_{thermal}) of thermal energy are available for recovery in molten slags produced by the elemental phosphorus industry, the iron and steel industry, the copper industry, and coal-fired slagging boilers. The major objectives of this study were to determine (1) the technical, economic, and environmental aspects of recovering this waste heat; (2) the impact of slag waste heat recovery on the industries listed above; and (3) the steps necessary to commercialize slag waste heat recovery technology. The elemental phosphorus industry was selected as a model for this preliminary feasibility study. All engineering and cost evaluations are based on a plant producing 33.9 kg/s (269,000 lb/hr) of slag. The results of this specific evaluation are extrapolated to the other industries of interest to determine the overall feasibility of slag waste heat recovery.

Two concepts to recover thermal energy from slag have been evaluated in detail. Since slag waste heat recovery is not currently being practiced in the phosphorus industry, the evaluations could not be based on actual operating experience. The two concepts are based on technologies used in other industries for purposes other than energy recovery.

The first concept is based on float chamber technology used in the flat-glass manufacturing industry. In this concept, molten slag is poured onto a pool of molten tin where it cools and solidifies. As the slag moves down the pool, heat is transferred by conduction to submerged steam pipes and by radiation to overhead tubes. High pressure steam or low pressure steam produced

in this system can be used directly, or converted to electricity in a condensing turbine/generator unit (high pressure), or utilized in noncondensing turbines to displace electric motors used in the plant (low pressure). Thus, the float chamber system provides four energy utilization alternatives or options.

The second heat transfer concept is a contact tower, similar in design to technology used by the fiber insulation industry. In this concept, molten slag is dispersed into fine particles by a spinning disk and fiberized by compressed air. As the fibers fall down through the tower, their heat content is transferred to a cocurrently moving air stream. This hot air can be used directly or converted to high or low pressure steam in a heat recovery boiler. The steam utilization alternatives are the same as those described above. Hence, the contact tower offers five energy recovery alternatives and options.

Both the float chamber and contact tower heat recovery concepts are considered technically feasible and environmentally sound. The operating reliability of both systems would have to be demonstrated, however, because both systems differ to some degree from existing technology.

From an energy efficient standpoint, the float chamber is superior to the contact tower. The net energy recovery efficiency of steam generation via the float chamber is 78% compared to 9% for the contact tower. This differential is attributed to the large energy input needed to disperse and fiberize the slag in the contact tower. Electrical generating efficiencies are 22% for the float chamber and 0%^a for the contact tower. Electric motor displacement is an even less efficient alternative, whereby the contact tower consumes more energy than it produces, and the float chamber efficiency is reduced to 6%.

^aThe system consumes approximately the same amount of energy as it produces.

Based on energy credits alone, seven of the nine heat recovery options operate at a net economic loss. This conclusion is based on typical energy prices or generation costs incurred by the phosphorus industry -- \$0.018/kWh for electric power, \$3.50/1,000 lb for high pressure (600 psig) steam, and \$3.00/1,000 lb for low pressure (250 psig) steam. Two recovery options in which the float chamber is used to generate steam are marginally profitable at current steam generation costs. A return-on-investment of 9% can be realized for high pressure steam production. However, the phosphorus industry has very little, if any, use for large quantities of steam; only those options which produce electricity or displace electrical requirements are of practical interest.

Electricity generation via the float chamber is the most viable of all alternatives producing or displacing electric energy. With a power cost of \$0.03/kWh, energy revenues from this option are sufficient to cover operating costs. However, prices approaching \$0.05/kWh are required for a return-on-investment in excess of 10%. Thus, based on energy credits alone, slag waste heat recovery in the elemental phosphorus industry is not economical at this time.

Both the float chamber and contact tower produce a slag byproduct. The contact tower converts slag to insulation-quality fibers, a readily marketable commodity. The float chamber produces slag sheets, which could be cut to specification and sold as a substitute for ornamental stone or brick. Preliminary evaluations indicate that additional revenue from byproduct sales could make waste heat recovery profitable.

This analysis does not account for additional costs to finish, package, or market the products. A detailed marketing analysis would be required to establish price-demand relationships and to evaluate the potential for market saturation. Certainly, only a

fraction of the slag generated by the phosphorus industry could be sold without flooding current markets.

If byproduct credit is, in fact, sufficient to make heat recovery profitable, the impact on the phosphorus industry would be quite significant. Industry-wide application of the float chamber for electricity generation would reduce power generation capacity requirements by approximately 70 MW. Implementation of the contact tower for electricity generation would not save energy directly, but it would offer a means to produce insulation in a much more energy-efficient manner than that currently employed. Attendant cost savings could result in lower insulation prices, thus stimulating demand and leading to indirect energy savings on a national scale. But it is overly presumptuous at this time to assume that phosphorus producers would consider entering the insulation market.

The iron and steel industry presents a much larger market for slag waste heat recovery technology than the phosphorus industry. Because iron slag is similar in composition to phosphorus slag, energy recovery via both the float chamber and contact tower concepts should be applicable to this industry from a technical standpoint. Moreover, the economics of energy recovery are somewhat more attractive due to higher electricity prices paid by iron and steel companies. In some cases, the float chamber generating electricity may be economically attractive at current electricity prices. In general, however, revenues from energy recovery will not be sufficient to yield a reasonable return-on-investment at current electricity prices. As in the phosphorus industry, the profitability of waste heat recovery is dependent on production of a salable slag byproduct.

Analysis of the potential for slag waste heat recovery in the copper industry is complicated by the fact that the industry is currently undergoing changes in smelting technology. Although

electricity is generally the most attractive energy product, some processes may be able to utilize thermal energy to preheat furnace feed and/or combustion air. Slag end use also varies from one process to another.

In cases where the slag is recycled for residual copper recovery, the float chamber or contact tower heat recovery may create material handling problems. In other processes (e.g., the Normanda process, where slag is cooled over a 2-day period and sent to a flotation process for residual copper recovery), slag waste heat recovery may enhance throughput and quality control via rapid, yet controlled cooling. These uncertainties, coupled with the fact that copper slag is approximately twice as dense as phosphorus slag, warrant industry-specific testing to verify the technical feasibility and practicality of slag waste heat recovery.

Wet-bottom coal-fired utility boilers, which are being phased out of operation in the United States, do not represent a significant market for slag waste heat recovery technology. Recoverable energy is less than 1% of that contained in the coal consumed; even limited heat recovery is rendered technically unfeasible by the rheological properties and handling problems associated with this type of slag.

It may be concluded from this study that slag waste heat recovery via the float chamber process is a potential means for large-scale energy conservation in the phosphorus, iron and steel, and (possibly) copper industries. Pilot-scale tests are warranted to verify the technical feasibility of this process and to develop more accurate cost data. An extensive marketing analysis is also recommended to determine the salability of slag sheets, since byproduct credit is required to make the process economical at current energy prices. These studies will probably require government funding (at least initially) due to the uncertainties and risks inherent in this technology.

1. INTRODUCTION

Conservation is an essential part of this country's energy policy. Some 80×10^{12} Btu/yr^a of thermal energy (equivalent to 35,000 bpd of petroleum or about $3.0 \text{ GW}_{\text{thermal}}$) are contained in molten slags produced by pig iron blast furnaces (56×10^{12} Btu/yr), copper reverberatory smelting furnaces (9×10^{12} Btu/yr), elemental phosphorus electric furnaces (9×10^{12} Btu/yr), and coal-fired slagging boilers (7×10^{12} Btu/yr). Slags are tapped from furnaces at temperatures above $1,093^{\circ}\text{C}$ ($2,000^{\circ}\text{F}$) and represent a potential source for high grade thermal energy recovery. Because of the difficulties associated with handling slags in the molten phase, industry in general does not reclaim the slag waste heat. Typically, slag is dumped in a nearby area where it cools (it may be water-quenched) and its heat is dissipated. Upon cooling, slag is dug out and used as construction material (road ballast, aggregate, railroad ballast, etc.).

The purpose of this study was to investigate the technical, environmental, economic, and commercialization feasibility of phosphorus slag waste heat recovery via two concepts: (1) a float chamber (cooling the slag on a molten metal support); and (2) a contact tower (cooling the dispersed slag in a stream of air).

Specifically, the program had four objectives: (1) to establish the current use of slag waste heat in the elemental phosphorus industry; (2) to develop technical, economic, and environmental

^aRefer to conversion tables in Appendix N for conversion of English to metric units.

aspects of slag waste heat recovery and utilization in the elemental phosphorus industry based on two proposed concepts; (3) to investigate the expected impact of slag waste heat recovery, conversion, and end-use application on the elemental phosphorus industry; and (4) to propose an approach for applying such slag waste heat recovery technology in the elemental phosphorus industry with technical transfer potential to the iron and steel industry, copper industry, and coal-fired slagging boilers.

Section 2 summarizes the results in four subsections corresponding to the four program objectives.

2. SLAG WASTE HEAT RECOVERY IN THE ELEMENTAL PHOSPHORUS INDUSTRY

2.1 CURRENT USE OF SLAG HEAT

Because of the relative difficulties in handling molten slag, the elemental phosphorus industry today does not reclaim the heat (9×10^{12} Btu/yr) contained in phosphorus furnace slag. This was confirmed by industry survey. Undoubtedly, with increasing cost and reduced availability of energy, many plants will periodically reevaluate the effect of energy costs on overall production. This should ultimately stimulate higher interest in areas related to energy conservation, including recovery of energy from molten slags. However, at present this is not the case, and government support in this area appears warranted.

2.2 TECHNICAL, ENVIRONMENTAL, AND ECONOMIC FEASIBILITIES

In order to evaluate the technical, environmental, and economic feasibility of slag waste heat recovery, the elemental phosphorus industry was selected for case study. The results developed for this industry were also to form a basis for examining the applicability of slag waste heat recovery technology to the iron and steel industry, copper industry, and coal-fired slagging boilers.

A brief description of the elemental phosphorus production process is included in Appendix A, Section 1. Table A-1 in Appendix A lists the location, annual capacity, and waste heat recovery potential for the nine elemental phosphorus plants in the United States. Three of these plants (FMC Corporation's Pocatello, Idaho, plant and Monsanto Company's Columbia, Tennessee, and

Soda Springs, Idaho, plants) account for two-thirds of the elemental phosphorus production and/or estimated waste heat recovery potential.

The annual capacity average of these three plants was arbitrarily chosen to establish the slag and heat rates for the slag waste heat recovery options evaluated herein. Based on an 8,000-hr/yr operation, each of the slag waste heat recovery options was designed to recover heat from 33.9 kg/s (269,000 lb/hr) of molten slag at 1,482°C (2,700°F).

2.2.1 Waste Heat Recovery Options

Molten slag from phosphorus furnace(s) is delivered to the slag waste heat recovery facility via steel pots and collected in a surge vessel. Depending on the concept selected, molten slag from the surge vessel is supplied at a constant rate either to a float chamber or to a contact tower where heat is extracted from the slag and converted into steam or hot air, respectively. In the float chamber, steam tubes above and beneath a large pool of molten tin extract heat from the thin sheet of slag formed, supported, and transported on the tin surface. In the contact tower, dispersed small molten slag particles produced by a spinning disk give up heat as they fall in a stream of air. The hot air from the tower can be used either to supply process heat or to generate steam in a heat recovery boiler.

Utilization of steam may take several forms. In addition to direct use for process or space heating, the steam may be converted to electricity in a condensing turbine/electricity generator unit or used to drive plant prime movers by displacing electric motors with noncondensing steam turbine drives.

Each of these alternative energy utilization schemes involves selection of one of the slag waste heat recovery options

illustrated in Figure 1. The options are comprised of several operating steps that can be classified according to the recovery concept (float chamber or contact tower) and the form of recovered energy (hot air, steam, electricity). Altogether, nine options were conceptualized:

1. Float chamber/high pressure steam
2. Float chamber/low pressure steam
3. Float chamber/condensing turbine/electricity
4. Float chamber/noncondensing turbines/electric motor displacement
5. Contact tower/hot air
6. Contact tower/high pressure steam
7. Contact tower/low pressure steam
8. Contact tower/condensing turbine/electricity
9. Contact tower/noncondensing turbines/electric motor displacement

Separate discussions of individual operating steps are presented in Appendices E, G, I, J, and K.

2.2.2 Technical and Environmental Feasibilities of the Slag Waste Heat Recovery Options

No fundamental technical or environmental obstacles were identified which would proscribe implementation of the waste heat recovery options considered in this study. However, demonstration is required in order to resolve a number of technical uncertainties associated with the design and operation of these options.

The use of steel pots to handle molten slag is practiced in the iron and steel industry and is finding its way into the elemental phosphorus industry. Since this method is being used commercially, its technical feasibility is well demonstrated. Environmental

Figure 1. Slag waste heat recovery options and systems can be classified by the energy forms produced (circled).

consequences of using slag pots are judged no worse than those of any other slag handling practice. In fact, industrial users of this method claim that it is environmentally superior to any other currently practiced slag handling technique. However, no experimental data are available for comparing the environmental implications of different slag handling methods (see Appendix C).

The surge vessel is used as a transfer vessel for slag. While its demonstration is desirable, no major technical or environmental problems are anticipated for its use (see Appendix D).

Both the float chamber and the contact tower represent technologies which, in their original form, are fully demonstrated and practiced by other industries.

The float chamber process is used by the glass industry to manufacture flat glass; present calculations indicate no major size difference between the float chamber for slag waste heat recovery and that for flat-glass manufacture (see Appendices E and F). The float chamber proposed for use in the elemental phosphorus industry is an elongated rectangular structure, 55.5 m long, 7.3 m wide, and 1.2 m high, with a tin inventory of 20.6 m³ (i.e., the pool of molten metal is 5.1 cm deep).

Very few engineering and design data on commercial float chambers are available through the open literature. However, contacts with glass industry representatives suggested significant anticipated differences between the float chamber as used for flat-glass manufacture and the float chamber as used for heat recovery from slag. The former produces a marketable product and involves precision and product quality control procedures matched by no other heavy industry. The latter is to handle a byproduct waste; hence, the demands on precision and quality control should be considerably more relaxed. Also, while the former application

consumes heat in order to obtain desired product quality, the latter is designed to extract heat from slag and from the float chamber. This is accomplished by radiant heat transfer to 258 overhead steam pipes and by heat conduction to 212 submerged steam pipes, encased in graphite blocks, which run across the width of the float chamber.

Consequently, several aspects of the float chamber in its waste heat recovery application require verification and demonstration. These include: (1) formation of slag into a thin uniform sheet; (2) adequate performance of the heat transfer surfaces, particularly the graphite-encased steam pipes; (3) adequate sealing of the float chamber to minimize heat loss and consumption of the reducing atmosphere; and (4) achievement of an acceptable tin loss rate.

No serious environmental implications are expected to result from the float chamber. Past assessment of the environmental impact of flat-glass manufacturing plants concluded that there are no appreciable air or water emissions from the float chamber itself. Based on material balance, possible tin losses to the atmosphere were estimated to be less than 0.001 g of tin per kg of glass melted. Water is used for cooling purposes only.^{1,2} Some tin will be carried over with the slag, but this is not expected to result in any adverse environmental effects.

The contact tower concept is used by the insulation industry to manufacture fiberglass and/or mineral wool insulation. This indicates the basic technical feasibility of the concept. The tower designed to recover waste heat from slag is a hollow concrete cylinder, 5.5 m in diameter and 15 m high. Molten slag is dispersed by a spinning disk and fiberized by compressed air emanating from nozzles around the periphery of the disk. The fiberized slag falls by gravity while exchanging heat with a cocurrent air stream.

The open literature contains very little information on the design and operation of towers used to produce fiberglass and/or mineral wool insulation. Conceptually, the contact tower could operate in two different modes: (1) slag insulation production mode and (2) slag heat recovery mode. While the two operating modes are conceptually similar, they may differ significantly in their energy consumption.

The insulation production mode is energy intensive. Large amounts of energy in the form of compressed air are consumed to produce insulation of desired quality in the slag fiberization step. Commercial insulation production towers operate in this mode. The slag heat recovery mode places no emphasis on quality requirements of the slag product. Consequently, this mode can be optimized for maximum energy recovery.

An objective of this study was to design a tower that would operate in the heat recovery mode and establish its technical, environmental, and economic feasibility. This objective was met with limited success due to insufficient data on slag fiberization energy requirements as a function of the degree to which the slag is fiberized. Since heat transfer in the contact tower can be facilitated with fibers of larger diameters than those needed to produce insulation, the contact tower could operate with a lesser degree of slag fiberization and, hence, lower energy consumption than the commercial insulation towers.

To determine the energy requirements for the contact tower operating in the heat recovery mode, an extensive experimental program is necessary. Such a program was beyond the scope of this study.

Because of the insufficient data on energy requirements for towers operating in the heat recovery mode, the slag waste heat recovery options presented in this report and using the contact

tower concept are based on energy requirements characteristic of the insulation production mode. However, it is estimated that the slag waste heat recovery mode may be 4 to 28 times less energy intensive than the slag insulation mode (see Appendices G and H). Such a reduction would significantly improve the economics as well as the energy recovery potential of all slag waste heat recovery options based on the contact tower concept.

Several aspects of the contact tower process for waste heat recovery require verification and demonstration. These include: (1) fiberization of slag with energy requirements lower than those needed for insulation production; and (2) achievement of adequate heat transfer from fibers of large diameters in towers of practical dimensions.

Environmental assessment of the fiberglass manufacturing industry indicates particulate and hydrocarbon emissions from the fiber forming operation.³ In general, the contact tower produces particles that are relatively large and, therefore, not difficult to control. The hydrocarbon emissions are due to an organic binder sprayed into the fiber forming operation. Use of the contact tower for heat recovery only (as opposed to insulation production) would eliminate the need for this binder and the consequent hydrocarbon emissions. The contact tower is expected to generate no water pollution. Disposal and utilization alternatives for slag fibers need to be investigated. Here, no adverse environmental effects are expected.

The remaining operating steps involve heat recovery boilers, condensing turbine/electricity generator combinations, and noncondensing turbines. All these items represent standard technologies with well-demonstrated technical feasibility (see Appendices I, J, and K). The environmental effects include thermal pollution and increased water usage. These two effects, however, are

judged no worse than those resulting from present slag handling practices, including slag water-quenching.

2.2.3 Systems Economics

2.2.3.1 Energy Recovery Credits, Costs, and Profitability Estimates--

Waste heat from slag may be recovered in several forms including hot air, steam, and electricity. The energy forms and amounts produced by each option are summarized in Table 1. The values of these energy forms, calculated using an assumed price for each one, are included in Table 1.

Capital and operating costs for the nine heat recovery options are summarized in Table 2. Capital costs for each option component include materials and purchased equipment costs, fabrication and field installation costs, indirect construction expenses, and allowances for startup and working capital. These costs are detailed in the appendices to this report.

The operating cost estimates include all direct and fixed charges associated with the various unit operations, except for corporate investment charges. This cost is calculated separately for each of the nine energy recovery options and presented as part of the profitability summary in Table 2. The operating costs do not include charges/credits for slag disposal or utilization. This topic is treated separately in the following subsection. Also, the operating costs are based on assumed utility costs listed in Table 1.

Profitability is defined in terms of return on investment (ROI), as shown below.

$$\text{ROI (\%)} = \frac{\text{Net income after taxes}}{\text{Capital cost}} (100)$$

TABLE 1. ENERGY RECOVERY CREDITS

Option	Technology	Product	Unit price	Energy credit, \$
1	Float chamber	Steam (146,000 lb/hr @ 600 psig, 750°F)	\$3.50/1,000 lb ^a	4,088,000
2	Float chamber	Steam (159,000 lb/hr @ 250 psig, 450°F)	\$3.00/1,000 lb ^a	3,816,000
3	Float chamber	Electricity (Turbo-generator @ 15.5 MW 1.24 x 10 ⁸ kWh/yr)	\$0.018/kWh ^b	2,232,000
4	Float chamber	Electricity (Displacement @ 5.0 MW 4.00 x 10 ⁷ kWh/yr)	\$0.018/kWh ^b	720,000
5	Contact tower	Hot air (468,000 lb/hr @ 1,300°F)	\$0.64/1,000 lb ^c	2,395,000
6	Contact tower	Steam (89,800 lb/hr @ 600 psig, 750°F)	\$3.50/1,000 lb ^a	2,514,000
7	Contact tower	Steam (96,300 lb/hr @ 250 psig, 450°F)	\$3.00/1,000 lb ^a	2,311,000
8	Contact tower	Electricity (Turbo-generator @ 9.4 MW 7.52 x 10 ⁷ kWh/yr)	\$0.018/kWh ^b	1,354,000
9	Contact tower	Electricity (Displacement @ 3.0 MW 2.39 x 10 ⁷ kWh/yr)	\$0.018/kWh ^b	430,000

^aThe \$3.00/1,000-lb price may be considered representative of the lower end of the current steam price range. More than 80% of steam cost is for fuel, and many plants pay \$4.00/1,000-lb or more for industrial steam. The steam price is likely to increase and reach \$6.00 levels by 1980.

^bWhile prices of electricity range considerably higher, the price of 18 mills/kWh represents a lower value currently paid by large power consumers. Industry Week listed the cost of industrial electricity in the East North Central region at 33 mills/kWh for 200-MW/hr quantities.⁴

^cAssumes \$1.60/1,000-ft³ natural gas price and 80% boiler efficiency. Natural gas price of \$2.00/1,000-ft³ is charged by some utilities to industrial customers.

TABLE 2. SLAG WASTE HEAT RECOVERY SYSTEMS,
SUMMARY OF COSTS AND PROFITABILITY^a

Item	Option 1	Option 2	Option 3	Option 4
Capital Cost (\$000)				
Slag handling	1,400	1,400	1,400	1,400
Surge vessel	170	170	170	170
Float chamber	4,900	4,900	4,900	4,900
Condensing turbine	--	--	3,300	--
Noncondensing turbines	--	--	--	780
Total	6,470	6,470	9,770	7,250
Operating Cost (\$000) ^b				
Slag handling	1,100	1,100	1,100	1,100
Surge vessel	25	25	25	25
Float chamber	1,500	1,500	1,500	1,500
Condensing turbine	--	--	610	--
Noncondensing turbines	--	--	--	180
Total	2,625	2,625	3,235	2,805
Profitability Summary (\$000)				
Gross energy credit	4,100	3,800	2,200	720
Less operating cost	2,625	2,625	3,235	2,805
Net energy credit	1,475	1,175	(1,035)	(2,085)
Less corporate investment charge @ 5% of capital	320	320	490	360
Income before taxes	1,155	855	(1,525)	(2,445)
Less income taxes @ 50%	577.5	427.5	(762.5) ^c	(1,222.5) ^c
Net income (loss)	577.5	427.5	(762.5)	(1,222.5)
ROI, %	8.9	6.6	(7.8)	(16.9)

(continued)

^aAll data based on electricity cost of \$0.018/kWh.

^bNo charges/credits for slag disposal/utilization included.

^cReflects decrease in tax on overall plant income.

TABLE 2 (continued)

Item	Option 5	Option 6	Option 7	Option 8	Option 9
Capital Cost (\$000)					
Slag handling	1,400	1,400	1,400	1,400	1,400
Surge vessel	170	170	170	170	170
Contact tower	4,000	4,000	4,000	4,000	4,000
Heat recovery boiler	--	1,900	1,700	1,900	1,700
Condensing turbine	--	--	--	2,300	--
Noncondensing turbines	--	--	--	--	440
Total	5,570	7,470	7,270	9,770	7,710
Operating Cost (\$000) ^{b,d}					
Slag handling	1,100	1,100	1,100	1,100	1,100
Surge vessel	25	25	25	25	25
Contact tower	2,200	2,200	2,200	2,200	2,200
Heat recovery boiler	--	480	480	480	480
Condensing turbine	--	--	--	460	--
Noncondensing turbines	--	--	--	--	71
Total	3,325	3,805	3,805	4,265	3,876
Profitability Summary (\$000)					
Gross energy credit	2,400	2,500	2,300	1,400	430
Less operating cost	3,325	3,805	3,805	4,265	3,876
Net energy credit	(925)	(1,305)	(1,505)	(2,865)	(3,446)
Less corporate investment charge @ 5% of capital	280	370	360	490	390
Income before taxes	(1,205)	(1,675)	(1,865)	(3,355)	(3,836)
Less income taxes @ 50%	(602.5) ^e	(837.5) ^e	(932.5) ^e	(1,677.5) ^e	(1,918) ^e
Net income (loss)	(602.5)	(837.5)	(932.5)	(1,677.5)	(1,918)
ROI, %	(10.8)	(11.2)	(12.8)	(17.2)	(24.9)

^dUtilities costs based on insulation production mode (see Appendix G).

^eReflects decrease in tax on overall plant income.

where net income after taxes equals 0.5 (gross energy credit - operating cost - corporate investment charge).

Operating costs and energy credits used in this calculation are base-year costs only; they do not reflect escalations in energy, material, or labor costs over the life of the project. Thus, the ROI estimates shown in Table 2 may be conservative since the cost of energy is expected to inflate more rapidly than that for materials or labor. However, the results should give a reasonable indication of economic feasibility, consistent with the accuracy of the design and cost data at this preliminary stage.

Figures 2, 3, and 4 show the influence of energy costs on profitability for all nine energy recovery options. The elemental phosphorus industry has very little use for steam or hot air. Thus, only options 3, 4, 8, and 9 (which produce or displace electricity) are applicable. None of these options can be justified economically at the \$0.018/kWh unit price used in Table 1 to calculate gross energy credit. However, option 3 could be economically feasible with higher electricity costs.

A unit cost of \$0.03/kWh would be sufficient for the float chamber-turbo generator, option 3, to break even. A unit price of \$0.046/kWh would yield a 10% return on investment. In many areas of the country, electric costs are now approaching (or have exceeded) the \$0.03/kWh level. Thus, even though elemental phosphorus plants are typically located in areas with inexpensive and abundant power supplies, option 3 should not be eliminated from future consideration.

Options 4, 8, and 9, on the other hand, do not appear to be economically attractive under any circumstances. An electricity cost of \$0.08/kWh would be required for the float chamber-electric motor displacement system, option 4, to break even. Options 8 and 9, based on the contact tower concept, operate at a net energy

Figure 2. Steam price has a significant influence on the ROI for steam generation options.

Figure 3. Option 3 is the most economically attractive electricity generation or displacement option.

Figure 4. Price of natural gas has a significant influence on the ROI for hot air generation.

loss. Thus, these options will never be feasible from an energy credit standpoint, so long as the contact tower is designed to produce insulation-quality fibers.

Of the nine energy recovery options under evaluation, only options 1 and 2 yield a profit at current energy prices. Even so, return on investment may not be sufficient to warrant the necessary capital expenditures in light of the technical uncertainties associated with the float chamber. However, low or high pressure steam generation could be economically attractive with a modest increase in fuel costs.

Of course, the feasibility of float chamber steam generation is contingent upon available markets for steam. The elemental phosphorus industry presents no such market in itself, but it might be possible to export the steam to nearby industries in certain

situations. An analysis of such an option (which is not universally applicable) would require an estimate of distribution costs, which are extremely site specific and beyond the scope of this program.

Based on energy credits alone, none of the heat recovery options appears to be attractive to the elemental phosphorus industry as a whole. Byproduct credits could significantly improve the economic outlook, however. For example, the contact tower heat recovery options are rendered undesirable by the large amounts of energy (and attendant costs) needed to produce insulation-quality fibers. Thus, it is inappropriate to completely eliminate these options from consideration without taking byproduct credit into account. The same may be said for the float chamber heat recovery options. Potential byproduct credits for both of these systems are discussed in the following subsection.

Capital and operating costs for slag handling and transportation also have a significant effect on the economics of waste heat recovery. Excluding slag handling costs, the float chamber generating electricity, option 3, will break even with an electricity cost of \$0.02/kWh. A unit cost of \$0.034/kWh will yield a 10% ROI. As shown in Figure 3, the corresponding breakeven and 10% ROI unit costs for option 3 with slag handling included are \$0.03/kWh and \$0.046/kWh, respectively. Thus, slag waste heat recovery may be more attractive for plants already practicing pot handling of slag. Even for these plants, however, energy recovery cannot be justified on an economic basis at current electricity costs. Byproduct credit is still necessary to make heat recovery profitable.

2.2.3.2 Potential Byproduct Credits--

Currently, the slag from phosphorus plants is dumped, allowed to

cool, dug up, and sold as aggregate for approximately \$2.50/ton.^a This translates into an annual revenue of \$0.5 million for an elemental phosphorus plant producing 33.9 kg/s (269,000 lb/hr) of slag.

The slag will change its consistency and appearance considerably once it passes through either the float chamber or the contact tower. It assumes the form of large thin sheets in the float chamber; fibrous slag is produced in the contact tower. Also in either concept the slag is force cooled; this is likely to result in a slag with a crystalline form different from that produced now. Whether current slag markets could be maintained is therefore unknown.

New slag markets may be created; for example, production of mineral wool insulation from slag appears to be a viable alternative. Industry is already using slag to produce insulation. It is estimated that 9×10^8 kg (2 billion pounds) of mineral fiber insulation was produced in the United States in 1976.^{3, b} For comparison, the elemental phosphorus industry produces about 44×10^8 kg of slag annually (9.8 billion pounds).

Undoubtedly, production of large amounts of insulation from slag could have a significant influence on insulation markets. Contact tower heat recovery is not viable for energy recovery alone, but it could be an economical means of producing mineral wool insulation. Energy recovery credits and the elimination of expenses

^a Prices may vary considerably from one location to another, depending on the market for aggregate and the availability of other materials; i.e., sand and gravel.

^b A large fraction of the present insulation market is held by fiberglass. Fiberglass is 2 to 3 times less dense than mineral wool for the same insulation characteristics, offering easier handling and lower shipping costs. While about 90% of home insulation is fiberglass, mineral wool is largely the material of choice for industrial applications.

associated with heating and melting the slag provide advantages unavailable to current insulation producers. Thus, it should be possible to produce insulation at reduced costs while using less energy. Reduced prices should stimulate use of more insulation in existing and/or new applications, affecting many areas of the U.S. economy. An increase in demand for insulation due to reduced prices was not fully explored. The total amount of slag available from slag producing industries far exceeds current insulation demands, raising the question of market saturation.

Slag sheets formed in the float chamber would represent a new product on the U.S. market. The product might be glossy, it might have an appealing appearance, and it could be colored and cut to specification. There appear to be many potential uses for this product including exterior and interior veneer for walls of buildings, flagging, roofing, flooring, etc. Due to this variety of possible applications, the marketing potential of slag sheets is difficult to evaluate without a more comprehensive study.

For the purposes of this study, slag sheets are assumed to be a viable substitute for dressed dimension stone (i.e., granite, limestone, marble, slate, etc.) and/or bricks. In 1974, approximately 6×10^8 kg (1.3 billion pounds) of dressed dimension stone were shipped or used by producers in the United States.⁵ Shipments of bricks amounted to 1.2×10^{10} kg (27 billion pounds)⁶ in the same year. Thus, it should be possible to sell large quantities of slag sheet without saturating current markets, assuming, of course, that slag in this form could be substituted for the materials listed above.

Even though large markets exist for insulation and dimension stone, it could not be expected that all slag produced by the phosphorus industry could be converted to and sold as a useful product. Only a small fraction of the slag generated would need

to be marketed to make energy recovery profitable. This is illustrated by the following analysis of heat recovery options 3 and 8. These options have been selected because they produce electricity, the only energy product of practical interest to the elemental phosphorus industry.^a

The data presented in Table 2 can be used to calculate the total revenue required for a phosphorus plant producing 33.9 kg/s (269,000 lb/hr) of slag to obtain a reasonable ROI from heat recovery option 3 or 8. By subtracting energy credits from this total, and adding \$0.5 million to reflect the loss of existing slag credits,^b the required byproduct credit is obtained. Applicable totals for options 3 and 8 are shown in Table 3, assuming 10% ROI as the criterion for profitability.

Once the necessary byproduct credit has been determined, the quantity of slag which must be sold (as insulation or dimension stone) can be established from current market prices for comparable materials. Current prices for loose wool insulation, various types of dimension stone, and brick are shown in Table 4.

Due to the preliminary nature of this study, the values presented in Table 3 are semiquantitative at best. Final product preparation, packaging, and marketing costs cannot be determined at this time and are not included in the overall cost estimate for each system. Byproduct credit requirements are optimistic in this respect.

^aIt should be noted that option 8, in which the contact tower heat recovery system is used to generate electricity, does not produce any net energy. It is included in this analysis to indicate the potential feasibility of producing mineral wool insulation directly from molten furnace slags. Thus, it may be considered an indirect energy conservation technique, since it is more energy efficient than conventional insulation producing processes.

^bHere, it is assumed that slag, in its new form, is no longer suitable as an aggregate material.

TABLE 3. BYPRODUCT CREDITS AND MARKET PENETRATION NEEDED TO
REALIZE A 10% ROI FROM HEAT RECOVERY OPTIONS 3 AND 8

	Option 3	Option 8
Total revenue required, \$	5.7×10^6	6.7×10^6
Current slag credits, \$	0.5×10^6	0.5×10^6
Less energy credits, \$	2.2×10^6	1.4×10^6
Total byproduct credit required, \$	4.0×10^6	5.8×10^6
Current market price for materials comparable to byproduct slag, \$/kg	0.07 ^a	0.29
Amount of byproduct slag which must be sold to obtain the required revenue, kg	5.7×10^7	2.0×10^7
Fraction of total slag produced which must be sold, ^b %	6	2
Percent market takeover required	9 ^c	2

^aBased on the current price for inexpensive dimension stone (e.g., sandstone).

^bBased on an annual slag production of 9.8×10^8 kg [33.9 kg/s (8,000 hr/yr)].

^cOnly the dressed dimension stone market is considered here. If brick production is also taken into account, the total market penetration is virtually nil.

TABLE 4. PRICING DATA FOR MATERIALS REPRESENTATIVE
OF POTENTIAL SLAG APPLICATIONS

Product	Price, \$/kg
Stones: ^{a, b}	
Marble	0.44
Granite	0.22
Slate	0.15
Limestone	0.09
Sandstone	0.07
Brick	0.06
Insulation:	
Fiberglass ^c	1.00
Mineral wool (loose)	0.29

^a1974 prices updated to early 1978 via Wholesale Price Index for Industrial Commodities; 1978 Index = 200; 1974 Index = 150.^{4, 7}

^bPrices are for dressed dimension stone as mined and shipped by quarries. Final processing of this material is believed to increase these prices by a considerable margin.

^c1976 price updated to early 1978 via Wholesale Price Index for Industrial Commodities; 1978 Index = 200, 1976 Index = 180.⁴

The float chamber is a much more efficient energy transfer system than the contact tower. However, the contact tower yields a slag product for which well-established markets exist. Slag sheets, on the other hand, may not be marketable at all unless their appearance and mechanical properties (e.g., fracture resistance) are competitive with those of alternative materials.

Since the slag is force cooled in the float chamber, some thermal stresses may remain in the slag sheets causing the material to be brittle. This problem could be overcome by incorporating programmed cooling into the float chamber design, but this would

reduce the energy efficiency of the process and increase the associated costs and operating complexities.

Assuming that the slag sheets are salable, a marketing study would still have to be performed to identify the price vs. quantity tradeoff which maximizes byproduct revenue. In this study, it is assumed that the slag will be priced comparably to the least expensive variety of dimension stone (sandstone). At this price, a single large phosphorus plant would have to capture 10% or more of the dimension stone market to achieve a 10% return on investment for option 3. Thus, market saturation may be a problem unless the slag sheets prove to be a viable substitute for other materials; e.g., brick and roofing shingles. The only other alternative is a higher selling price, which may or may not be feasible.

Prices for mineral wool insulation are well-established and sufficiently high to suggest the potential for market expansion at a reduced product cost. Seemingly, it would be possible for a phosphorus producer to manufacture insulation with no additional power expenditure (the energy recovered via option 8 is roughly equivalent to the energy required to produce fibers) and thus at a lower cost than current insulation producers. With this advantage, it might benefit the phosphorus producer to sell insulation below current prices. More slag would have to be sold in this case, but the market for insulation seems to be expanding and could absorb increased insulation production. Again, a detailed marketing study is needed to identify a realistic price/market size ratio.

2.3 EXPECTED IMPACT IN THE ELEMENTAL PHOSPHORUS INDUSTRY

The thermal energy available for recovery from phosphorus furnace slag is about 29% of initial furnace input energy.⁸ The amount of

energy actually recovered is dependent upon the energy transfer and conversion efficiency of the recovery option. In addition, each recovery option requires a certain amount of energy for its operation. Consideration of this operating energy along with the energy recovered yields an overall option efficiency. Both the energy recovery efficiencies and the overall option efficiencies for the nine energy recovery options are listed in Table 5.

TABLE 5. EFFICIENCIES OF SLAG WASTE HEAT RECOVERY OPTIONS

Option	Technology	Recoverable power		Energy recovery efficiency, % ^a	Operating power		System efficiency, %
		MW thermal	MW electrical		MW electrical	MW thermal ^b	
1	Float chamber	55	--	79	0.1	0.3	78
2	Float chamber	56	--	79	0.1 ^c	0.3	78
3	Float chamber	--	16	22	0.1 ^c	--	22
4	Float chamber	--	5	7	0.1 ^c	--	6
5	Contact tower	44	--	62	9.2 ^d	28	23
6	Contact tower	34	--	48	9.2 ^{d,e}	28	9
7	Contact tower	34	--	48	9.2 ^{d,e}	28	9
8	Contact tower	--	9	13	9.2 ^{d,e}	--	0
9	Contact tower	--	3	4	9.2 ^{d,e}	--	-9

^aBased on the power of 70 MW available in 33.9 kg/s of phosphorus furnace slag.

^bEquivalent thermal operating power assuming a thermal to electrical conversion efficiency of 33% at the power plant.

^cAdditional operating power associated with this option is negligible in comparison with option 1.

^dBased on insulation production mode of contact tower operation.

^eAdditional operating power associated with this option is negligible in comparison with option 1.

Relatively high option efficiencies are observed for the production of steam in the float chamber (options 1 and 2). However, there is a very low utilization potential for steam in the elemental phosphorus industry (see Appendix A, Section 2). System efficiencies are much lower for the production of electricity via the float chamber due to the relatively low efficiency associated with the small-scale conversion of thermal energy to electrical energy. Recovery efficiencies are low for all options involving the contact tower, and options 8 and 9, which produce electricity, do not recover any net energy.

Only option 3 is capable of producing electricity with a reasonably high overall efficiency. Hence, this is the only viable option (of those studied) for the elemental phosphorus industry. Potential energy savings have been estimated for the nine elemental phosphorus plants in the United States based on the system efficiency for option 3. The savings shown in Table 6 amount to approximately 71 MW of electric generation capacity.

The distribution of potential energy savings among phosphorus producers and power companies can be inferred from Table 7. This table gives the relative power consumption of each phosphorus producer and identifies the companies that supply the power.

2.4 COMMERCIALIZATION

2.4.1 Elemental Phosphorus Industry

Of the nine energy recovery options considered in this study, option 3, electricity production via the float chamber, is the only viable system for the elemental phosphorus industry. Suboption 3b (including cost of slag handling and transportation) is not economically feasible at current electricity prices paid by elemental phosphorus plants (approximately \$0.02/kWh). The break-even price of electricity for this option is \$0.03/kWh, and a price of approximately \$0.05/kWh is required to yield a reasonable return on investment. If the cost of slag handling and transportation equipment can be justified on other (e.g., environmental or industrial hygiene) grounds, then the energy recovery becomes marginally feasible. The break-even electricity price for this case (suboption 3a) is about \$0.02/kWh, and a reasonable return on investment is realized at a price between \$0.03/kWh and \$0.04/kWh.

Thus, it is unlikely that the phosphorus industry will sponsor development of this technology as an energy conservation measure.

TABLE 6. ESTIMATES OF POTENTIAL ELECTRICITY SAVINGS
FOR THE ELEMENTAL PHOSPHORUS INDUSTRY,
ELECTRICITY GENERATION EFFICIENCY OF 22%

Plant	Estimated annual elemental phosphorus capacity		Estimated thermal power content of slag ^a		Estimated potential electricity savings, MW
	Gg	Tons	MW	10 ⁶ Btu/hr	
Electro-Phos	18	20,000	12	40	2.6
FMC	122	135,000	79	268	17
Mobil	5	6,000	3	12	0.8
Monsanto - Columbia	118	130,000	76	258	17
- Soda Springs	89	98,000	57	194	13
Occidental	50	55,000	32	109	7.0
Stauffer - Mt. Pleasant	41	45,000	26	89	5.8
- Silver Bow	36	40,000	23	79	5.1
- Tarpon Springs	21	23,000	13	46	2.9
Total	500	552,000	321	1,095	71

^aAssumes 8.9 kg of slag per kg of elemental phosphorus (17,800 lb/ton),⁹ slag specific heat of 1.09 kJ/kg·K (0.26 Btu/lb·°F), sensible heat from 1,482°C (2,700°F) to 21°C (70°F), slag heat of fusion 481 kJ/kg (208 Btu/lb), and 8,000 hr/yr operation. Note that all variables entering the calculation of the estimated thermal power content for slag waste heat recovery involve some degree of uncertainty: plant annual production may vary from year to year depending on the elemental phosphorus market conditions and necessary furnace shutdowns for maintenance; the amount of slag per unit weight of elemental phosphorus varies with the quality of ore [values between 8.5 kg/kg and 9.5 kg/kg (17,000 lb/ton and 19,000 lb/ton) have been reported] and the phosphorus furnace efficiency; the slag heat capacity values reported in the literature range between 0.71 kJ/kg·K and 1.38 kJ/kg·K (0.17 Btu/lb·°F and 0.33 Btu/lb·°F); slag tapping temperature varies with the furnace operation and the furnace feed composition changes--the measurement of slag tapping temperature is not standardized, making the results dependent on the measurement technique used as well as the specific point of measurement [values between 1,427°C (2,600°F) and 1,538°C (2,800°F) are commonly accepted as typical slag tapping temperatures]; heat of fusion for CaSiO₃ was assumed and is expected to vary with slag composition--no data for phosphorus slags are available (heat of fusion may only be realized if the slag undergoes a phase change and crystallizes upon cooling). See Appendix B for further data on slag properties.

TABLE 7. RELATIVE POWER REQUIREMENTS BY PRODUCER AND POWER SUPPLIERS FOR THE ELEMENTAL PHOSPHORUS INDUSTRY

Phosphorus producers	Source of power
	Electricity suppliers
Monsanto	TVA; Idaho, Utah Power & Light
FMC	Idaho Power & Light
Stauffer	TVA; Bonneville, Florida Power & Light
Occidental	TVA
Electro-Phos	Tampa Electric
Mobil	Tampa Electric

Distribution of power	
Producer	Percent of industry power requirement
Monsanto	41
FMC	24
Stauffer	20
Occidental	10
Electro-Phos	4
Mobil	1
Total	100

Development and commercialization of the technology by industry appears to be a possibility only if it can be demonstrated that a marketable product can be produced from the slag which would make the recovery economically attractive. Hence, it can be concluded that eventual commercialization of the float chamber for slag waste heat recovery in the phosphorus industry is dependent upon a development and demonstration program funded, at least initially, by government sources.

The contact tower process is both technically and economically infeasible as an energy conservation measure in the phosphorus industry. However, it might prove economically attractive as an insulation production process. Although no net energy would be saved at the phosphorus plant, energy would be saved indirectly either by displacing more energy-intensive insulation manufacturing operations (which require energy to heat and melt the raw

material) or by increasing the use of insulation throughout the economy by lowering the price (as a result of lower energy and raw material costs). However, only 20% to 30% of available phosphorus slag would be sufficient to supply the entire current U.S. insulation market. In addition, the assumption that phosphorus manufacturers would be interested in entering the insulation market is highly speculative.

2.4.2 Iron and Steel Industry

The pig iron industry is the single largest producer of slag with 1976 production totaling 23.6×10^9 kg (26×10^6 tons). In general, this slag is sold to slag-processing companies that process and/or distribute the slag to well-established markets. The principal uses for pig-iron slag are as aggregate (for the production of cement and concrete and for highway construction), for railroad ballast, for production of mineral wool, as a roofing material, and for agricultural applications (liming). In 1976, sales of pig-iron slag totaled \$70 million. Quantities and values of the three types (air cooled, granulated, expanded) of salable slag produced by the pig iron industry are shown in Table 8.¹⁰⁻¹² An additional 5.5×10^9 kg (6×10^6 tons) of slag were produced in 1976 by the steel industry, with a market value of \$10 million.¹²

TABLE 8. 1976 PIG IRON SLAG PRODUCTION
AND PRICING DATA¹⁰⁻¹²

Slag type	Annual production		Value		
	10 ⁹ kg	10 ⁶ tons	\$10 ⁶	\$/kg	\$/ton
Air cooled	20.7	22.9	59.8	0.0029	2.61
Granulated	1.5	1.6	3.5	0.0024	2.19
Expanded	1.4	1.5	6.6	0.0047	4.40
Total	23.6	26	69.9	- ^a	- ^a

^aNot applicable.

In spite of the considerable market value of slag, iron and steel companies generally receive little or no net income from slag sales. Revenues from slag sales are often more than offset by slag handling and transportation costs. It is nevertheless advantageous for the iron and steel companies to sell slag rather than to incur the high cost of wasting the slag via other means (e.g., landfilling). (Personal communication with Kenneth Gee, Bethlehem Steel Corp., Bethlehem, Pennsylvania, 7 July 1978.)

Potential economic benefits of waste heat recovery from pig iron slag can be estimated as follows. Blast furnaces produce 0.546 kg of slag per kg of pig iron (1,090 lb/ton); this material represents 1.23×10^9 J (1.17×10^6 Btu), or 9.0% of the heat absorbed in the furnace.¹³ The energy available in pig iron slag is therefore:

$$\frac{(26 \times 10^6)(2,000)(1.17 \times 10^6)}{(1,090)} = 5.6 \times 10^{13} \text{ Btu/yr} \\ = 1.6 \times 10^{10} \text{ kWh/yr}$$

Assuming an operating time of 8,000 hr/yr, the above quantity is 2,030 MW_{thermal}, which may be used to generate 530 kg/s (4.2×10^6 lb/hr) of high pressure steam^a with an annual value of $\$120 \times 10^6$ ^b, or 3.6×10^9 kWh (448 MW_{el}) with an annual value of $\$65 \times 10^6$ ^c. This amount of electricity represents approximately ten percent of the electricity purchased annually by the iron and steel industry.

Steel plants have little use for process steam. However, recovered thermal energy could be used to preheat blast furnace air, thereby reducing coal consumption. Recovered energy could also be used to effect a significant reduction in overall plant

^a4.2 MPa (600 psig), 400°C (750°F), condensate recycle.

^bPrice of steam assumed to be \$0.0077/kg (\$3.50/1,000 lb).

^cPrice of electricity assumed to be \$0.018/kWh.

electricity purchases. Since iron and steel plants generally pay higher prices for electricity than do elemental phosphorus plants, the energy recovery options which produce electricity should be economically more attractive for the iron and steel industry. In fact, for those plants already possessing suitable slag handling facilities and that are located in areas of high electricity prices the float chamber process with electricity production (option 3a) may be economically attractive at current electricity prices. In general, however, electricity prices paid by iron and steel companies are too low to yield a reasonable return on investment, especially when new slag handling equipment is required for energy recovery (approximately \$0.50/kWh required for option 3b).

Technical uncertainties regarding transfer of slag energy recovery technology from the elemental phosphorus industry to the iron and steel industry are associated with differences in the following factors:

- Slag composition and corrosiveness
- Slag tapping frequency
- Slag flow rates

Differences in slag composition appear to be minor (Table 9), and should not have a significant effect on energy recovery. In particular, the same materials of construction should be applicable in both industries. Slag densities are similar, 2.5 for iron slag compared to 2.9 for phosphorus slag.

The difference in slag tapping frequency may pose a more serious problem. Blast furnaces are tapped every 3 to 3.5 hours compared to every 55 min for phosphorus furnaces. Thus, a much longer residence time in the surge vessel would be required for iron slags. This could result in greater heat losses and the need for auxiliary heating to prevent solidification of the slag.

TABLE 9. TYPICAL SLAG COMPOSITIONS^{10,11,14-17}
(weight percent)

Constituent	Slag			
	Phosphorus	Iron	Copper	Boilera
SiO ₂	40	33	35 to 40	45 to 50
CaO	48	45	6 to 12	1 to 5
Al ₂ O ₃	5	15	2 to 5	20 to 27 ^b
Fe (total)	0.2	0.1	37 to 51	15 to 20 ^b
MgO	0.5	2		1 to 2
CaS		4		
K ₂ O	1			1 to 3
C				4
Cu			0.4 to 3	
F	3			
S			0.9 to 1.2	
Zn			2.5 to 5	

^aBased on personal communication with Robert John, Ohio Edison, Niles, Ohio, 1978.

^bPercentages are for Fe₂O₃.

Slag flow rates from large blast furnaces are only 20% to 30% of those from the largest phosphorus furnaces. Hence, there may be a diseconomy of scale associated with technology transfer from the phosphorus industry.

In summary, the economics of slag waste heat recovery in the iron and steel industry appear to be more favorable than in the elemental phosphorus industry due to higher electricity prices paid by iron and steel companies. In general, however, the economics are not sufficiently favorable to warrant technology development by the industry without the incentive of a potentially significant byproduct market. There are no fundamental technological barriers to transfer of slag waste heat recovery technology from the phosphorus industry to the iron and steel industry. However, differences between the two industries are sufficiently great that additional work would be required to demonstrate the applicability to the iron and steel industry of technology developed for the phosphorus industry.

2.4.3 Copper Industry

Analysis of the potential for slag waste heat recovery in the copper industry is complicated by the fact that the industry is currently undergoing changes in smelting technology. These changes are best exemplified by penetration of the domestic market by the Noranda continuous smelting process. Other modifications under development include flash smelting, the Mitsubishi process, and the Worcra process.

The available energy in liquid copper slag can be estimated as follows. The specific heat and heat of crystallization of copper slag are assumed to be equal to those of phosphorus slag. The annual amount of slag generated by the copper industry is approximately 2.7 Gg (6.0×10^6 lb).¹⁸ A reduction in temperature from that at the smelter outlet ($1,150^\circ\text{C}$)¹⁴ to ambient levels (20°C) yields approximately 9.3×10^{15} J (8.8×10^{12} Btu) as the annual amount of energy available from copper slag. Note that this is less than one-sixth of the waste heat available in pig iron slag.

The most attractive energy recovery options for the copper industry are those that produce electricity, which could be used by all types of copper smelter furnaces (reverberatory, flash, electric, Noranda). However, the economics of these options have been shown to be marginal, at best, in the phosphorus industry.^a Reverberatory furnaces could possibly utilize thermal energy for preheat of furnace feed and/or combustion air. These furnaces use natural gas when it is available, but are easily converted to fuel oil firing. Thus, preheating would basically save fuel oil.

^aHigher electricity prices in the copper industry may improve the economics somewhat, but not enough to make these options economically attractive.

In the Noranda process, slag is cooled slowly over a 2-day period to form crystal slag. This crystal slag is subsequently sent to a flotation process to recover residual copper in the slag. Thus, an energy recovery system (such as the float chamber) in which the slag could be cooled in a controlled manner to produce crystals might be very attractive. Cooling the slag in a time period much shorter than 2 days could substantially reduce process capital costs while recovering energy. However, the only end use for thermal energy in the Noranda process appears to be a light drying of the furnace feed.

Technical uncertainties regarding transfer of slag energy recovery technology from the elemental phosphorus industry to the copper industry are associated with differences in the following factors:

- Slag composition and properties
- Slag tapping frequency
- End use of slag

From Table 9, it can be seen that the composition of copper slag differs substantially from that of phosphorus slag. Of particular concern is the high iron content of copper slag, which increases the density to roughly twice that of phosphorus slag. The higher density of copper slag has important implications for both the float chamber and contact tower. For example, the length of the float chamber required to effect a given degree of slag cooling is approximately proportional to slag density (see Appendix E). The higher density would increase the equilibrium thickness of the slag sheet formed on the tin surface by only about 5%. However, if the surface tension of copper slag is also higher than that of phosphorus slag, the equilibrium slag thickness would be increased further, thereby reducing the rate of heat transfer and increasing the required size of the float chamber. It would probably be necessary to use a greater tin depth in the float chamber due to the greater displacement of tin by the heavier copper slag. All

of these factors taken together could substantially increase the capital cost of the float chamber and energy recovery options based on this concept.

The difference in composition of copper and phosphorus slags also raises questions concerning the compatibility of the former with tin. Entrainment of tin in copper slag would have to be determined, since tin loss represents a substantial operating cost. There are also safety considerations involved in handling slag with a high iron content since reaction of iron with water or water vapor can result in an explosion (metal shot).

Copper smelting furnaces are tapped approximately every 4 hours, similar to the tapping frequency for iron blast furnaces. Hence, the problem of slag storage in the surge vessel is similar to that discussed previously for the iron and steel industry.

In copper smelting, the slag may be recycled in the process to recover residual copper contained in the slag. Therefore, any system to recover energy from the slag must produce cooled slag in a form which is compatible with process requirements. It is uncertain whether either the float chamber or contact tower meets this requirement.

In summary, the economics of slag waste heat recovery in the copper industry do not appear to be substantially better than those for the phosphorus industry. A possible exception is a potential capital cost advantage in the Noranda process. In addition there are substantial technical uncertainties associated with the transfer of waste heat recovery technology from the phosphorus industry to the copper industry. The situation is further complicated by the technological changes currently taking place in the copper industry.

2.4.4 Wet-Bottom Coal-Fired Utility Boilers

According to Federal Power Commission statistics, 175 wet-bottom utility boilers in the United States consumed approximately 3,530 Gg/yr (3.88×10^7 tons/yr) of pulverized anthracite and lignite coal in 1972.¹⁹ These figures represented 17.2% of the total number of coal-fired utility boilers and 13.5% of coal consumption by the same. Data for 1971 indicate that 4.5 Gg/yr (5×10^6 tons/yr) of slag are tapped from wet-bottom boilers.¹⁹ This slag is used as fill material for roads and construction sites, as a replacement for cement in concrete products, and for other purposes.¹⁷

The energy conservation potential of transferring phosphorus slag heat recovery technology to wet-bottom utility boilers can be calculated as follows. Assume that the boiler tapping temperature is equal to a typical ash fusion temperature, 1,150°C (2,100°F), and the specific heat and heat of crystallization of boiler slag are equal to those of phosphorus slag. The available energy is therefore:

$$(5 \times 10^6)(2,000)[(0.26)(2,100 - 70) + 207] = 7 \times 10^{12} \text{ Btu/yr}$$

This energy is equivalent to approximately 260 Gg/yr (2.9×10^5 tons/yr) of anthracite coal and about \$7 million (assuming a heating value of 26×10^6 Btu/ton and a coal price of \$25/ton). This amount is less than 1% of the coal burned in all wet-bottom utility boilers.

From the discussion above, wet-bottom boilers, which are being phased out of operation in the United States, do not appear to be good candidates for slag heat recovery. Additional detriments to implementation of the technology proposed herein exist. For example, slag exits the boilers either having a very high viscosity or undergoing a phase change to solid chunks, creating a difficult

handling problem. Considerable amounts of energy would be required to remelt the slag before processing in a recovery system, resulting in an obvious penalty to overall efficiency.

REFERENCES

1. Reznik, R. B. Source Assessment: Flat Glass Manufacturing Plants. EPA-600/2-76-032b, U.S. Environmental Protection Agency, Research Triangle Park, North Carolina, March 1976. 159 pp.
2. Train, R. E., R. Strelow, and A. Cywin. Development Document for Effluent Limitations Guidelines and New Source Performance Standards for the Flat Glass Segment of the Glass Manufacturing Point Source Category. U.S. Environmental Protection Agency, Washington, D.C., January 1974.
3. Screening Study to Determine Need for Standards of Performance for New Sources in the Fiber Task Order 23, U.S. Environmental Protection Agency, Research Triangle Park, North Carolina, December 1976.
4. Business Trends. Industry Week, 196(5):106, 1978.
5. Drake, H. L. Stone. In: Minerals Yearbook 1974, Vol. I, U.S. Bureau of Mines, Washington, D.C., 1976. pp. 1215-1234.
6. Statistical Abstract of the United States 1977, 98th Annual Edition, U.S. Department of Commerce, Bureau of the Census, Washington, D.C., 1977. p. 818.
7. Business Trends. Industry Week, 191(2):138, 1976.
8. Hurst, T. L. Manufacture of Elemental Phosphorus and Its Major Inorganic Derivatives. In: Phosphorus and Its Compounds, Vol. 2, R. Van Wazer, ed. Interscience, New York, New York (1961). pp. 1160-1209.
9. Van Wazer, J. R. Phosphorus and the Phosphides. In: Kirk-Othmer Encyclopedia of Chemical Technology, Second Edition, Vol. 15. John Wiley & Sons, Inc., New York, New York, 1968. pp. 276-295.
10. Personal communication with H. K. Eggleston, National Slag Association, April 1978.
11. Processed Blast-Furnace Slag, the All-Purpose Construction Aggregate, Publication NSA 178-1, National Slag Association, Alexandria, Virginia.

12. Evans, J. R. Slag - Iron and Steel. Preprint from the 1976 Bureau of Mines Minerals Yearbook. U.S. Department of the Interior, Washington, D.C., 1978.
13. Grosvenor, A. W. Basic Metallurgy, Volume I - Principles, American Society for Metals, Cleveland, Ohio, 1956.
14. Ehelich, R. P. Copper Metallurgy. Proceedings of the Extractive Metallurgy Division Symposium, The Metallurgical Society, Denver, Colorado, 1970.
15. Parker, R. H. An Introduction to Chemical Metallurgy. Pergamon Press, New York, New York, 1967.
16. Floyd, J. M. The Recovery of Copper from Liquid Slags. TMS Paper Selection, Paper No. A74-6, The Metallurgical Society of AIME, New York, New York, 1974.
17. Brackett, C. E. Production and Utilization of Ash in the United States. Information Circular 8640, U.S. Bureau of Mines, Washington, D.C., 1973.
18. Mineral Facts and Problems, 1975 Edition. Bulletin 667, U.S. Bureau of Mines, Washington, D.C., 1976.
19. Steam-Electric Plant Air and Water Quality Control Data for the Year Ended 31 December 1972. Publication FPC-5-246, Federal Power Commission, Washington, D.C., 1975.
20. Personal communication with D. C. DeWitt, Monsanto Company, St. Louis, Missouri.
21. Personal communication with H. M. Stevens, Monsanto Company, St. Louis, Missouri.
22. Kirk-Othmer Encyclopedia of Chemical Technology, Second Edition, Vol. 10. John Wiley & Sons, Inc., New York, New York, 1966. p. 598.
23. Shand, E. B. Glass Engineering Handbook. McGraw-Hill Book Company, Inc., New York, New York, 1958. pp. 23-24.
24. Personal communication with H. Bangert, Bangert Bros. Road, Inc., St. Louis, Missouri.
25. Weaver, J. B. Cost and Profitability Estimation. In: Chemical Engineers' Handbook, Fifth Edition, Chapter 25, R. H. Perry and C. H. Chilton, eds. McGraw-Hill Book Co., New York, New York, 1973.
26. Peters, M. S., and K. D. Timmerhaus. Plant Design and Economics for Chemical Engineers, Second Edition. McGraw Hill Book Co., New York, New York, 1968. 850 pp.

27. Personal communication with Plibrico Sales and Service, Dayton, Ohio.
28. Hale, E. Application of Natural Gas in the Manufacture of Float Glass. *Gas Engineering Management*, 15(2):59-67, 1975.
29. Bondarev, K. T. Production of Glass by the Float Process. *Glass Ceramics*, 29(7/8):502-506, 1972.
30. Carnock, H. The Float Glass Process. *Physics Bulletin*, 21(4):153-156, 1970.
31. Pilkington, L. A. B. The Float Glass Process. *Proceedings of the Royal Society, London*. A.314:1-25, December 1969.
32. Michalik, E. R., and G. W. Misson. Method and Apparatus Having Sealing Means and Gaseous Takeoff for Float Glass. U.S. Patent 3,615,315 (to PPG Industries, Inc.), October 26, 1971.
33. Branch, G. H. Cold End Automation in the Flat Glass Industry. *Glass Technology*, 17(1):14-20, 1976.
34. Svec, J. J. LOF Operates World's Largest Glass Furnace. *Ceramic Industry*, 103(2):30-32, 1974.
35. Mase, H., and Y. Sasagawa. Mathematical Modeling of Glass Tank Furnace. *Proceedings of International Symposium on Automatic Control in Glass*. Purdue University, West Lafayette, Indiana. September 1973.
36. Hummel, M. J. Method of Improving Quality of Flat Glass Formed in Molten Tin. U.S. Patent 3,925,052 (to PPG Industries, Inc.), December 9, 1975.
37. Lawrenson, J. Apparatus for the Manufacture of Float Glass Utilizing Buoyant Depressible Barrier Means. U.S. Patent 3,485,617 (to Pilkington Brothers, Ltd), December 23, 1969.
38. Pilkington, L. A. B. Float: An Application of Science, Analysis, and Judgement. *Glass Technology*, 12(4):76-83, 1971.
39. Kern, D. Q. *Process Heat Transfer*. McGraw-Hill Book Company, New York, New York, 1950. 871 pp.
40. McAdams, W. H. *Heat Transmission*. McGraw-Hill Book Company, New York, New York, 1954. 532 pp.
41. Carnahan, B., H. A. Luther, and J. O. Wilkes. *Applied Numerical Methods*. John Wiley & Sons, Inc., New York, New York, 1969. 604 pp.

42. Touloukian, Y. S., R. W. Powell, C. Y. Ho, and P. G. Klemens. Thermophysical Properties of Matter: The TPRC Data Series, Vol. I, Thermal Conductivity-Metallic Elements and Alloys. Plenum Publishing Corporation, New York, New York, 1970.
43. Pilkington, L. A. B., and K. Bickerstaff. Manufacture of Flat Glass. U.S. Patent 2,911,759 (to Pilkington Brothers, Ltd), November 10, 1959.
44. Svec, J. J. Float Plant a Showcase at Pilkington. Ceramic Industry, 101(6):34-36, 1973.
45. Bishop, H. S. Edge-Roll Machine in Float Glass Apparatus. U.S. Patent 3,709,673 (to PPG Industries, Inc.), January 9, 1973.
46. Holman, J. P. Heat Transfer, Third Edition. McGraw-Hill Book Company, New York, New York, 1972. 462 pp.
47. Greenler, R. J. Chamber for the Manufacture of Float Glass with a Graphite Liner of Varied Thickness. U.S. Patent 3,486,878 (to Ford Motor Company), December 30, 1969.
48. Littleton, C. T. Industrial Piping, Second Edition. McGraw-Hill Book Company, New York, New York, 1962. 349 pp.
49. Steam: Its Generation and Use. The Babcock & Wilcox Company, New York, New York, 1972.
50. The Properties of Tin. Tin Research Institute, Greenford, Middlesex, England, 1965. 55 pp.
51. Sieger, J. S. Chemical Characteristics of Float Glass Surfaces. Journal of Non-Crystalline Solids. 19:213-220, 1975.
52. Hayes, F. H. Thermodynamic Considerations in the Float Glass Process. In: Metallurgical Chemistry, Proceedings of a Symposium. HMSO, London, England, 1971. pp. 581-588.
53. Price Trends. Industry Week, 196(2):110, 1978.
54. Estimate provided by MRC pilot plant personnel.
55. Private communication, B. G. Danis, Co., Dayton, Ohio.
56. Private communication, Babcock & Wilcox Refractories Division, Cleveland, Ohio.
57. Private communication Babcock & Wilcox Refractories Division, Greenwood, Indiana.
58. Private communication, Union Carbide Corporation, Chicago, Illinois.

59. Estimate prepared by MRC machine shop personnel.
60. Private communication, Babcock & Wilcox Power Generating Division, Cleveland, Ohio.
61. Elonka, S. M., and A. L. Kahan. Standard Boiler Operator's Questions and Answers. McGraw-Hill Book Co., New York, New York 1969. 365 pp.
62. Private communication, Zurn Industries, Inc., Energy Division, Erie, Pennsylvania.
63. Private communication, G & M Corporation, Material Handling and Engineered Products, Dayton, Ohio.
64. Johnson, W., A. G. Mamalis, and H. Hunt. Small Spherical Lead Shot Forming from the Liquid, Using a Shot Tower. *Metallurgia and Metal Forming*, 43(30), 1976.
65. Ruskan, R. E. Prilling vs. Granulation for Nitrogen Fertilizer Production. *Chemical Engineering*, June 7:114-118, 1976.
66. Schweizer, von, P., B. Covelli, and F. Widmer. Dimensionierung eines Prillturmes. *Chimia*, 29, No. 2, February 1975.
67. Stein, W. A. Berechnung des Wärmeübergangs in Sprühturm. *Chemie-Ing.-Techn.*, 43 Jahrg. 1971, Nr. 21. pp. 1153-1157.
68. Bradley, J. K. Drop Size and Rates of Flow of Water and Molten Ammonium Nitrate Trough Steel Nozzles. *J. Appl. Chem. Biotechnol.*, 25:891-900, 1975.
69. Lowenstein, K. L. The Manufacturing Technology of Continuous Glass Fibers. Elsevier Scientific Publishing Company, New York, New York, 1973. 280 pp.
70. Ritchie, K. M. Mineral Wool-Rock, Slag and Glass Wool. In: *Industrial Minerals and Rocks*, Joseph L. Gillson, ed. Seeley W. Mudd Series A.I.M.E. 3rd Ed., 1960. pp. 595-604.
71. Goode, A. H., M. E. Tyrrell, and I. L. Feld. Glass Wool from Waste Glass. U.S. Department of the Interior, Bureau of Mines Report No. 7708, 1972. 16 pp.
72. Greaves-Walker, A. F., and Welch, P. A. The Development of Mineral Wool from Florida Minerals. *Engineering Progress at the University of Florida*, Vol. VII, No. 2, Bulletin Series No. 59, Gainesville, Florida, February 1953. 28 pp.
73. Chohey, N. P. New Plant Features Latest Look in Making Glass Fibers. *Chemical Engineering*, May 15, 1961. pp. 136-139.

74. Friedman, S. J., F. A. Gluckert, and W. A. Marshall, Jr. Centrifugal Disk Atomization. Chemical Engineering Progress, 48(4):181-191, 1952.
75. Marshall, W. R., Jr., and E. Seltzer. Principles of Spray Drying. Part I - Fundamentals of Spray-Dryer Operation. Chemical Engineering Progress, 46(10):501-508, 1950.
76. Marshall, W. R., Jr., and E. Seltzer. Principles of Spray Drying. Part II - Elements of Spray-Dryer Design. Chemical Engineering Progress, 46(11):575-584, 1950.
77. Adler, C. R., and W. R. Marshall, Jr. Performance of Spinning Disk Atomizers, Part I. Chemical Engineering Progress, 47(10):515-522, 1951.
78. Adler, C. R., and W. R. Marshall, Jr. Performance of Spinning Disk Atomizers, Part II. Chemical Engineering Progress, 47(12):601-608, 1951.
79. Walton, W. H., and W. C. Prewett. The Production of Sprays and Mists of Uniform Drop Size by Means of Spinning Disc Type Sprayers. The Proceedings of the Physical Society, Vol. 62, Part 6, No. 354B, 1 June 1949. pp. 341-350.
80. Rau, E., and W. H. Kibbel, Jr. Method of Preparing Fibers from By-Product Slag of a Phosphorus Furnace. U.S. Patents 3,360,592 and 3,360,593, December 26, 1967.
81. Downey, R. M. Method and Apparatus for Fiberizing Molten Material. U.S. Patent 2,646,593, July 28, 1953.
82. Porter, W. F. Apparatus for Forming Fibers. U.S. Patents 3,372,011 and 3,372,013, March 5, 1968.
83. Garrison, J. W. Apparatus for Forming Fibers. U.S. Patent 3,372,014, March 8, 1968.
84. Richardson, C. D. Apparatus for Forming Mineral Wool. U.S. Patent 2,793,395, May 28, 1957.
85. Leva, M. Fluidization. McGraw-Hill Book Co., Inc., New York, New York, 1959. pp. 179-246.
86. McAdams, W. H. Heat Transmission, Second Edition. McGraw-Hill Book Co., Inc., New York, New York, 1942. pp. 45-86.
87. Knudsen, J. G., and D. L. Katz. Fluid Dynamics and Heat Transfer. McGraw-Hill Book Co., Inc., New York, New York, 1958.

88. Heisler, M. P. Temperature Charts for Induction and Constant-Temperature Heating. Transactions of the A.S.M.E., April 1947. pp. 227-236.
89. Lapple, C. E., and C. B. Shepherd. Calculation of Particle Trajectories. Industrial and Engineering Chemistry, 32(5): 605-617, 1940.
90. Private communication, Joy Manufacturing Co., Cleveland, Ohio.
91. Gould, T. R. Refractory Fibers. In: Kirk-Othmer Encyclopedia of Chemical Technology, Second Edition, Vol. 17. pp. 285-295. H. F. Mark, J. J. McKetta, Jr., D. F. Othmer, and A. Standen, eds. John Wiley & Sons, Inc., New York, New York (1968).
92. Guthrie, K. M. Data and Techniques for Preliminary Capital Cost Estimating. Chemical Engineering, 76(6):114-142, 1969.
93. Private communication, Mason Steel Fabricating, Inc., Decatur, Illinois.
94. Private communication, First Colony Corp., Marietta, Ohio.
95. Private communication, Earl D. Creager, Inc., Dayton, Ohio.
96. Vogt Water Tube Boilers. Bulletin WTB-2, Henry Vogt Machine Company, Louisville, Kentucky.
97. Private communication with Ralph Fisher, Monsanto Company, Central Engineering Department, St. Louis, Missouri.
98. Communication from R. M. Hamill of General Electric Company, Oak Brook, Illinois, 14 March 1978.
99. Communication from Roland Gregory of the Terry Steam Turbine Company, Windsor, Connecticut, 3 March 1978.

APPENDICES

SLAG WASTE HEAT RECOVERY Technical, Environmental, Economic and Commercial Feasibilities

These appendices contain the engineering evaluations, design calculations, and cost estimates which form the basis for the nine slag waste heat recovery options in the main body of the report. As envisioned here, these options recover heat from 33.9 kg/s of phosphorus slag and involve four operating steps: slag handling/transportation, surge vessel, slag waste heat recovery process, and waste heat utilization. Background information for commercialization of these options is also included.

The authors believe that the information herein also has utility for slag producing industries in general. Depending on application, not all four steps as envisioned here may be necessary to make slag waste heat recovery useful. Other steps may be substituted or added. The final selection of the operating steps and their sequence should be determined by the technical and economic requirements of each specific application.

APPENDIX A

ELEMENTAL PHOSPHORUS PRODUCTION

Process Description and Energy Utilization Alternatives

CONTENTS

	<u>Page</u>
1. Process Description	47
2. Energy Utilization Alternatives	49
2.1 Heated Gas (Air) Utilization	50
2.1.1 Gas Preheating.	50
2.1.2 Ore Drying.	50
2.2 Steam Utilization.	51
2.2.1 Electric Motor Displacement	51
2.2.2 Displacement of Current Steam Needs	51
2.3 Electricity Utilization.	51

1. PROCESS DESCRIPTION

Elemental phosphorus is produced by reduction of phosphate ore at elevated temperatures in an electric furnace, Figure A-1. Phosphate ore (or rock), primarily fluorapatite, $\text{Ca}_5\text{F}(\text{PO}_4)_3$, is mined, washed, and blended to provide a uniform stock. The ore is then fed into a rotary nodulizing kiln where it is heated and nodules are produced. The nodules are sized, mixed with silica and coke, and fed into the electric furnace.

As the feed mix moves through the furnace it is heated and melted, and a reaction takes place in which coke reduces phosphate ore, forming elemental phosphorus and carbon monoxide. This reaction, which takes place at about $1,482^\circ\text{C}$ ($2,700^\circ\text{F}$), is approximated by

Elemental phosphorus leaves the furnace as vapor and is recovered by condensation, then purified and sold. Carbon monoxide is recovered for fuel in the rotary kiln. The remainder of the feed leaves the furnace as molten slag and metal.

A furnace typically has three tap holes, two for slag tapping and one for metal tapping. Because of its high density, iron contained in the phosphate ore sinks through molten slag and is tapped at the bottom of the furnace. This iron is also referred to as metal or ferrophos. Slag which floats on the metal is tapped several inches off the furnace bottom. Furnaces may be tapped about once an hour to remove accumulated slag. Metal is tapped about once a day. Typically, the molten slag is dumped into a nearby cooling bed and allowed to cool for about 2 days before it is water-quenched. (Earlier quenching would cause the slag to react with water, releasing undesirable fluoride emissions.) The slag is then dug up and used as a construction material.

Figure A-1. Elemental phosphorus production.

Elemental phosphorus manufacture is an energy-intensive process. In the furnace itself, electrical energy is largely the source of energy, providing heat for the chemical reaction as well as for heating the reactants to the reaction temperature. After the reaction is completed the molten byproducts leave the furnace containing about 29% of the electric furnace energy input. This energy is available for recovery.

There are nine elemental phosphorus plants in operation in the United States. Table A-1 gives an estimate of annual capacity and available heat at these plants.

TABLE A-1. PRODUCTION CAPACITY AND AVAILABLE HEAT
IN SLAG FOR THE ELEMENTAL PHOSPHORUS
PLANTS IN THE UNITED STATES, 1977 ESTIMATES

Plant	Estimated annual phosphorus elemental capacity, Gg (tons)	Estimated potential for annual slag waste heat recovery, PJ (10^{12} Btu) ^a
Electro-Phos	18.0 (20,000)	0.33 (0.32)
FMC	122.0 (135,000)	2.26 (2.14)
Mobil	5.0 (6,000)	0.10 (0.09)
Monsanto-Columbia	118.0 (130,000)	2.18 (2.06)
-Soda Springs	89.0 (98,000)	1.64 (1.56)
Occidental	50.0 (55,000)	0.92 (0.87)
Stauffer-Mt. Pleasant	41.0 (45,000)	0.75 (0.71)
-Silver Bow	36.0 (40,000)	0.67 (0.63)
-Tarpon Springs	21.0 (23,000)	0.39 (0.36)
Total	500.0 (552,000)	9.24 (8.76)

^aSee Table 4 for assumptions.

2. ENERGY UTILIZATION ALTERNATIVES

This study identified several potential alternative uses for energy from slag waste heat recovery options. These can be classified into the three general categories discussed below:

- 1) utilization of heated gas (air), 2) utilization of steam, and
- 3) utilization of electricity.

2.1 Heated Gas (Air) Utilization

Alternatives in this category fall into two areas: 1) preheating kiln combustion air and/or CO fuel, and 2) drying wet phosphate ore kiln feed.

2.1.1 Gas Preheating--

Based on industry estimates, the first alternative involves about 40 kg/s (88 lb/s) of preheated primary combustion air and 11 kg/s (25 lb/s) of CO gas typically saturated with moisture. It is estimated that further preheating of these gases to temperatures as high as 538°C (1,000°F) may represent a potential utilization for 15% of the heat in the molten slag.

Utilization of the contact tower concept for preheating air and CO gases is judged technically feasible. However, because the above air and CO fuel rates may represent a total for several kiln units, additional costs would be incurred for the hot gas distribution if the preheating was done in one tower. Another alternative is to have three separate preheating towers, which may create a space problem. In either case, the potential energy savings do not appear high enough to justify this alternative economically.

2.1.2 Ore Drying--

Phosphate ore feed into the kiln contains moisture. It is estimated that molten slag contains sufficient energy [78 MW (2.68×10^8 Btu/hr)] for evaporating this moisture. The contact tower/drying tower combination is judged potentially feasible in this application. Further testing would be required to verify that kiln fuel usage is actually reduced through the use of dried kiln feed.

2.2 Steam Utilization

Alternatives in this category fall into two areas: 1) displacing electric motors with steam turbines, and 2) displacing current plant steam needs with steam made from slag.

2.2.1 Electric Motor Displacement--

Elemental phosphorus manufacture involves the operation of many prime movers. These represent a potential application for small steam turbines. The application is evaluated in more detail in Appendix K.

2.2.2 Displacement of Current Steam Needs--

Elemental phosphorus manufacture requires very little steam. This alternative offers no potential for slag waste heat recovery.

2.3 Electricity Utilization

Electricity is the preferred form of energy for elemental phosphorus plants, so this energy could be used in a variety of applications. This utilization alternative is evaluated in Appendix J.

APPENDIX B
SLAG PROPERTIES
Summary

Several different parameters and assumptions were used in the engineering calculations throughout this report. These primarily concern slag properties. In general, very little is known about specific slags. The literature contains some property data on slags and glasses of different origins. These were considered in approximating the properties of phosphorus slag. For consistency, a list of slag properties was developed and used throughout this report.^a The properties are summarized below.

Property	Unit	Value
Viscosity ^{20, b}	Pa•s	0.768 @ 1,300°C (2,372°F) 0.615 @ 1,325°C (2,417°F) 0.433 @ 1,365°C (2,489°F) 0.423 @ 1,375°C (2,507°F) 0.318 @ 1,425°C (2,597°F) 0.203 @ 1,500°C (2,732°F) 0.149 @ 1,545°C (2,813°F) 0.126 @ 1,610°C (2,930°F)

(continued)

^aMany slag thermal properties are not well defined. This may have a noticeable influence on the slag waste heat recovery option's design and energy balance. For example, heat capacities for slags or glasses range anywhere from 0.17 to 0.33 Btu/lb-°F, depending on temperature, composition, and source. Since there is a proportional relationship between the heat capacity and the energy available in slag, the assumption of heat capacity directly influences the amount of recoverable energy. For consistency, the heat capacity of 0.26 Btu/lb-°F was used throughout this report.

^bSee Figure B-1.

Property	Unit	Value
Density ²¹	kg/m ³ (lb/ft ³)	2,910 (182)
Heat capacity ^{21,22, a}	kJ/kg (Btu/lb-°F)	1.09 (0.26) @ all temperatures
Thermal conductivity ²¹	W/m•K (Btu/hr-ft•°F)	1.1 (0.64)
Heat of fusion (CaSiO ₃) ²¹	J/mole (cal/mole) (Btu/lb)	56,100 (13,400) (208)
Fusion temperature ²¹	°C (°F)	1,445 (2,633)
Freezing temperature ²¹	°C (°F)	1,260 (2,300)
Surface tension ²³	N/m (dynes/cm)	0.20 (200)

Figure B-1. Slag viscosity as a function of temperature at 1.01 SiO₂/CaO weight ratio.

APPENDIX C

SLAG HANDLING AND TRANSPORTATION

Technical Discussion and Economics

CONTENTS

	<u>Page</u>
1. Technical Discussion	55
2. Economics	57
2.1 Capital Costs	57
2.2 Operating Costs	59

1. TECHNICAL DISCUSSION

Transportation of molten slag from the point of generation (tapping) to the surge vessel can be done by means of ladles. A ladle is a large, conically shaped steel pot, conceptually illustrated in Figure C-1, and sized to contain slag melt from one tapping. Typically, a ladle may have 11-m³ to 17-m³ (400-ft³ to 600-ft³) holding capacity. Once filled, the ladle is picked up by a specially designed truck carrier, conceptually illustrated in Figure C-2, and hauled to a specified location for dumping. Where conditions permit direct feeding of molten slag into a slag waste heat recovery process, slag handling and transportation may not be needed.

Figure C-1. Transportation of molten slag is implemented by means of steel ladles 400 ft³ to 600 ft³ in capacity.

Figure C-2. Slag pot carrier.

Pot handling of molten slag, practiced in the iron and steel industry, is finding its way into the elemental phosphorus industry. Since the method is being used commercially, its technical feasibility is well proven. For example, Monsanto Company has tested slag ladle handling at its Columbia plant for several years and found it superior to any other slag handling practice.

The method appears to create a safer and cleaner environment around slag producing operations than do other practices. Without it, slag is usually dumped directly from the furnaces to a pit, generating pollution and sometimes restricting access to the furnaces. Pot handling removes the slag melt from the general working area, making the above problems virtually nonexistent.

Also, the method is simple and offers the flexibility needed for the waste heat recovery process to handle slag produced by

several discrete furnace units. The slag from these units can be collected and transported away from the slag producing operations to a convenient central location for heat recovery.

During transportation, slag may lose some heat. A crude estimate of this heat loss is presented in Appendix L.

2. ECONOMICS

2.1 Capital Costs

Slag runners	\$ 19,000
Slag pots	560,000
Slag pot carriers	625,000
Installation	6,000
Site preparation	<u>18,000</u>
Total direct costs	1,228,000
Contingency	120,000
Fixed capital investment	1,348,000
Startup	47,000
Total capital investment	<u>\$1,395,000</u>

Equipment requirements and costs are based on industry estimates. Six elemental phosphorus furnaces are assumed to produce the slag rate of 33.9 kg/s (269,000 lb/hr) used in this study. Each of the six furnaces is equipped with two slag runners (troughs or channels in which the molten slag flows and is transported from the furnace tap holes to ladles) estimated at \$1,600 each (December 1977-January 1978 prices). A total of 14 slag pots are provided to receive the slag, two for each furnace plus two spares. Each of these pots costs approximately \$40,000. Three slag pot carriers are also needed to transport the slag, two of which operate at all times. The vehicles cost about \$208,000 each, based on current prices.

Only the slag runners require installation. The cost for this operation is estimated at \$6,000 or 30% of the purchase price. Because this installation is a relatively small task, it is assumed that the installation is handled by plant personnel. Hence, no allowance is made for indirect construction expenses or a contractor's fee.

Some site preparation is needed to allow the slag carriers to dump their loads into the surge vessel. It is assumed that a 15-m (50-ft) wide, 4° ramp elevating 4.6 m (15 ft) with a 30-m (100-ft) plateau will be sufficient for this operation. The cost to build this ramp and a concrete retaining wall is estimated at \$18,000.²⁴

The equipment cost estimates cited in the preceding discussion are purchased costs. They do not include freight charges, which will be site specific. Thus, a 10% contingency allowance is made to cover these and other unforeseen capital requirements.

Because the slag handling system is a simple, proven technology, no significant startup problems are anticipated. However, \$47,000 is allotted to cover employee salaries over the period immediately preceding startup. Here, it is assumed that supervisory personnel will be hired 3 months before actual operation, and that operating personnel will be hired 2 months prior to startup. Overhead rates of 25% for supervisory personnel and 10% for operating labor are added onto the base salaries.²⁵ Wage rates used in these calculations, as well as the number of personnel involved, are discussed in the following section.

Because there are no significant raw material or product inventories associated with the operation, no allowance is made for working capital.

2.2 Operating Costs

Labor and supervision	\$ 201,000
Fuel	25,000
Maintenance	54,000
Total direct costs	280,000
Depreciation	689,000
Taxes and insurance	27,000
Total fixed costs	716,000
Overhead	114,000
Total operating costs	<u>\$1,110,000</u>

The slag handling system will operate about 8,000 hours per year on a continuous basis. Thus, four shifts are required. Two men are assumed per shift to drive the slag pot carriers. Further assuming a \$7.00/hour wage rate, the annual cost for operating labor is \$122,000. One supervisor is also needed for each shift to oversee the slag handling operations. At \$9.00 per hour, the total cost for supervision is \$79,000. Thus, the overall labor cost is \$201,000 per year.

Two slag carriers operating 8,000 hours per year are assumed to consume 55,000 gallons of fuel. At \$0.45/gal, this results in an annual fuel expenditure of \$25,000.

Annual costs for maintenance, materials, and labor are difficult to project on a direct basis. Therefore, this cost is estimated at 4% of the fixed capital investment (or \$54,000), a typical value for simple operations.^{25,26} It is assumed that 50% of the maintenance cost is labor.

Depreciation charges are quite significant due to the short life expectancies of the slag handling equipment. Slag runners and carriers are expected to last about 5 years. Slag pots, however, last only about 1 year. Thus, the full \$560,000 invested

in the slag pots is written off over the first year of operation. Likewise, the \$644,000 invested in the runners and carriers is written off over a 5-year period. Here, straight-line depreciation is assumed with no salvage values for the various equipment items.

Like maintenance, taxes/insurance and overhead costs are estimated on a percentage basis. Local taxes and insurance premiums are assumed at 2% of the fixed capital investment. Incremental overhead costs are estimated at 50% of the expenditure for supervisory, maintenance, and operating labor. This charge covers fringe benefits and other small increases in total plant overhead.

APPENDIX D
SURGE VESSEL
Technical Discussion and Economics

CONTENTS

	<u>Page</u>
1. Technical Discussion	62
2. Economics	63
2.1 Capital Costs	63
2.2 Operating Costs	64

1. TECHNICAL DISCUSSION

A surge vessel equalizes the surges of slag resulting from batch tapping and provides sufficient storage for molten slag to assure continuous operation of a slag waste heat recovery process. An additional function of the surge vessel is to separate lumps of unmelted materials and ferrophos from the molten slag and prevent these from entering the waste heat recovery process. Where conditions permit direct feeding of molten slag into the process, the surge vessel may not be needed.

The surge vessel is a holding tank. Its size is approximated based on an assumption that three out of the six furnaces are tapped simultaneously. This would result in an instant slag volume of 21 m^3 (740 ft^3). Additional volume was assigned to provide for slag/ferrophos separation, ferrophos tapping, and a void space above the slag, bringing the total vessel holding capacity to an estimated 28 m^3 ($1,000 \text{ ft}^3$). Provisions are made for feeding the vessel with the molten slag, discharging the slag from the vessel, and tapping the ferrophos. A shielded manhole on the top is used to feed the vessel. A tapping hole at the bottom is used for ferrophos tapping and also for draining the vessel. An opening in the vessel wall is used for discharging the molten slag from the vessel into the slag waste heat recovery process. To minimize heat losses all the outside surfaces of the vessel are insulated.

Because it has no heat input, it is not known whether the surge vessel can operate trouble-free for extended periods of time. A crude calculation of potential heat loss from the surge vessel is presented in Appendix M. Though small, the heat losses may result in a continuous buildup of solidified slag on the walls eventually phasing the vessel out of service. On the other hand, the buildup will reduce heat losses, and it is plausible that the vessel will reach thermal equilibrium, providing feasible

conditions for long and trouble-free service. While the potential difficulties cited above cannot be fully resolved at this time, the wealth of available industrial experience in handling and containing hot melts provides a strong basis for considering the surge vessel technically feasible. Future demonstration of the surge vessel operation is desirable, however.

Because of the high differential between the density of slag and that of ferrospinel, no difficulties are anticipated in separating the two melts in the surge vessel and removing the ferrospinel via tapping. Also, no difficulties are envisioned in feeding the surge vessel with the slag melt.

2. ECONOMICS

2.1 Capital Costs

Slag runners	\$ 3,000
Surge vessel	87,000
Installation	35,000
Total direct costs	125,000
Engineering, supervision, and construction	25,000
Total direct and indirect costs	150,000
Contractor's fee	8,000
Contingency	15,000
Fixed capital investment	173,000
Total capital investment	<u>\$173,000</u>

The surge vessel is equipped with two slag runners; one connects the surge vessel with the heat recovery process, the other provides for ferrospinel removal. It is assumed that these runners cost the same as the furnace runners--approximately \$1,600 each as of January 1978.

The surge vessel cost estimate is based on the current price for slag pots (\$40,000 for a 600-ft³ pot, see Appendix C). A direct

cost-to-capacity ratio is assumed, with 10% added to account for fabrication of an enclosed vessel with entrance and exit apertures. The vessel surface (550 ft^2) is insulated with 0.1 m (4 in.) of high alumina insulation at a cost of $\$270/\text{m}^2$ ($\$25/\text{ft}^2$).²⁷ Thus, the total vessel cost is

$$\left(\frac{1,000 \text{ ft}^3}{600 \text{ ft}^3} \right) (\$40,000) (1.1) + 550 \text{ ft}^2 (\$25/\text{ft}^2) = \$87,000$$

The cost to install the surge vessel is estimated at 40% of the purchase price. This may be considered an upper bound estimate.²⁶

No indirect costs were taken into account when estimating the capital requirements for the slag transportation system because it could be installed by plant personnel. However, the surge vessel is, by definition, an integral part of a much larger energy recovery system which would have to be constructed by an outside contractor. Thus, a proportionate amount of the indirect construction costs must be attributed to the surge vessel. These costs (i.e., engineering, supervision, and construction overhead) are estimated at 20% of the installed equipment cost. Likewise, a contractor's fee is estimated at 5% of the direct and indirect system costs.²⁶

Due to the preliminary nature of this estimate, a contingency factor is recommended; a value of 10% of the direct and indirect costs is used. No working capital or startup costs are anticipated, however. Site preparation costs are included in the capital requirements for the slag handling and transportation system.

2.2 Operating Costs

Maintenance	\$ 2,000
Depreciation	17,000
Taxes and insurance	3,000
Overhead	2,000
Total operating costs	<u>\$25,000</u>

The surge vessel should function with minimum attention; there is no labor requirement for this unit operation *per se*. Thus, the operating costs are reduced to maintenance with related overhead and fixed charges to capital.

Maintenance charges will depend primarily on the occurrence of slag buildup and inlet or outlet clogging. With proper design, these problems should be minimal. Thus, maintenance material and labor costs are estimated at 2% of the fixed capital investment.²⁶ Associated overhead charges are estimated at 50% of the maintenance labor cost, or about \$2,000. Here, it is assumed labor constitutes nearly all the total maintenance cost.

Unlike slag handling equipment, the surge vessel will not be subject to constant thermal cycling. Thus, it is difficult to estimate the life expectancy of this unit. Certainly, it will be greater than the economic lives of the furnace runners and slag pots. Therefore, 10-year straight line depreciation is assumed with no salvage value.

As before, annual costs for taxes and insurance are estimated at 2% of the fixed capital investment.

APPENDIX E

SLAG WASTE HEAT RECOVERY - FLOAT CHAMBER

Technical Discussion

CONTENTS

	<u>Page</u>
1. Float Glass Process	69
2. Float Chamber Process Design Considerations	73
2.1 Derivation of the Mathematical Model	74
2.1.1 Equations of Motion for Tin	74
2.1.2 Equations of Motion for Slag	74
2.1.3 Energy Equation for Tin	76
2.1.4 Energy Equation for Slag	77
2.1.5 Correction for Finite Slag Thickness	82
2.1.6 Nonconstant Profiles for T_o and T_w	83
2.2 Numerical Results	84
2.2.1 Results for Constant T_o and T_w	84
2.2.2 Results for Nonconstant T_o and T_w	87
2.2.3 Length Required to Remove Latent Heat of Fusion	97
2.2.4 Comparison of Model System with Commercial Float Glass Systems	97
2.3 A Closed-Form Solution	98
2.4 Summary and Justification of Major Assumptions	103
2.4.1 Summary of Assumptions	103
2.4.2 Conduction Through Slag in Flow Direction Is Negligible	105
2.4.3 Free Convection from Slag Is Negligible	106
2.4.4 Conduction Through Tin in Flow Direction Is Negligible	108
2.4.5 Physical Properties are Constant	109
2.4.6 Conditions are Uniform in the z-Direction	109
2.4.7 Forced Convection in Tin Is Negligible	109
2.4.8 Developing Slag Flow in the Entrance Region is Neglected	109
2.4.9 Slag Temperature Is Uniform in y-Direction	110
2.4.10 Temperature Profile is Specified at $y = 0$	110

CONTENTS (continued)

	<u>Page</u>
2.5 Heat-Transfer Calculations for Float Chamber Process	111
2.5.1 Heat-Transfer Coefficients	111
2.5.1.1 Piping Configuration.	111
2.5.1.2 Heat-Transfer Coefficient for Boiling Section	112
2.5.1.3 Heat-Transfer Coefficient for Superheating Section	115
2.5.2 Surface Area Required for Conductive Heat Transfer	115
2.5.2.1 Conductive Heat-Transfer Rate	115
2.5.2.2 Steam Rate	115
2.5.2.3 Area Required for Boiling Section	116
2.5.2.4 Area Required for Superheating Section	116
2.5.2.5 Total Requirements for Conduction	117
2.5.3 Surface Area Required for Radiative Heat Transfer	117
2.5.3.1 Radiative Heat-Transfer Rate	117
2.5.3.2 Steam Rate	117
2.5.3.3 Area Required for Boiling Section	118
2.5.3.4 Area Required for Superheating Section	118
2.5.3.5 Total Requirements for Radiation.	119
2.5.4 Summary of Calculations	119
2.5.5 Area Required for First Section of Float Chamber	119
2.5.5.1 Conduction	120
2.5.5.2 Radiation	120
2.5.6 Area Required in Final Section of Float Chamber	120
2.5.6.1 Conduction	120
2.5.6.2 Radiation	121
2.5.7 Pressure Drop Calculations	121
2.5.7.1 Superheating Section	121
2.5.7.2 Boiling Section	123
2.5.8 Air vs. Steam as a Heat-Transfer Medium	124
2.6 Amount of Refractory Material Required for Float Chamber	124

CONTENTS (continued)

	<u>Page</u>
2.7 Startup Boiler Estimate	125
2.7.1 Latent Heat of Tin	125
2.7.2 Sensible Heat of Tin	126
2.7.3 Sensible Heat of Refractory	126
2.7.4 Sensible Heat of Graphite	126
2.7.5 Total Energy Required.	126
2.7.6 Boiler Horsepower.	127
2.8 Estimate of Tin Loss by Penetration Into Slag . .	127
2.8.1 Estimate Based on Concentration Profiles in Reference 51.	128
2.8.2 Estimate Based on Reference 36	129
3. Float Chamber Process Design Summary	130
3.1 Float Chamber	130
3.2 Steam and Feedwater Handling	135
Abbreviations for Appendix E	141

Thermal energy from phosphorus slag can be partially recovered by means of a float chamber process. In the process, molten tin (or another heavy metal or alloy) is used to support molten and/or solidified slag, facilitating slag handling, transportation and cooling. The process closely resembles current commercial technology for manufacture of flat and sheet glass.

Almost all flat glass in the United States is produced by pouring the glass melt onto, and transporting the formed sheet of glass across, a large molten tin pool. Wide acceptability of this technology, commonly referred to as the float glass process by the glass industry, suggests and supports the technical feasibility of slag waste heat recovery via such a process.

Background information concerning the float glass technology is presented in Section 1. This is followed by engineering evaluations and design calculations, Section 2, and process design summary, Section 3; the latter two sections pertain to the application of the float chamber process for waste heat recovery from phosphorus slag.

1. FLOAT GLASS PROCESS

The float process for the manufacture of flat glass was introduced in 1959 by Pilkington Brothers, Ltd., of England. Since then float glass has virtually replaced polished plate glass and has captured a large share of the market traditionally held by sheet glass. It is anticipated that the float process will eventually become the single major process for the manufacture of nearly all types of flat glass.²⁸

The essence of the float process is the formation of a continuous ribbon of glass on the surface of a bath of molten tin. The molten glass attains a perfectly flat, fire-polished surface while floating on the tin. It is then cooled as it is advanced

along the surface of the tin until it is hard enough to be lifted out of the tin bath without marring the surface.

A float chamber for the manufacture of float glass is illustrated in Figure E-1.²⁹ Molten glass flows from the end of the glass-melting furnace (1) along a shallow channel (2) past a cutoff gate (3) and a flow-control gate (4). The glass then flows onto the tin bath via the spout (5).

Figure E-1. Diagram of float chamber for production of float glass.²⁹

The tank containing the tin consists of a steel housing (6) lined with refractory bricks (7) which are resistant to molten tin. The side walls and roof (8) of the tank are also constructed of refractory material enclosed in a steel shell. Precise temperature control is maintained along the entire length of the chamber by means of electric heaters (9) and cooling tubes (10) above the glass ribbon. Heaters and cooling tubes are also located in the tin layer itself or are embedded in the refractory beneath the tin.

In order to prevent oxidation of the tin, a protective atmosphere of nitrogen and hydrogen or nitrogen, hydrogen and carbon monoxide is maintained in the space above the tin bath.

At the outlet (11) of the float chamber, the glass ribbon is conveyed on rollers to the annealing Lehr (12). The tractive force necessary to move the glass ribbon along the tin bath is supplied by a pair of traction rollers located between the float chamber and the annealing Lehr.

In the float glass process, the glass ribbon attains an equilibrium thickness of between 6 mm and 7 mm under the opposing forces of surface tension and gravity. The equilibrium thickness, T , is given by:³⁰

$$T^2 = (S_g + S_{gt} - S_t) \frac{2\rho_t}{g\rho_g(\rho_t - \rho_g)} \quad (E-1)$$

where S_g = surface tension of glass melt
 S_t = surface tension of tin
 S_{gt} = tin-glass interfacial tension
 ρ_g = density of glass
 ρ_t = density of tin
 g = gravitational constant

In order to produce glass with thickness greater than the equilibrium value, the initial outward spread of the molten glass is restricted by nonwetted parallel longitudinal guides (e.g., graphite bars) as shown in Figure E-2.³¹ Under these conditions, the thickness of the ribbon is determined by the mass flow rate of the molten glass and the speed of the traction rollers. Float glass up to 30 mm thick is produced in this manner.

Figure E-2. System for production of float glass of thickness greater than equilibrium value.³¹

Float glass of thickness less than the equilibrium value can be produced from a wide primary ribbon (Figure E-3). The width and thickness of the primary ribbon are controlled by pairs of power-driven, water-cooled edge rolls that grip the edges of the cooled ribbon (700°C) and define its speed. Downstream of the edge rolls, the ribbon is reheated to approximately 850°C and stretched. The formation of the primary ribbon under conditions of low tractive force is separated from the stretching process requiring a high tractive force by the relatively stiff section of cooled ribbon. The necessary force reaction is thus provided by the edge rolls which grip the stiffer section.³¹ Float glass as thin as 3 mm is made in this manner.

An alternative method of producing thin float glass is to alter either the surface tension or density terms in Equation E-1. For example, the apparent density of the glass relative to that of tin can be increased by selectively applying a fluid pressure over the glass which is greater than that over the tin at the edges of the glass ribbon.³²

The glass sheet emerging from the annealing lehr must be inspected for defects, cut to size, packaged, stored and shipped. Highly

Figure E-3. System for production of float glass of thickness less than the equilibrium value.³¹

sophisticated and automated computer controlled methods have been developed to perform these operations in order to handle the extremely high production rates characteristic of the float process. These so-called cold-end operations are described in References 33 and 34.

2. FLOAT CHAMBER PROCESS DESIGN CONSIDERATIONS

The essential part of the float chamber process is the float chamber itself. It contains the tin pool, and heat transfer takes place here. Consequently, the design of the float chamber process is primarily based on the design of the float chamber.

In order to determine the size of the float chamber, a mathematical model describing the transport of mass, heat, and momentum in the tin-slag system (Figure E-4) is required. The system is described by the following set of equations:

- Equations of motion for tin
- Equations of motion for slag
- Energy equation for tin
- Energy equation for slag

Figure E-4. Schematic representation of float chamber.

This set of coupled partial differential equations is nonlinear due to the radiative boundary condition at the upper slag surface and the temperature dependence of the transport properties (viscosity and thermal conductivity).

The effort required to solve these equations rigorously is prohibitive in a preliminary feasibility study.^a Therefore, in deriving the model a number of assumptions were made which greatly simplify the mathematics while preserving the ability of the model to describe the gross features of the system.

The simplified mathematical model is derived and solved in Section 2.1. Results of numerical calculations are presented and discussed in Section 2.2. In Section 2.3, a closed-form solution is obtained by linearizing the model. This solution is useful in establishing an upper bound on the size of the float chamber. The major assumptions used in deriving the mathematical model are summarized and justified in Section 2.4.

2.1 Derivation of the Mathematical Model

2.1.1 Equations of Motion for Tin--

Momentum transfer from the slag moving over the surface of the

^aA detailed two-dimensional model of the float glass process is derived in Reference 35. The considerable numerical difficulties involved in solving the model equations are also discussed in the reference.

tin causes a circulatory flow pattern in the tin bath. At the upper tin surface, hot tin flows toward the cold end of the bath, while at the bottom of the bath, cold tin tends to flow toward the hot end of the bath. The net effect is the convective transport of heat in the x-direction. This heat transport is undesirable in the present application since it increases the length of the float chamber required to effect a given degree of slag cooling.

In the present derivation it is assumed that the float chamber will be designed to minimize convective heat transfer in the tin. For example, it may be necessary to place carbon dams (i.e., baffles or spacers) at intervals in the tin bath to interrupt the convective flow pattern.^{32,36,37} As an approximation then, the convective heat transfer in the tin is neglected, and the tin bath is treated as a quiescent liquid.

It should be noted that free convection effects are absent in the tin bath because the tin is cooler at the bottom than at the top.

2.1.2 Equations of Motion for Slag--

In the entrance region of the tin bath, the molten slag will spread over the surface of the tin until it achieves an equilibrium thickness under the combined forces of surface tension and gravity.^{31,38} The equilibrium thickness of slag on tin is assumed to be the same as that of float glass; namely, 6.4 mm (1/4 in.). After an initial cooling period, the slag solidifies and continues to move down the tin bath as a solid sheet.

For the present study, the flow in the entrance region is neglected and the slag is assumed to move as a solid sheet of constant dimensions over the entire length of the tin bath. The equation of continuity (conservation of mass) for the slag is then:

$$\rho_s u \delta W = \rho_s V \quad (E-2)$$

or

$$u = \frac{V}{\delta W} \quad (E-3)$$

where^a

- ρ_s = slag density
- u = velocity of slag sheet
- δ = thickness of slag sheet
- W = width of slag sheet
- V = volumetric flow rate of slag

2.1.3 Energy Equation for Tin--

Neglecting convection, the energy equation for the tin reduces to the conduction equation:

$$\vec{\nabla} \cdot (k_t \vec{\nabla} T_t) = 0 \quad (E-4)$$

where $\vec{\nabla}$ = "del" operator
 k_t = thermal conductivity of tin
 T_t = temperature of tin

Assuming constant thermal conductivity, Equation E-4 simplifies to:

$$\nabla^2 T_t = 0 \quad (E-5)$$

Assuming uniform temperature across the tin bath width (z-direction), Equation E-5 becomes:

$$\frac{\partial^2 T_t}{\partial x^2} + \frac{\partial^2 T_t}{\partial y^2} = 0 \quad (E-6)$$

^aA complete list of symbols may be found at the end of this appendix.

Heat conduction in the x-direction (down the bath) can be neglected in comparison with conduction in the y-direction (across the tin depth) since the temperature gradient in the y-direction is much greater than that in the x-direction. (This statement will be justified in Section 2.4.) Hence, Equation E-6 reduces to:

$$\frac{d^2 T_t}{dy^2} = 0 \quad (E-7)$$

The solution is:

$$T_t = C_1 + C_2 y \quad (E-8)$$

where C_1 and C_2 are constants. That is, the temperature varies linearly across the tin depth.

2.1.4 Energy Equation for Slag--

As a first approximation, the resistance to heat transfer within the slag transverse to the flow direction is assumed negligible compared with the resistance of the tin layer and the radiative resistance. This assumption is strictly valid in the limit of zero slag thickness. An approximate correction for finite slag thickness will be considered later in this discussion. In addition, the temperature is assumed uniform across the width of the slag and the float chamber.

With the above assumptions, the temperature is uniform over the entire width and depth of the slag sheet, and an energy balance can be made on a stationary volume element of length Δx in the slag sheet (Figure E-5).

Figure E-5. Energy balance diagram for slag sheet.

The energy fluxes in Figure E-5 are defined as follows:

- q_c = conductive flux through tin
- q_r = radiative flux from slag to overhead boiler tubes
- q_{cv} = free convective flux from upper slag surface
- q_x = conductive flux in the direction of slag flow
- $\rho_s c_s u (T_s - T_{Ref})$ = flux due to bulk flow of slag through the volume element

In the bulk flow term, T_s is the slag temperature and T_{Ref} is an arbitrary reference temperature. Since the temperature has been assumed uniform in the y - and z -directions, $T_s = T_s(x)$.

It will be shown in Section 2.4 that the conductive flux, q_x , is negligible compared with the bulk flow term. In addition, the free convection term, q_{cv} , is small compared with q_r and q_c , and can be neglected. Thus, the steady state energy balance for the slag sheet element is as follows:

$$\left\{ \begin{array}{l} \text{Rate of energy} \\ \text{leaving element} \end{array} \right\} = \left\{ \begin{array}{l} \text{Rate of energy} \\ \text{entering element} \end{array} \right\}$$

$$\rho_s c_s u (T_s - T_{Ref}) \Big|_{x + \Delta x} \delta W + q_c W \Delta x + q_r W \Delta x = \rho_s c_s u (T_s - T_{Ref}) \Big|_x \delta W \quad (E-9)$$

Dividing by $W\Delta x$ and rearranging yields:

$$\rho_s c_s u \delta \left[\frac{(T_s - T_{Ref})|_{x + \Delta x} - (T_s - T_{Ref})|_x}{\Delta x} \right] + q_c + q_r = 0 \quad (E-10)$$

Taking the limit as $\Delta x \rightarrow 0$:

$$\rho_s c_s u \delta \left(\frac{dT_s}{dx} \right) + q_c + q_r = 0 \quad (E-11)$$

In obtaining Equation E-11, it has been assumed that the slag density, ρ_s , and heat capacity, c_s , are constant, independent of temperature.

Since the temperature varies linearly through the tin layer, the flux q_c is given by:

$$q_c = k_t \left(\frac{T_s - T_o}{D} \right) \quad (E-12)$$

where D = depth of tin layer

T_o = temperature at bottom of tin layer

The radiative flux is approximated by:^{39,40}

$$q_r = \sigma F_\epsilon \bar{F}_{21} (T_s^4 - \bar{T}_w^4) \quad (E-13)$$

where σ = Stefan-Boltzman constant

F_ϵ = emissivity factor

$\bar{F}_{21}$ = geometric view factor

T_w = overhead boiler tube wall temperature (overbar on T_w denotes average over width of float chamber)

Substituting Equations E-12 and E-13 into Equation E-11 and rearranging yields:

$$\frac{dT_s}{dx} + \frac{k_t}{\rho_s c_s u \delta D} (T_s - T_o) + \frac{\sigma F_{\epsilon} \bar{F}_{21}}{\rho_s c_s u \delta} (T_s^4 - \bar{T}_w^4) = 0 \quad (E-14)$$

For the case in which T_o and $\bar{T}_w^4$ are constants, Equation E-14 can be written as follows:

$$\frac{dT_s}{dx} + AT_s + BT_s^4 = C \quad (E-15)$$

where A, B, and C are the following constants:

$$A = \frac{k_t}{\rho_s c_s u \delta D} \quad (E-16)$$

$$B = \frac{\sigma F_{\epsilon} \bar{F}_{21}}{\rho_s c_s u \delta} \quad (E-17)$$

$$C = \frac{k_t T_o}{\rho_s c_s u \delta D} + \frac{\sigma F_{\epsilon} \bar{F}_{21} \bar{T}_w^4}{\rho_s c_s u \delta} \quad (E-18)$$

Equation E-15 can be integrated by separating variables as follows:

$$\frac{dT_s}{AT_s + BT_s^4 - C} = - dx \quad (E-19)$$

$$\int_0^L dx = - \int_{T_1}^{T_2} \frac{dT}{AT + BT^4 - C} \quad (E-20)$$

$$L = \int_{T_2}^{T_1} \frac{dT}{AT + BT^4 - C} \quad (E-21)$$

where L = length of float chamber

T_1 = absolute slag temperature at entrance to float chamber

T_2 = absolute slag temperature at exit from float chamber

The integral in Equation E-21 can be evaluated numerically to obtain the required length of the float chamber corresponding to given values of slag exit temperature, tin depth, and other parameters appearing in Equations E-16, E-17 and E-18.

It is shown below that the length of the float chamber is inversely proportional to the width of the slag sheet. From Equation E-3,

$$u\delta = V/W \quad (E-22)$$

Substituting this result into Equation E-14 gives:

$$\frac{dT_s}{dx} + \frac{k_t W}{\rho_s c_s V D} (T_s - T_o) + \frac{\sigma F_\epsilon \bar{F}_{21} W}{\rho_s c_s V} (T_s^4 - \bar{T}_w^4) = 0 \quad (E-23)$$

Using the same assumption of $T_o = \text{constant}$ and $\bar{T}_w^4 = \text{constant}$, this equation can be rewritten as:

$$\frac{1}{W} \frac{dT_s}{dx} + \hat{A} T_s + \hat{B} T_s^4 = \hat{C} \quad (E-24)$$

where $\hat{A}$, $\hat{B}$ and $\hat{C}$ are given by:

$$\hat{A} = k_t / \rho_s c_s V D \quad (E-25)$$

$$\hat{B} = \sigma F_\epsilon \bar{F}_{21} / \rho_s c_s V \quad (E-26)$$

$$\hat{C} = k_t T_o / \rho_s c_s V D + \sigma F_e \bar{F}_{21} \bar{T}_w^4 / \rho_s c_s V \quad (E-27)$$

Integrating as before yields:

$$\frac{1}{W} \frac{dT_s}{\hat{A}T_s + \hat{B}T_s^4 - \hat{C}} = -dx \quad (E-28)$$

$$L = \frac{1}{W} \int_{T_2}^{T_1} \frac{dT}{\hat{A}T + \hat{B}T^4 - \hat{C}} \quad (E-29)$$

Since the constants $\hat{A}$, $\hat{B}$ and $\hat{C}$ do not depend on W , the integral in Equation E-29 is independent of W . Hence, the length, L , is inversely proportional to W .

2.1.5. Correction for Finite Slag Thickness--

If the resistance to heat transfer through the slag is not negligible, the slag temperature must strictly be considered a function of both x and y ; i.e., $T_s = T_s(x, y)$. In order to retain the one-dimensional character of the mathematical model, however, the slag resistance is accounted for in the following approximate manner. The slag temperature is assumed to be a function of x only and Equation E-12 is replaced by:

$$q_c = \frac{T_s - T_o}{R} \quad (E-30)$$

where R represents the combined resistance to heat transfer of both the slag and the tin. The effect of slag resistance on radiative heat transfer is neglected.

Since the tin and slag resistances are in series, they are additive; i.e.,

$$R = \frac{D}{k_t} + \frac{1/2\delta}{k_s} \quad (E-31)$$

In writing Equation E-31 it is assumed that only the lower half of the slag sheet contributes to the resistance to conductive heat transfer.

With the above change, Equations E-15 through E-29 remain valid if the term (D/k_t) is replaced by R wherever it appears in these equations.

2.1.6 Nonconstant Profiles for T_o and T_w --

The solution represented by Equation E-21 assumes that the temperature at the bottom of the tin bath, T_o , and the overhead tube wall temperature, T_w , do not vary with distance down the float chamber. However, this assumption can be relaxed without appreciably increasing the mathematical complexity of the model. For the present analysis, it is assumed that T_o and T_w can be represented by the same quadratic function of slag temperature, T_s .^a Thus, let

$$T_o = \alpha + \beta T_s + \gamma T_s^2 \quad (E-32)$$

$$T_w = \alpha + \beta T_s + \gamma T_s^2 \quad (E-33)$$

where α , β and γ are constants. Note that T_o and T_w are not quadratic functions of x , since T_s itself is a nonlinearly decreasing function of distance along the tin bath.

Substituting Equations E-32 and E-33 into Equation E-14 (and substituting R for D/k_t yields:

$$\frac{dT_s}{dx} + \frac{(1 - \beta)}{\rho_s c_s u \delta R} T_s - \frac{\gamma}{\rho_s c_s u \delta R} T_s^2 + \frac{\sigma F_\epsilon \bar{F}_{21}}{\rho_s c_s u \delta} T_s^4 - \frac{\sigma F_\epsilon \bar{F}_{21}}{\rho_s c_s u \delta} (\alpha + \beta T_s + \gamma T_s^2)^4 = \frac{\alpha}{\rho_s c_s u \delta R} \quad (E-34)$$

^a A more general case would be represented by nonidentical quadratic functions of slag temperature, T_s . This case was not developed further because the results of such calculation would not have significant influence on the conclusions made in this study.

This equation can be written more concisely as follows:

$$\frac{dT_s}{dx} + B_1 T_s - B_2 T_s^2 + B_3 T_s^4 - B_3 (\alpha + \beta T_s + \gamma T_s^2)^4 = B_4 \quad (E-35)$$

where

$$B_1 = \frac{1 - \beta}{\rho_s c_s u \delta R} \quad (E-36)$$

$$B_2 = \frac{\gamma_s}{\rho_s c_s u \delta R} \quad (E-37)$$

$$B_3 = \frac{\sigma F_\epsilon \bar{F}_{21}}{\rho_s c_s u \delta} \quad (E-38)$$

$$B_4 = \frac{\alpha}{\rho_s c_s u \delta R} \quad (E-39)$$

Equation E-35 can be integrated in the same manner as Equation E-14 to give:

$$L = \int_{T_2}^{T_1} \frac{dT}{B_1 T - B_2 T^2 + B_3 T^4 - B_3 (\alpha + \beta T + \gamma T^2)^4 - B_4} \quad (E-40)$$

2.2 Numerical Results^a

2.2.1 Results for Constant T_o and T_w

The integral appearing in Equation E-29 was evaluated numerically using a 15-point Gauss-Legendre quadrature.⁴¹ The integration was carried out in steps of 50°F each, the 15-point quadrature being performed at each step. Thus, the position along the tin bath was computed corresponding to each 50°F temperature change in the slag. The amount of heat transferred by radiation and conduction was also computed at each step. With this procedure,

^aWhere desirable, refer to conversion tables in Appendix N for conversion of English to metric units.

closure of the overall energy balance on the slag was obtained to four significant figures.

Values of parameters assumed in the calculations are listed in Table E-1. The values of T_1 , ρ_s , c_s , k_s , and V are based on data summarized in Appendix B. The value of k_t is an average value based on data in Reference 42. Values of W , δ , and D were selected on the basis of published information on the float glass process^{28,29,43,44}. Values for T_o , T_w and T_2 are based on the assumption that steam at a minimum temperature of 600°F is produced via crossflow heat exchange. Results of alternative assumptions for these temperatures are given in the following subsection. The value for F_ϵ is based on the assumptions that

TABLE E-1. SUMMARY OF PARAMETER VALUES USED IN NUMERICAL CALCULATIONS

Parameter	Value(s)	Units
T_1	3,160	°R
T_2	1,260	°R
k_t	22	Btu/hr-ft·°F
ρ_s	180	lb/ft ³
c_s	0.26	Btu/lb-°F
V	1,500	ft ³ /hr
W	20	ft
δ	0, 1/4, 1/2, 1	in.
D	2, 3, 4, 6	in.
k_s	0.64	Btu/hr-ft·°F
σ	0.1712×10^{-8}	Btu/hr-ft ² ·°R ⁴
F_ϵ^a	0.95	Dimensionless
$\bar{F}_{21}$	0.9	Dimensionless
T_o	1,160	°R
T_w	1,110	°R

^aThis value is probably optimistic. A value of 0.8 to 0.9 is probably more realistic.

the slag surface is gray and that the overhead boiler tubes are embedded in graphite blocks^a (gray surface). The value of $\bar{F}_{21}$ was obtained from Figure 4-11 of Reference 40 using the curve for radiation between long, narrow rectangular planes connected by nonconducting but reradiating walls.

Results of the calculations are summarized in Table E-2, which gives the required length of the float chamber as a function of tin depth and slag thickness. Although the results correspond to a 6.1 m (20 ft) wide slag sheet, the length corresponding to any other slag width can be easily obtained since the length is inversely proportional to slag width.

A slag thickness of 6.4 mm (1/4 in.) corresponds approximately to the equilibrium thickness, assuming slag behavior similar to that of float glass. In order to minimize the amount of tin required, and also to maximize the rate of heat transfer, a tin depth of 5.1 cm (2 in.) is preferred to the larger depths. [In practice, a tin depth of 10 cm to 15 cm (4 in. to 6 in.) would probably be employed in the entrance region of the bath where the slag flows onto the tin. However, this region has been neglected in the mathematical model.] The computer printout for this case ($W = 20$ ft, $\delta = 1/4$ in., $D = 2$ in.) is given in Table E-3. The values in the columns labelled Q_c and Q_r are cumulative heat-transfer rates by conduction and radiation over the region from the chamber entrance to a distance x downstream from the entrance. The values in Table E-3 have not been rounded; therefore, no inference should be made concerning the accuracy of calculations based on these data.

^aIn this case, T_w represents the surface temperature of the graphite blocks rather than the tube wall temperature.

TABLE E-2. DIMENSIONS OF FLOAT CHAMBER,
6.1 m (20 ft) SLAG WIDTH

Depth of tin bath, in.	Thickness of slag sheet, in.	Length of tin bath, ft
2	0	68
	1/4	166
	1/2	229
	1	308
3	0	95
	1/4	183
	1/2	241
	1	315
4	0	119
	1/4	198
	1/2	252
	1	321
6	0	160
	1/4	226
	1/2	272
	1	333

The slag temperature profile corresponding to the data in Table E-3 is shown graphically in Figure E-6.

2.2.2 Results for Nonconstant T_o and T_w --

The integral appearing in Equation E-40 was evaluated in the same manner as described previously for the integral in Equation E-29. Parameter values listed in Table E-1 were used with the exception of T_o and T_w values. Solutions were computed for the following cases:

TABLE E-3. TEMPERATURE PROFILE, HEAT FLUXES, AND CUMULATIVE HEAT TRANSFER RATES FOR THE CASE W = 20 ft, D = 2 in., $\delta = 1/4$ in.

T_s , °F	x, ft	q_c , Btu/hr-ft ²	q_r , Btu/hr-ft ²	Q_c , Btu/hr	Q_r , Btu/hr
2650.	0.8	82809.23	139222.37	1309856.13	2202185.50
2600.	1.6	80712.79	130414.05	2652508.25	4371615.00
2550.	2.5	78616.35	122023.70	4028656.25	6507591.00
2500.	3.4	76519.91	114037.88	5438953.50	8609421.00
2450.	4.4	74423.48	106443.40	6884206.00	10676421.00
2400.	5.4	72327.05	99227.22	8364970.00	12707917.00
2350.	6.5	70230.61	92376.59	9881948.00	14703247.00
2300.	7.6	68134.17	85878.93	11435788.00	16661766.00
2250.	8.8	66037.73	79721.94	13027118.00	18582848.00
2200.	10.1	63941.30	73893.45	14656546.00	20465888.00
2150.	11.5	61844.86	68381.59	16324653.00	22310306.00
2100.	12.9	59748.43	63174.67	18031994.00	24115552.00
2050.	14.4	57651.99	58261.24	19779090.00	25881110.00
2000.	16.0	55555.55	53630.04	21566426.00	27606498.00
1950.	17.7	53459.12	49270.06	23394446.00	29291274.00
1900.	19.5	51362.68	45170.50	25263552.00	30935044.00
1850.	21.5	49266.25	41320.77	27174096.00	32537462.00
1800.	23.6	47169.81	37710.50	29126376.00	34098236.00
1750.	25.8	45073.37	34329.56	31120634.00	35617136.00
1700.	28.1	42976.94	31168.02	33157044.00	37093996.00
1650.	30.7	40880.50	28216.17	35235716.00	38528720.00
1600.	33.4	38784.07	25464.54	37356684.00	39921288.00
1550.	36.4	36687.63	22903.85	39519896.00	41271768.00
1500.	39.5	34591.20	20525.06	41725216.00	42580320.00
1450.	43.0	32494.76	18319.34	43972412.00	43847204.00
1400.	46.8	30398.32	16278.08	46261140.00	45072800.00
1350.	50.9	28301.89	14392.91	48590936.00	46257616.00
1300.	55.4	26205.45	12655.64	50961204.00	47402312.00
1250.	60.4	24109.01	11058.33	53371188.00	48507724.00
1200.	66.0	22012.58	9593.26	55819940.00	49574908.00
1150.	72.2	19916.14	8252.91	58306284.00	50605208.00
1100.	79.3	17819.71	7029.99	60828740.00	51600332.00
1050.	87.4	15723.27	5917.44	63385416.00	52562536.00
1000.	96.9	13626.83	4908.41	65973832.00	53494888.00
950.	108.3	11530.40	3996.25	68590584.00	54401812.00
900.	122.3	9433.96	3174.57	71230728.00	55290232.00
850.	140.4	7337.53	2437.17	73886456.00	56172336.00
800.	165.7	5241.09	1778.08	76544080.00	57073956.00

Figure E-6. Slag temperature profile for the case
 $W = 20$ ft, $D = 2$ in., $\delta = 1/4$ in.

Case 1

Equations E-32 and E-33 were fitted to the following data points:

	$T_s, ^\circ\text{F}$	$T_o \text{ and } T_w, ^\circ\text{F}$
Entrance	2,700	700
Intermediate	1,000	650
Exit	600	500

} crossflow
 } semicounter-
 } current flow

The resulting values of the constants in Equations E-32 and E-33 are (for T_o, T_w, T_s in $^\circ\text{R}$):

$$\alpha = 307.8$$

$$\beta = 0.7897$$

$$\gamma = -1.6457 \times 10^{-4}$$

The idea here is to have crossflow heat exchange in the hotter portion of the float chamber and semicountercurrent flow in the cooler portion as indicated in Figure E-7. This flow pattern

Figure E-7. Schematic diagram of flow pattern for heat exchange corresponding to Case 1.

would permit T_O and T_W to fall below 600°F while still producing steam at 600°F. Note that T_O cannot be permitted to drop below 500°F because of the danger of the tin solidifying (melting point of tin $\approx$ 450°F).

Case 2

Both T_O and T_W were varied linearly with slag temperature as follows:

	$T_S, ^\circ\text{F}$	T_O and $T_W, ^\circ\text{F}$
Entrance	2,700	700
Exit	600	500

The constants in Equations E-32 and E-33 for this case are (for T_O, T_W, T_S in $^\circ\text{R}$):

$$\alpha = 859.0$$

$$\beta = 0.09524$$

$$\gamma = 0$$

This case results in a greater slag cooling rate than Case 1, but requires a countercurrent type flow pattern over a larger fraction of the float chamber in order to produce 600°F steam.

Case 3

Both T_O and T_W were varied linearly with slag temperature as follows:

	$T_S, ^\circ\text{F}$	T_O and $T_W, ^\circ\text{F}$
Entrance	2,700	900
Exit	600	500

The constants in Equations E-32 and E-33 for this case are (for T_O , T_W , T_S in $^{\circ}R$):

$$\alpha = 758.1$$

$$\beta = 0.1905$$

$$\gamma = 0$$

This temperature profile permits the production of steam at temperatures in excess of 600°F.

Solutions for the above three cases are given in Tables E-4, E-5, and E-6 for $W = 20$ ft, $D = 2$ in. and $\delta = 1/4$ in. Slag temperature profiles for these three runs are shown graphically in Figure E-8.

Solutions for Cases 1 through 3 are compared with the constant profile solution in Table E-7. It can be seen that for the same length float chamber (166 ft), variable temperature profiles result in additional slag cooling compared with the constant profile case, with a corresponding increase in the rate of energy recovery. Alternatively, for the same rate of energy recovery, variable temperature profiles result in a shorter float chamber. Also, Case 3 shows that steam can be produced at temperatures greater than 600°F without increasing the length of the chamber or decreasing the energy recovery rate over the base case values.

TABLE E-7. COMPARISON OF SOLUTIONS FOR
CONSTANT AND VARIABLE T_O AND T_W

Case	Slag temperature at $x = 166$ ft, $^{\circ}F$	Cumulative heat recovery rate ($Q_C + Q_R$) at $x = 166$ ft, Btu/hr	Additional energy recovery over base case, %	Length required to cool slag to 800°F, ft	Reduction in length over base case, %
Constant T_O & T_W (base case)	800	1.34×10^8	-	166	-
Case 1	750	1.37×10^8	2.2	147	11
Case 2	670	1.43×10^8	6.7	122	27
Case 3	700	1.41×10^8	5.2	131	21

TABLE E-4. SOLUTION WITH QUADRATIC PROFILES FOR
 T_o AND T_w : $700^{\circ}\text{F} \rightarrow 650^{\circ}\text{F} \rightarrow 500^{\circ}\text{F}$

T_s , $^{\circ}\text{F}$	x , ft	q_c , Btu/hr-ft ²	q_r , Btu/hr-ft ²	Q_c , Btu/hr	Q_r , Btu/hr
2650.	0.8	82558.68	138738.55	1310279.88	2201904.50
2600.	1.6	79971.83	129820.09	2649129.25	4375289.00
2550.	2.5	77419.48	121324.33	4017319.25	6519384.00
2500.	3.4	74901.64	113238.50	5415638.00	8633403.00
2450.	4.4	72418.30	105549.95	6844890.50	10716544.00
2400.	5.5	69969.46	98246.20	8305897.50	12767987.00
2350.	6.6	67555.12	91314.98	9799495.00	14786899.00
2300.	7.8	65175.27	84744.23	11326533.00	16772432.00
2250.	9.0	62829.93	78522.02	12887875.00	18723726.00
2200.	10.3	60519.11	72636.62	14484395.00	20639910.00
2150.	11.7	58242.77	67076.44	16116976.00	22520104.00
2100.	13.2	56000.93	61830.11	17786508.00	24363420.00
2050.	14.8	53793.59	56886.39	19493888.00	26168964.00
2000.	16.5	51620.76	52234.22	21240016.00	27935844.00
1950.	18.3	49482.43	47862.70	23025790.00	29663162.00
1900.	20.2	47378.60	43761.10	24852104.00	31350030.00
1850.	22.3	45309.27	39918.84	26719844.00	32995566.00
1800.	24.5	43274.44	36325.51	28629888.00	34598900.00
1750.	26.9	41274.12	32970.88	30583094.00	36159176.00
1700.	29.4	39308.29	29844.86	32580304.00	37675560.00
1650.	32.1	37376.96	26937.55	34622328.00	39147248.00
1600.	35.1	35480.14	24239.18	36709952.00	40573464.00
1550.	38.2	33617.81	21740.20	38843920.00	41953472.00
1500.	41.7	31789.99	19431.18	41024932.00	43286584.00
1450.	45.4	29996.67	17302.89	43253640.00	44572164.00
1400.	49.4	28237.85	15346.27	45530640.00	45809632.00
1350.	53.8	26513.53	13552.44	47856456.00	46998476.00
1300.	58.6	24823.71	11912.71	50231544.00	48138264.00
1250.	63.8	23168.39	10418.56	52656284.00	49228640.00
1200.	69.6	21547.57	9061.67	55130964.00	50269348.00
1150.	75.9	19961.26	7833.91	57655784.00	51260228.00
1100.	82.9	18409.44	6727.35	60230844.00	52201228.00
1050.	90.7	16892.13	5734.26	62856140.00	53092420.00
1000.	99.3	15409.32	4847.12	65531564.00	53933992.00
950.	109.1	13961.01	4058.62	68256896.00	54726276.00
900.	120.2	12547.20	3361.67	71031824.00	55469740.00
850.	132.8	11167.89	2749.40	73855944.00	56165004.00
800.	147.4	9823.08	2215.16	76728792.00	56812848.00
750.	164.6	8512.77	1752.56	79649912.00	57414232.00
700.	185.1	7236.97	1355.41	82618976.00	57970304.00
650.	210.2	5995.67	1017.78	85636040.00	58482460.00
600.	242.3	4788.86	734.00	88702264.00	58952428.00

TABLE E-5. SOLUTION WITH LINEAR PROFILES
FOR T_o AND T_w : $700^\circ\text{F} \rightarrow 500^\circ\text{F}$

T_s , $^\circ\text{F}$	x , ft	q_c , Btu/hr-ft ²	q_r , Btu/hr-ft ²	Q_c , Btu/hr	Q_r , Btu/hr
2650.	0.8	82910.80	138816.03	1313255.00	2198758.50
2600.	1.6	81014.03	130050.67	2661302.75	4362760.50
2550.	2.5	79117.27	121702.79	4044963.50	6491188.00
2500.	3.4	77220.49	113758.89	5465055.00	8583224.00
2450.	4.4	75323.72	106205.82	6922391.50	10638056.00
2400.	5.4	73426.95	99030.54	8417779.00	12654878.00
2350.	6.5	71530.17	92220.28	9952016.00	14632893.00
2300.	7.6	69633.41	85762.51	11525887.00	16571318.00
2250.	8.8	67736.62	79644.87	13140161.00	18469386.00
2200.	10.1	65839.86	73855.26	14795589.00	20326346.00
2150.	11.4	63943.09	68381.77	16492897.00	22141474.00
2100.	12.8	62046.31	63212.73	18232786.00	23914072.00
2050.	14.3	60149.54	58336.70	20015928.00	25643472.00
2000.	15.8	58252.78	53742.41	21842956.00	27329038.00
1950.	17.5	56356.00	49418.87	23714470.00	28970178.00
1900.	19.3	54459.23	45355.27	25631022.00	30566340.00
1850.	21.1	52562.46	41541.03	27593118.00	32117018.00
1800.	23.1	50665.68	37965.79	29601214.00	33621764.00
1750.	25.2	48768.92	34619.41	31655710.00	35080180.00
1700.	27.5	46872.15	31491.98	33756944.00	36491936.00
1650.	29.8	44975.37	28573.79	35905188.00	37856760.00
1600.	32.4	43078.60	25855.35	38100652.00	39174456.00
1550.	35.1	41181.83	23327.43	40343464.00	40444896.00
1500.	38.0	39285.06	20980.96	42633688.00	41668036.00
1450.	41.2	37388.29	18807.12	44971296.00	42843904.00
1400.	44.5	35491.52	16797.31	47356188.00	43972620.00
1350.	48.1	33594.75	14943.14	49788180.00	45054384.00
1300.	52.0	31697.97	13236.46	52267000.00	46089492.00
1250.	56.3	29801.20	11669.33	54792288.00	47078324.00
1200.	60.9	27904.43	10234.00	57363600.00	48021360.00
1150.	65.9	26007.66	8922.99	59980416.00	48919164.00
1100.	71.4	24110.88	7729.01	62642128.00	49772404.00
1050.	77.5	22214.12	6644.97	65348056.00	50581836.00
1000.	84.3	20317.34	5664.06	68097456.00	51348312.00
950.	91.9	18420.57	4779.62	70889536.00	52072776.00
900.	100.4	16523.80	3985.27	73723472.00	52756276.00
850.	110.3	14627.03	3274.80	76598472.00	53399952.00
800.	121.7	12730.26	2642.26	79513840.00	54005056.00
750.	135.4	10833.49	2081.88	82469168.00	54572984.00
700.	152.1	8936.71	1588.15	85464752.00	55105332.00
650.	173.7	7039.94	1155.75	88502768.00	55604088.00
600.	203.7	5143.17	779.60	91591608.00	56072292.00

TABLE E-6. SOLUTION WITH LINEAR PROFILES
FOR T_o AND T_w : $900^\circ \rightarrow 500^\circ\text{F}$

T_s $^\circ\text{F}$	x , ft	q_c' Btu/hr-ft ²	q_r' Btu/hr-ft ²	Q_c' Btu/hr	Q_r' Btu/hr
2650.	0.8	74619.82	136505.05	1241293.00	2270747.50
2600.	1.7	72922.75	127834.11	2517017.50	4507102.00
2550.	2.6	71225.69	119578.26	3828065.50	6708173.50
2500.	3.6	69528.62	111724.11	5175331.50	8873068.00
2450.	4.6	67831.55	104258.47	6559713.00	11000890.00
2400.	5.7	66134.50	97168.38	7982104.50	13090745.00
2350.	6.8	64437.43	90441.12	9443398.00	15141743.00
2300.	8.0	62740.36	84064.15	10944478.00	17153000.00
2250.	9.3	61043.29	78025.17	12486219.00	19123644.00
2200.	10.6	59346.22	72312.11	14069481.00	21052816.00
2150.	12.0	57649.16	66913.11	15695108.00	22939674.00
2100.	13.5	55952.10	61816.51	17363922.00	24783398.00
2050.	15.1	54255.03	57010.91	19076718.00	26583194.00
2000.	16.8	52557.97	52485.09	20834262.00	28338300.00
1950.	18.5	50860.91	48228.07	22637284.00	30047988.00
1900.	20.4	49163.84	44229.08	24486474.00	31711568.00
1850.	22.4	47466.77	40477.59	26382480.00	33328398.00
1800.	24.5	45769.71	36963.25	28325898.00	34897888.00
1750.	26.8	44072.64	33675.97	30317268.00	36419496.00
1700.	29.2	42375.58	30605.86	32357072.00	37892748.00
1650.	31.8	40678.52	27743.25	34445724.00	39317236.00
1600.	34.5	38981.45	25078.68	36583572.00	40692620.00
1550.	37.5	37284.38	22602.93	38770884.00	42018636.00
1500.	40.6	35587.32	20306.99	41007852.00	43295104.00
1450.	44.0	33890.26	18182.06	43294580.00	44521928.00
1400.	47.6	32193.19	16219.58	45631084.00	45699108.00
1350.	51.5	30496.12	14411.18	48017292.00	46826728.00
1300.	55.7	28799.06	12748.73	50453036.00	47904980.00
1250.	60.3	27101.99	11224.33	52938052.00	48934152.00
1200.	65.3	25404.93	9830.27	55471980.00	49914636.00
1150.	70.8	23707.86	8559.08	58054360.00	50846932.00
1100.	76.7	22010.80	7403.50	60684640.00	51731648.00
1050.	83.3	20313.73	6356.49	63362180.00	52569492.00
1000.	90.6	18616.67	5411.24	66086260.00	53361292.00
950.	98.8	16919.60	4561.15	68856088.00	54107976.00
900.	108.1	15222.54	3799.83	71670824.00	54810584.00
850.	118.6	13525.47	3121.13	74529616.00	55470276.00
800.	130.9	11828.41	2519.10	77431688.00	56088332.00
750.	145.4	10131.34	1988.03	80376464.00	56666172.00
700.	163.1	8434.28	1522.40	83363912.00	57205412.00
650.	185.6	6737.21	1116.94	86395432.00	57708000.00
600.	216.2	5040.15	766.59	89476872.00	58176676.00

Figure E-8. Slag temperature profiles for variable T_o and T_w .

2.2.3 Length Required to Remove Latent Heat of Fusion--

The previous calculations have neglected the latent heat of crystallization of the slag. The latent heat amounts to about 484 kJ/kg (208 Btu/lb) slag and is evolved (assuming crystallization occurs) over the temperature range 1,427°C to 1,260°F (2,600°F to 2,300°F). An approximation to the length of tin bath required to remove the latent heat is derived below for the preferred configuration (W = 20 ft, D = 2 in., $\delta = 1/4$ in.).

The rate of latent heat evolution is:

$$Q_l = \rho_s V \Delta H_f$$

$$Q_l = 180 \text{ lb/ft}^3 \times 1,500 \text{ ft}^3/\text{hr} \times 208 \text{ Btu/lb}$$

$$Q_l = 5.62 \times 10^7 \text{ Btu/hr (16.5 MW)}$$

From Table E-3, at 1,343°C (2,450°F) the heat flux is:

$$q_c + q_r = 74,423 + 106,443 = 1.81 \times 10^5 \text{ Btu/hr-ft}^2$$

Hence, the area required is:

$$A = \frac{Q_l}{q_c + q_r} = \frac{5.62 \times 10^7}{1.81 \times 10^5} = 310.5 \text{ ft}^2 \text{ (28.9 m}^2\text{)}$$

For the desired width (W = 20 ft), the length required is:

$$L_l = \frac{A_l}{W} = \frac{310.5}{20} = 15.5 \text{ ft (4.7 m)}$$

Adding this value to the length required for the removal of sensible heat gives the total length required:

$$L_T = 165.7 + 15.5 \approx 181 \text{ ft (55.2 m)}$$

2.2.4 Comparison of Model System with Commercial Float Glass Systems--

The preferred configuration of the model system is compared with published data on commercial float glass plants in Table E-8.

TABLE E-8. COMPARISON OF MODEL SYSTEM AND
COMMERCIAL FLOAT GLASS SYSTEMS

System	Tin depth, in.	Width of tin bath, ft	Length of bath, ft	Velocity of slag or glass, ft/s
Model	2	22 ^a	180	1.0
Commercial float glass plant	2 to 4	15 to 27	160 to 200	up to 11

^a Assuming 5% clearance on either side of slag sheet.

Commercial float chambers are typically 48.8 m (160 ft) long and 4.6 m (15 ft) to 8.2 m (27 ft) wide.^{36,45} Pilkington's float glass line in England reportedly employs a tin bath 5 cm to 10 cm (2 to 4 in.) deep, 7 m (23 ft) wide, and 51.8 m (170 ft) long.^{28,29,44} A plant operated by Libbey-Owens-Ford employs a bath 61 m (200 ft) in length, and the glass ribbon travels at speeds of 3.0 m/s to 3.3 m/s (10 to 11 ft/s).³⁴ Thus, the configuration of the model system is very similar to the systems used in commercial float glass plants.

2.3 A Closed-Form Solution

A closed-form solution to the transport equations derived in Section 2.1 can be obtained if the radiation term is linearized. For the purpose of the present study, the linearization is accomplished by simply neglecting the radiation term entirely. Although this approximation is obviously unrealistic in the present application, the resulting solution is useful for several reasons. First, the simple closed-form solution allows one to easily comprehend the manner in which the various parameters affect the solution. Second, the solution should give an upper bound on the size of the float chamber, since neglecting radiative heat transfer should outweigh those approximations which tend to underestimate the bath size. Finally, the solution can be used to rigorously justify neglecting conduction in the x-direction in the energy balance.

Referring to Figure E-5, the energy balance is made neglecting the free convection and radiation terms, q_{cv} and q_r . For this case, Equation E-9 becomes:

$$\begin{aligned} \rho_s c_s u \delta W (T_s - T_{ref})|_{x + \Delta x} + q_x \delta W|_{x + \Delta x} \\ + q_c W \Delta x = \rho_s c_s u \delta W (T_s - T_{ref})|_x + q_x \delta W|_x \end{aligned} \quad (E-41)$$

Rearranging gives:

$$\rho_s c_s u \delta \left(\frac{T_s|_{x + \Delta x} - T_s|_x}{\Delta x} \right) + \delta \left(\frac{q_x|_{x + \Delta x} - q_x|_x}{\Delta x} \right) + q_c = 0 \quad (E-42)$$

Taking the limit as $\Delta x \rightarrow 0$ gives:

$$\rho_s c_s u \delta \frac{dT_s}{dx} + \delta \frac{dq_x}{dx} + q_c = 0 \quad (E-43)$$

The two fluxes are given by:^a

$$q_x = -k_s \frac{dT_s}{dx} \quad (E-44)$$

$$q_c = k_t \frac{(T_s - T_o)}{D} \quad (E-45)$$

Thus, Equation E-43 becomes:

$$\frac{d^2 T_s}{dx^2} - \frac{\rho_s c_s u}{k_s} \frac{dT_s}{dx} - \frac{k_t}{k_s \delta D} (T_s - T_o) = 0 \quad (E-46)$$

^aIn this derivation, the expression for q_c corresponding to zero slag thickness is used.

For the case of constant T_o , the boundary conditions are:

$$T_s(0) = T_1 \quad (E-47)$$

$$\lim_{x \rightarrow \infty} T_s(x) = T_o \quad (E-48)$$

It is convenient at this point to nondimensionalize the problem by introducing the following dimensionless variables:

$$\eta = x/D \quad (E-49)$$

$$\theta = \frac{T_s - T_o}{T_1 - T_o} \quad (E-50)$$

Equation E-46 then becomes:

$$\frac{d^2\theta}{d\eta^2} - \frac{\rho_s c_s u D}{k_s} \frac{d\theta}{d\eta} - \frac{k_t D}{k_s \delta} \theta = 0 \quad (E-51)$$

The boundary conditions transform to:

$$\theta(0) = 1 \quad (E-52)$$

$$\lim_{\eta \rightarrow \infty} \theta(\eta) = 0 \quad (E-53)$$

Thus, it can be seen that the dimensionless slag temperature profile depends only on the two dimensionless parameters:

$$N_1 \equiv \frac{\rho_s c_s u D}{k_s} = \frac{\rho_s c_s V D}{k_s \delta W} \quad (E-54)$$

$$N_2 \equiv \frac{k_t D}{k_s \delta} \quad (E-55)$$

Equation E-51 is a linear second-order differential equation whose general solution is:

$$\theta = \hat{C}_1 e^{m_1 \eta} + \hat{C}_2 e^{m_2 \eta} \quad (\text{E-56})$$

where $\hat{C}_1$ and $\hat{C}_2$ are constants and m_1 and m_2 are the roots of the characteristic equation:

$$m^2 - N_1 m - N_2 = 0 \quad (\text{E-57})$$

Thus,

$$m_1 = 1/2 N_1 + 1/2 \sqrt{N_1^2 + 4N_2} \quad (\text{E-58})$$

$$m_2 = 1/2 N_1 - 1/2 \sqrt{N_1^2 + 4N_2} \quad (\text{E-59})$$

Since m_1 is positive, $\hat{C}_1$ must be identically zero in order to satisfy the boundary condition E-53. The boundary condition E-52 then gives:

$$\hat{C}_2 = 1 \quad (\text{E-60})$$

and Equation E-56 becomes:

$$\theta = e^{m_2 \eta} \quad (\text{E-61})$$

Defining

$$\theta_2 = \frac{T_2 - T_o}{T_1 - T_o} \quad (\text{E-62})$$

and

$$\eta_2 = L/D \quad (\text{E-63})$$

where T_2 is the slag temperature at the exit from the float chamber and L is the length of the chamber, Equation E-61 yields:

$$\theta_2 = e^{m_2 \eta_2} \quad (\text{E-64})$$

or

$$\eta_2 = m_2^{-1} \ln \theta_2 \quad (\text{E-65})$$

Substituting for m_2 from Equation E-59 and rearranging gives:

$$\eta_2 = L/D = - \ln \theta \left\{ \frac{N_1}{2N_2} + \sqrt{\left(\frac{N_1}{2N_2}\right)^2 + \frac{1}{N_2}} \right\} \quad (\text{E-66})$$

or

$$L = - D \ln \theta_2 \left\{ \frac{N_1}{2N_2} + \sqrt{\left(\frac{N_1}{2N_2}\right)^2 + \frac{1}{N_2}} \right\} \quad (\text{E-67})$$

In terms of the physical parameters, Equation E-67 is:

$$L = D \ln \theta_2 \left\{ \frac{\rho_s c_s V}{2Wk_t} + \sqrt{\left(\frac{\rho_s c_s V}{2Wk_t}\right)^2 + \frac{k_s \delta}{k_t D}} \right\} \quad (\text{E-68})$$

The data in Table E-1 (for $D = 2$ in. and $\delta = 1/4$ in.) give the following values:

$$N_1 = 43,875$$

$$N_2 = 275$$

$$\frac{N_1}{2N_2} = 79.77$$

$$\left(\frac{N_1}{2N_2}\right)^2 = 6363.69$$

$$\frac{1}{N_2} = 0.0036$$

Clearly, $1/N_2 \ll \left(\frac{N_1}{2N_2}\right)^2$, and Equation E-67 reduces to

$$L = - \frac{DN_1}{N_2} \ln \theta_2 \quad (\text{E-69})$$

or

$$L = - \left(\frac{\rho_s c_s VD}{Wk_t} \right) \ln \theta_2 \quad (\text{E-70})$$

Thus, in this linear approximation, the length is directly proportional to the volumetric slag flow rate and the tin depth, and inversely proportional to the width of the slag sheet. The length does not depend on slag thickness since the resistance to heat transfer through the slag sheet has been neglected. Making an approximation for slag resistance similar to that given in Section 2.1.5 yields:

$$L = - \left(\frac{\rho_s c_s VR}{W} \right) \ln \theta_2 \quad (\text{E-71})$$

where

$$R = \frac{D}{k_t} + \frac{\delta}{k_s} \quad (\text{E-72})$$

Numerical values are listed in Table E-9 for comparison with results in Table E-2 obtained with the nonlinear model. For the preferred system ($W = 20$ ft, $D = 2$ in., $\delta = 1/4$ in.), the nonlinear model gives $L \approx 50.6$ m (166 ft), while the linearized model gives $L \approx 128$ m (420 ft). The latter value can be considered an upper bound on the required length.

2.4 Summary and Justification of Major Assumptions

2.4.1 Summary of Assumptions--

The major assumptions employed in modelling the float process are summarized here for convenience.

TABLE E-9. DIMENSIONS OF FLOAT CHAMBER COMPUTED WITH LINEARIZED MODEL, 6.1 m (20 ft) SLAG WIDTH

Depth of tin bath, in.	Thickness of slag sheet, in.	Length of tin bath, ft
2	0	80
	1/4	423
	1/2	765
	1	1,450
3	0	120
	1/4	462
	1/2	805
	1	1,490
4	0	160
	1/4	502
	1/2	845
	1	1,530
6	0	239
	1/4	582
	1/2	925
	1	1,610

- a) Conduction through slag in flow direction is negligible compared to heat transfer by bulk flow.
- b) Free convective heat transfer from upper slag surface is negligible compared to radiative transfer and conductive transfer through tin.
- c) Conduction through tin in flow direction (x-direction) is negligible compared to conduction in transverse direction (y-direction).

- d) Physical properties of tin and slag are constant, independent of temperature.
- e) Conditions are uniform in the lateral direction (z-direction).
- f) Forced convection in the tin is negligible compared to conduction.
- g) Developing slag flow in the entrance region of the tin bath is neglected.
- h) Slag temperature is uniform in the transverse direction (y-direction).
- i) The temperature profile is specified at bottom of tin ($y = 0$).

Each of the above assumptions is justified and/or discussed in the following subsections.

2.4.2 Conduction Through Slag in Flow Direction Is Negligible--
Neglecting conduction in the flow direction can be justified by reference to the linearized model discussed in Section 2.3. If q_x is set equal to zero, Equation E-43 becomes:

$$\rho_s c_s u \delta \frac{dT_s}{dx} + \frac{k_t}{D} (T_s - T_o) = 0 \quad (E-73)$$

In terms of dimensionless variables, Equation E-73 reads:

$$\frac{\rho_s c_s u \delta}{D} \frac{d\theta}{d\eta} + \frac{k_t \theta}{D} = 0 \quad (E-74)$$

Rearranging gives:

$$\frac{d\theta}{d\eta} + \frac{k_t}{\rho_s c_s u \delta} \theta = 0 \quad (\text{E-75})$$

or

$$\frac{d\theta}{d\eta} + \frac{N_2}{N_1} \theta = 0 \quad (\text{E-76})$$

The solution to Equation E-76 subject to the boundary condition $\theta(0) = 1$ is:

$$\theta = e^{-(N_2/N_1)\eta} \quad (\text{E-77})$$

The length of the float chamber is found by taking logarithms of E-77 and evaluating at the end of the bath:

$$\eta_2 = L/D = - \frac{N_1}{N_2} \ln \theta_2 \quad (\text{E-78})$$

$$L = - \frac{DN_1}{N_2} \ln \theta_2 \quad (\text{E-79})$$

Equation E-79 is identical to Equation E-69 which was obtained in Section 2.3 by applying the condition $1/N_2 \ll \left(\frac{N_1}{2N_2}\right)^2$. Thus, neglecting conduction through the slag in the flow direction is equivalent to this condition, which was shown to hold in Section 2.3.

2.4.3 Free Convection from Slag Is Negligible--

The free convective heat flux is estimated using the following empirical equation for free convection in enclosed horizontal air spaces:⁴⁶

$$\frac{k_e}{k} = 0.068 \text{ Gr}^{1/3} \quad \text{for } \text{Gr} > 4 \times 10^5 \quad (\text{E-80})$$

where k_e = "effective" thermal conductivity
 k = thermal conductivity of air

The Grashof number, Gr , is given by :

$$Gr = \frac{g \beta \Delta T \delta^3}{\nu^2} \quad (E-81)$$

where g = gravitational constant
 β = volume coefficient of expansion, approximately equal to inverse of absolute air temperature
 ΔT = temperature difference between two planes
 δ = distance between two planes
 ν = kinematic viscosity of air

Assuming $\delta = 0.3$ m (1 ft) and an upper surface temperature of 343°C (650°F), the Grashof number is approximately 1.1×10^7 at both the entrance and exit of the float chamber. Hence, Equation E-80 gives:

$$k_e = 15.1 k \quad (E-82)$$

and

$$q_{cv} = k_e \frac{\Delta T}{\delta} \quad (E-83)$$

$$q_{cv} = 15.1 k \Delta T \quad (E-84)$$

At the bath entrance, the conductivity of air is 0.067 $\text{W/m}\cdot\text{K}$ (0.039 $\text{Btu/hr}\cdot\text{ft}\cdot^\circ\text{F}$) and ΔT is $1,121^\circ\text{C}$ ($2,050^\circ\text{F}$). Hence:

$$q_{cv} = 15.1 \times 0.039 \times 2,050 \quad (E-85)$$

$$q_{cv} \approx 1,200 \text{ Btu/hr}\cdot\text{ft}^2\cdot^\circ\text{F} \quad (6,800 \text{ W/m}^2\cdot\text{K}) \quad (E-86)$$

At the bath exit, $k = 0.047$ $\text{W/m}\cdot\text{K}$ (0.027 $\text{Btu/hr}\cdot\text{ft}\cdot^\circ\text{F}$) and $\Delta T = 66^\circ\text{C}$ (150°F). Hence:

$$q_{cv} = 15.1 \times 0.027 \times 150 \quad (E-87)$$

$$q_{cv} \approx 60 \text{ Btu/hr-ft}^2\text{-}^\circ\text{F} \text{ (340 W/m}\cdot\text{K)} \quad (E-88)$$

The fluxes due to radiation, conduction and free convection at the entrance and exit ends of the float chamber are listed in Table E-10. Clearly, the free convection term is negligible compared with the radiation and conduction terms.

TABLE E-10. COMPARISON OF HEAT TRANSFER BY RADIATION, CONDUCTION AND FREE CONVECTION

Heat transfer mechanism	Flux, Btu/hr-ft ² - °F	
	At bath entrance	At bath exit
Radiation	139,200	1,780
Conduction (D = 2 in., $\delta = 1/4$ in.)	82,800	5,240
Conduction (D = 6 in., $\delta = 1/4$ in.)	50,600	3,200
Free convection	1,200	60

2.4.4 Conduction Through Tin in Flow Direction Is Negligible-- Referring to Table E-3, in the first section of the float chamber the temperature gradient in the flow direction (x-direction) is approximately

$$\frac{\Delta T}{\Delta x} = \frac{50}{0.8} \approx 63^\circ\text{F/ft} \text{ (115}^\circ\text{C/m)}$$

In the last section of the bath:

$$\frac{\Delta T}{\Delta x} = \frac{50}{25.3} \approx 2^\circ\text{F/ft} \text{ (4}^\circ\text{C/m)}$$

The corresponding gradients in the y-direction are 20,800°C/m (11,400°F/ft) and 1,100°C/m (600°F/ft). Thus, the gradient in the y-direction is more than two orders of magnitude greater than

the gradient in the x-direction. Since the fluxes are proportional to the temperature gradients, the flux in the x-direction can be neglected compared to that in the y-direction.

2.4.5 Physical Properties Are Constant--

Physical properties of tin and slag were assumed constant primarily because data on the temperature dependence of slag properties were not available. If available, these data could be incorporated into the model with little difficulty.

2.4.6 Conditions Are Uniform in the z-Direction--

Neglecting heat transfer in the z-direction can be justified on the grounds that temperature gradients are much greater in the y-direction than in the z-direction. Data in Reference 47 indicate a maximum temperature change of 66°C (150°F) from the center to the edge of the float chamber in the entrance region. For a width of 6.1 m (20 ft), the average temperature gradient in the z-direction would be at most 27°C/m (15°F/ft) compared with $20,800^{\circ}\text{C/m}$ ($11,400^{\circ}\text{F/ft}$) in the y-direction.

2.4.7 Forced Convection in Tin Is Negligible--

As noted in Section 2.1, this assumption is justified on the basis that the tin bath for the present application will be designed so as to minimize convective transport of heat in the flow direction. This design feature is necessary in order to minimize the size of the float chamber and the amount of tin required (both represent major capital expense).

2.4.8 Developing Slag Flow in the Entrance Region Is Neglected--

Slag flow in the entrance region of the float chamber is neglected on the basis that this region is small compared with the total length of the chamber.

2.4.9 Slag Temperature Is Uniform in y-Direction--

As noted previously, this assumption is strictly valid only in the limit of zero slag thickness. It becomes progressively worse as the slag resistance to heat transfer increases relative to the tin resistance and radiative resistance. For 6.4 mm (1/4 in.) thick slag and a 5.1 cm (2 in.) deep tin layer, the ratio of slag to tin resistance is:

$$\frac{R_{\text{slag}}}{R_{\text{tin}}} \approx \frac{1/2 \delta k_t}{D k_s} = \frac{1/2 \times 1/4 \times 22}{2 \times 0.64} \approx 2.2$$

Hence, it can be concluded that the assumption of uniform slag temperature in the y-direction is rather poor in the present application, and a two-dimensional model would be more appropriate than the one-dimensional model employed in the analysis.

It should be noted, however, that the calculated float chamber size is in good agreement with existing processes and that lower and upper bounds on the size of the float chamber have been established by the solution for zero slag thickness and the solution neglecting radiation. Thus, for the case $\delta = 6.4$ mm (1/4 in.) and $D = 5.1$ cm (2 in.), the calculated length of approximately 51.8 m (170 ft) is bounded below by (approximately) 21.3 m (70 ft) and above by (approximately) 128 m (420 ft). Although this is a rather wide range, it does indicate the extent of the difference in float chamber size that could result from a more rigorous analysis.

2.4.10 Temperature Profile Is Specified at $y = 0$ --

Normally, a model such as the one employed in this analysis would be solved subject to a convective boundary condition at $y = 0$. This boundary condition would specify the rate of heat transfer to the cooling medium. However, the present problem is complicated by the auxiliary condition that the tin cannot be allowed to solidify. In order to impose this condition, the temperature

- profile at $y = 0$ was specified. It is shown in Section 2.5 that sufficient cooling surface area is available to handle the resulting heat load. Therefore, the specified temperature profiles are theoretically achievable.

2.5 Heat-Transfer Calculations for Float Chamber Process

In this section, the results obtained from the mathematical model of the float chamber are used to calculate the surface area required to generate steam from the recovered heat. The calculation is made for the results given in Table E-3, which correspond to the following conditions:

- width of slag sheet = 6.1 m (20 ft)
- tin depth = 5.1 cm (2 in.)
- slag thickness = 6.4 mm (1/4 in.)
- temperature at bottom of tin = 371°C (700°F)
- overhead tube wall temperature = 343°C (650°F)

2.5.1 Heat-Transfer Coefficients^a--

2.5.1.1 Piping Configuration--Steam piping is assumed to be 1-1/2 in. schedule 80 seamless carbon steel pipe. Pressure drop calculations presented in Section 2.5.7 show that this size pipe is suitable for the present application. To prevent corrosion by molten tin, all piping is assumed to be embedded in graphite blocks of standard 4 in. by 10 in. cross section as illustrated in Figure E-9. The blocks are 5 ft in length.

Each of the pipes is assumed to extend the entire width of the float chamber. For 1-1/2 in. schedule 80 pipe, the inside diameter is 1.500 in., the outside diameter is 1.900 in., and the

^aSince standard equipment is specified in most cases, English units are extensively used for equipment identification. Refer to conversion tables in Appendix N for conversion of English to metric units.

Figure E-9. Schematic of graphite block with embedded steam pipes.

inside surface area is 0.3927 ft^2 per linear ft.⁴⁸ Thus, the ratio of graphite surface area to inside pipe surface area is:

$$\frac{\text{graphite area}}{\text{inside pipe area}} = \frac{10/12 \times L}{0.3927 \times 3 \times L} = 0.71$$

where L = the length of the pipe

For the present calculation, superheated steam at 3.10 MPa (450 psia) and 316°C (600°F) is assumed to be generated in two steps. First, saturated steam at 3.10 MPa (450 psia) is produced and separated from the liquid phase in a steam drum. Then the saturated steam is passed through the float chamber to generate superheated steam. The saturated water from the steam drum is mixed with feed water and recycled through the float chamber. For the present calculations, the flow pattern for the water and steam relative to the slag is assumed to be strictly crossflow.

The pipes in which saturated steam is produced are collectively referred to as the boiling section, and those in which superheated steam is produced are termed the superheating section. However, the two sets of pipes are assumed to be integrated along the length of the float chamber.

2.5.1.2 Heat-Transfer Coefficient for Boiling Section--In the boiling section, 15% of the flow is assumed to be converted to steam.⁴⁹ Since 85% of the flow is recycled, it is assumed that

boiling heat transfer takes place over essentially the entire length of the pipes. Due to the large film coefficient in the boiling regime,^{4,9} the major contributions to heat-transfer resistance are the graphite block, the pipe wall, and the thermal contact resistance at the pipe-graphite interface.

For the present estimate, it is assumed that good thermal contact can be maintained between the pipe and graphite (e.g., by means of a thermal contact cement) so that the thermal contact resistance is small. The actual piping configuration (Figure E-9) is approximated by a hollow graphite rod with the steel pipe passing through the center as illustrated in Figure E-10. In this figure, r_0 is the inside pipe radius (0.75 in.), r_1 is the outside pipe radius (0.95 in.), and r_2 is the radius of the graphite rod (2 in.).

Figure E-10. Schematic of approximate piping configuration used to estimate heat-transfer coefficient.

The combined resistance in the pipe wall and graphite is given by:

$$R = \frac{\ln(r_1/r_0)}{2\pi k_{st} L} + \frac{\ln(r_2/r_1)}{2\pi k_{gr} L} \quad (E-89)$$

or

$$R = \frac{1}{2\pi L} \left[\frac{\ln(r_1/r_0)}{k_{st}} + \frac{\ln(r_2/r_1)}{k_{gr}} \right] \quad (E-90)$$

where k_{st} and k_{gr} denote the thermal conductivities of steel and graphite, respectively, and L is the length of the pipe.

The rate of heat transfer is given by:

$$Q = \frac{\Delta T}{R} = U \cdot 2\pi r_0 L \Delta T \quad (E-91)$$

where ΔT is the temperature difference between the fluid in the pipe and the outside graphite surface, and U is the overall heat-transfer coefficient based on the inside pipe surface area. Combining Equations E-90 and E-91 yields the following expression for U :

$$U = \frac{1}{r_0 \left[\frac{\ln(r_1/r_0)}{k_{st}} + \frac{\ln(r_2/r_1)}{k_{gr}} \right]} \quad (E-92)$$

Substituting the appropriate values for r_0 , r_1 , and r_2 , and the thermal conductivities (26 and 85 Btu/hr-ft-°F for steel and graphite, respectively) gives:

$$U = 897 \cong 900 \text{ Btu/hr-ft}^2\text{-°F} \quad (5.1 \text{ kW/m}^2\text{·K}) \quad (E-93)$$

Based on this result, a design coefficient of 2.8 kW/m²·K (500 Btu/hr-ft²-°F) was chosen for the boiling section. This lower value was selected in order to allow for fouling of the tubes and also to take into account the fact that in the actual system the bottom half of the pipe is exposed to a lower temperature than the top half.

2.5.1.3 Heat-Transfer Coefficient for Superheating Section--In the superheating section, the film resistance was assumed to be controlling, and a design coefficient of $0.28 \text{ kW/m}^2 \cdot \text{K}$ ($50 \text{ Btu/hr-ft}^2 \cdot ^\circ\text{F}$) was chosen. This value was based on the correlations given in Figures 4-8 and 4-9 and Equation 18, pages 4-8 and 4-9 of Reference 49.

2.5.2 Surface Area Required for Conductive Heat Transfer--

2.5.2.1 Conductive Heat-Transfer Rate--From Table E-3, the cumulative rate of heat transfer from the slag by conduction is $7.65 \times 10^7 \text{ Btu/hr}$. To this amount is added the rate of latent heat evolution ($5.66 \times 10^7 \text{ Btu/hr}$) scaled by the ratio of the conductive flux to the total flux at $2,450^\circ\text{F}$. The total rate of heat transfer by conduction is thus:

$$Q_{\text{ct}} = 7.65 \times 10^7 + 5.66 \times 10^7 \times \frac{0.744}{1.81}$$

$$Q_{\text{ct}} = 9.98 \times 10^7 \text{ Btu/hr (29.2 MW)}$$

2.5.2.2 Steam Rate--Boiler feed water (condensate) is assumed to be available at 38°C (100°F). Saturated steam at 3.10 MPa (450 psia) is produced in the boiling section, and steam at 3.10 MPa (450 psia) and 316°C (600°F) is produced in the superheating section. The relevant enthalpies are as follows:

$$H_{\text{saturated water @ } 100^\circ\text{F}} = 68 \text{ Btu/lb}$$

$$H_{\text{saturated steam @ } 450 \text{ psia}} = 1,205 \text{ Btu/lb}$$

$$H_{\text{superheated steam @ } \begin{cases} 450 \text{ psia} \\ 600^\circ\text{F} \end{cases}} = 1,303 \text{ Btu/lb}$$

The total enthalphy change for the water is:

$$\Delta H_{\text{TOT}} = 1,303 - 68 = 1,235 \text{ Btu/lb}$$

The rate of steam generation, assuming no heat losses, is:

$$\begin{aligned}\text{Steam rate} &= \frac{Q_{ct}}{\Delta H_{TOT}} \\ &= \frac{9.98 \times 10^7 \text{ Btu/hr}}{1,235 \text{ Btu/lb}} \\ &= 8.08 \times 10^4 \text{ lb/hr}\end{aligned}$$

2.5.2.3 Area Required for Boiling Section--

$$\Delta H = 1,205 - 68 = 1,137 \text{ Btu/lb}$$

$$\begin{aligned}Q &= 1,137 \text{ Btu/lb} \times 8.08 \times 10^4 \text{ lb/hr} \\ &= 9.19 \times 10^7 \text{ Btu/hr}\end{aligned}$$

$$Q = U A \Delta T$$

$$A = \frac{Q}{U \Delta T} = \frac{9.19 \times 10^7}{500(700 - 456)}$$

$$A = 753 \text{ ft}^2 = \text{inside tube surface area}$$

$$\text{Number of tubes} = \frac{753}{0.393 \times 20} = 96$$

$$\text{Graphite surface area} = 0.71 \times 753 = 545 \text{ ft}^2$$

2.5.2.4 Area Required for Superheating Section^a--

$$\Delta H = 1,303 - 1,205 = 98 \text{ Btu/lb}$$

^aThe mean temperature difference, ΔT_m , is the logarithmic mean of the inlet and outlet temperature differences. For convenience, the arithmetic mean temperature difference is used in these calculations. The latter is a good approximation to the logarithmic mean in the present application.

$$Q = 98 \text{ Btu/lb} \times 8.08 \times 10^4 \text{ lb/hr}$$

$$= 7.92 \times 10^6 \text{ Btu/hr}$$

$$A = \frac{Q}{U \Delta T_m} = \frac{7.92 \times 10^6}{50 \left[\frac{(700 - 456) + (700 - 600)}{2} \right]}$$

$$A = \frac{7.92 \times 10^6}{50 \times 172} = 921 \text{ ft}^2$$

$$\text{Number of tubes} = \frac{921}{0.393 \times 20} = 117$$

$$\text{Graphite surface area} = 0.71 \times 921 = 654 \text{ ft}^2$$

2.5.2.5 Total Requirements for Conduction--

$$\text{Number of tubes} = 213$$

$$\text{Inside tube surface area} = 156 \text{ m}^2 (1,670 \text{ ft}^2)$$

$$\text{Graphite surface area} = 110 \text{ m}^2 (1,190 \text{ ft}^2)$$

$$\text{Graphite surface area/slag surface area} = 0.33$$

2.5.3 Surface Area Required for Radiative Heat Transfer--

2.5.3.1 Radiative Heat-Transfer Rate--From Table E-3, the total rate of heat transfer from the slag by radiation is:

$$\begin{aligned} Q_{rt} &= 5.71 \times 10^7 + 5.66 \times 10^7 \times \frac{1.06}{1.81} \\ &= 9.02 \times 10^7 \text{ Btu/hr} \end{aligned}$$

2.5.3.2 Steam Rate--

$$\Delta H_{tot} = 1,235 \text{ Btu/lb}$$

$$\begin{aligned}
 \text{Steam rate} &= Q_{rt} / \Delta H_{tot} \\
 &= \frac{9.02 \times 10^7 \text{ Btu/hr}}{1.235 \times 10^3 \text{ Btu/lb}} \\
 &= 7.30 \times 10^4 \text{ lb/hr}
 \end{aligned}$$

2.5.3.3 Area Required for Boiling Section--

$$\Delta H = 1,137 \text{ Btu/lb}$$

$$\begin{aligned}
 Q &= 1,137 \text{ Btu/lb} \times 7.3 \times 10^4 \text{ lb/hr} \\
 &= 8.30 \times 10^7 \text{ Btu/hr}
 \end{aligned}$$

$$Q = U A \Delta T$$

$$A = \frac{Q}{U \Delta T} = \frac{8.30 \times 10^7}{500(650-456)} = 856 \text{ ft}^2$$

$$\text{Number of tubes} = \frac{856}{0.393 \times 20} = 109$$

$$\text{Graphite surface area} = 0.71 \times 856 = 608 \text{ ft}^2$$

2.5.3.4 Area Required for Superheating Section--

$$\Delta H = 98 \text{ Btu/lb}$$

$$Q = 98 \times 7.3 \times 10^4 = 7.15 \times 10^6 \text{ Btu/hr}$$

$$A = \frac{Q}{U \Delta T_m} = \frac{7.15 \times 10^6}{50 \left[\frac{(650 - 456) + (650 - 600)}{2} \right]}$$

$$A = \frac{7.15 \times 10^6}{50 \times 122} = 1,170 \text{ ft}^2$$

$$\text{Number of tubes} = \frac{1,170}{0.393 \times 20} = 149$$

$$\text{Graphite surface area} = 0.71 \times 1,170 = 831 \text{ ft}^2$$

2.5.3.5 Total Requirements for Radiation--

Number of tubes = 258

Graphite surface area = 1,440 ft²

Graphite surface area/slag surface area = 0.40

2.5.4 Summary of Calculations--

The results of the calculations performed in the preceding sections are summarized in Table E-11.

TABLE E-11. SUMMARY OF HEAT TRANSFER CALCULATIONS

Item	Specifications		
	Conduction	Radiation	Total
No. of tubes (1-1/2 in., schedule 80, 20 ft long)	213	258	471
Inside tube - surface area, m ² (ft ²)	156 (1,670)	188 (2,030)	344 (3,700)
Graphite surface area, m ² (ft ²)	110 (1,190)	134 (1,440)	244 (2,630)
Graphite surface area/slag surface area	0.33	0.40	--
Energy recovered, MW (Btu/hr)	29.3 (9.98 x 10 ⁷)	26.4 (9.02 x 10 ⁷)	55.7 (1.9 x 10 ⁸)
Steam rate, kg/s (lb/hr) (450 psia, 600°F)	10.2 (80,800)	9.20 (73,000)	19.4 (153,800)

2.5.5 Area Required in First Section of Float Chamber--

The heat-transfer surface area required in the first section of the float chamber is calculated and compared with the available area referred to as slag surface area. The calculation assumes that only boiling heat transfer (no superheating) takes place in this initial section. It is shown that sufficient areas for conduction through tin and radiation to the float chamber ceiling are available to handle the heat load. However, the area needed for radiation heat transfer closely matches the area available. Therefore, in order to facilitate radiation heat removal from the first section of the float chamber, the radiation tubes must be designed for boiling heat transfer.

2.5.5.1 Conduction-- Q_{ct} equals 1.31×10^6 Btu/hr from Table E-3.

$$A_c = \frac{Q}{\Delta T} = \frac{1.31 \times 10^6}{500(700 - 456)} = 10.7 \text{ ft}^2$$

$$A_{c, \text{ graphite}} = 0.71 \times 10.7 = 7.6 \text{ ft}^2$$

$$A_{\text{slag}} = 0.8 \times 20 = 16 \text{ ft}^2$$

Graphite surface area/slag surface area = 0.48

2.5.5.2 Radiation-- Q_{rt} equals 2.20×10^6 Btu/hr from Table E-3.

$$A_r = \frac{2.20 \times 10^6}{500(650 - 456)} = 22.7 \text{ ft}^2$$

$$A_{r, \text{ graphite}} = 0.71 \times 22.7 = 16.1 \text{ ft}^2$$

Graphite surface area/slag surface area = 1.0

2.5.6 Area Required in Final Section of Float Chamber--

The heat-transfer surface area required in the final section of the float chamber is calculated for comparison with that required in the initial section. It is assumed that only superheating (no boiling) takes place in the final section. The available area exceeds the area needed to handle the heat load.

2.5.6.1 Conduction-- Q_{ct} equals 2.66×10^6 Btu/hr from Table E-3.

$$A_c = \frac{2.66 \times 10^6}{50 \Delta T_m} = \frac{2.66 \times 10^2}{50 \times 172} = 309 \text{ ft}^2$$

$$A_{c, \text{ graphite}} = 219 \text{ ft}^2$$

$$A_{\text{slag}} = 25.3 \times 20 = 506 \text{ ft}^2$$

Graphite surface area/slag surface area = 0.43

2.5.6.2 Radiation-- Q_{rt} equals 9.02×10^5 Btu/hr from Table E-3.

$$A_r = \frac{9.02 \times 10^5}{50 \times \Delta T_m} = \frac{9.02 \times 10^5}{50 \times 112} = 148 \text{ ft}^2$$

$$A_{r, \text{ graphite}} = 105 \text{ ft}^2$$

Graphite surface area/slag surface area = 0.21

2.5.7 Pressure Drop Calculations--

Pressure drops and Reynolds numbers in the boiling and superheating sections are calculated and found to be reasonable. The calculations are performed only for heat transfer by conduction since the flow rates and piping configurations are similar for radiation and conduction.

As indicated in Section 2.2.2, it may be advantageous to operate the system in a semicountercurrent flow pattern. In this case, multiple tube passes would be required. The calculations show that pressure drops in both boiling and superheating sections are reasonable for up to two tube passes. Up to four passes could be used in the superheating section [$\Delta P \approx 6$ psi for four passes (calculation not shown)]. Additional passes or long runs between passes (for example, from cold end to hot end of float chamber) would probably require larger diameter pipe.

2.5.7.1 Superheating Section--

Number of tubes = 117

Cross-sectional area per tube = 1.767 in.^2

Steam rate = 80,800 lb/hr

For One Pass

$$G = \frac{80,800 \times 144}{117 \times 1.767} = 56,280 \frac{\text{lb}}{\text{hr-ft}^2}$$

$$\mu \approx 0.05 \text{ lb/ft-hr}$$

$$Re = \frac{G\hat{d}}{\mu} = \frac{56,280 \times 1.5}{0.05 \times 12} = 1.41 \times 10^5$$

where G = mass flux

$\hat{d}$ = pipe diameter

μ = viscosity

Re = Reynolds number

Relative roughness:

$$\epsilon/d \approx 0.00012$$

Friction factor:

$$f \approx 0.017$$

Pressure drop:

$$\Delta P = \frac{fL}{\hat{d}} v \left(\frac{G}{10^5} \right)^2$$

where v = specific volume of stream $\approx 1.5 \text{ ft}^3/\text{lb}$

$$\Delta P = \frac{0.017 \times 20 \times 1.5 (0.563)^2}{1.5}$$

$$\Delta P = 0.1 \text{ psi}$$

For Two Passes

$$G = 56,280 \times 2 = 1.13 \times 10^5$$

$$Re = 2.82 \times 10^5$$

$$f \approx 0.016$$

$$\Delta P = \frac{0.016 \times 40 \times 1.5 (1.13)^2}{1.5}$$

$$= 0.8 \text{ psi}$$

2.5.7.2 Boiling Section--Assuming 15% of water is vaporized, flow rate is:

$$W = \frac{80,800}{0.15} = 5.39 \times 10^5 \text{ lb/hr}$$

Number of tubes equals 96. Assuming single pass,

$$G = \frac{5.39 \times 10^5 \times 144}{96 \times 1.767} = 4.58 \times 10^5 \frac{\text{lb}}{\text{hr-ft}^2}$$

$$Re = \frac{G \hat{d}}{\mu} = \frac{4.58 \times 10^5 \times 1.5}{0.29 \times 12} = 1.97 \times 10^5$$

$$f \cong 0.0165$$

$$v = 0.0195 \text{ ft}^3/\text{lb} \text{ for saturated water at 450 psia}$$

$$v = 1.012 \text{ ft}^3/\text{lb} \text{ for saturated steam at 450 psia}$$

Using a weighted average specific volume for the steam-water mixture gives:

$$\begin{aligned} v_{\text{wtd}} &= 0.0195 \times 0.85 + 1.012 \times 0.15 \\ &= 0.17 \text{ ft}^3/\text{lb} \end{aligned}$$

$$\begin{aligned} \Delta P &= \frac{fLv}{\hat{d}} \left(\frac{G}{10^5} \right)^2 \\ &= \frac{0.0165 \times 20 \times 0.17}{1.5} (4.58)^2 \end{aligned}$$

$$\Delta P \cong 0.8 \text{ psi}$$

Assuming Two Passes:

$$G = 9.16 \times 10^5$$

$$Re \cong 4 \times 10^5$$

$$f \cong 0.015$$

$$\Delta P = \frac{0.015 \times 40 \times 0.17}{1.5} (9.16)^2$$

$$\Delta P \cong 6 \text{ psi}$$

2.5.8 Air vs. Steam as a Heat-Transfer Medium--

The option to produce hot air rather than steam in the float chamber was eliminated due to the low heat-transfer coefficient associated with air flowing through the pipes. A heat-transfer coefficient of only 57 to 170 W/m²·K (10 to 30 Btu/hr-ft² °F) could be expected for air, compared to 2,800 W/m²·K (500 Btu/hr-ft² °F) for steam in the superheating section (see Section 2.5.1.3). Thus, there would be insufficient surface area available in the hot end of the float chamber to handle the high heat fluxes with such a low coefficient of heat transfer. The result would be to increase the length of the float chamber. Since the float chamber (including the tin) is a major capital cost item (refer to Appendix F), an increase in the size of the float chamber would result in a corresponding increase in capital cost for the process. Hence, it was concluded that the hot-air option would be less attractive economically than the steam option.

2.6 Amount of Refractory Material Required for Float Chamber

Assume 1-ft-thick refractory lining all around bath. Bath dimensions: 22 ft wide x 180 ft long.

Top and Bottom Volume

$$\begin{aligned} V &= 1 \times (22 + 2) \times (180 + 2) = 4,368 \text{ ft}^3 \text{ for each} \\ &= 8,736 \text{ ft}^3 \text{ for both} \end{aligned}$$

Front and Back

Assume 2-ft height for chamber.

$$V = 1 \times 2 \times 22 = 44 \text{ ft}^3 \text{ for each}$$

$$= 88 \text{ ft}^3 \text{ for both}$$

Sides

$$V = 1 \times 2 \times 180 = 360 \text{ ft}^3 \text{ for each}$$

$$= 720 \text{ ft}^3 \text{ for both}$$

Bottom Area Between Graphite Blocks

$$V = 0.67 \times \frac{4}{12} \times 22 \times 180 = 884 \text{ ft}^3$$

Top Area Between Graphite Blocks

$$V \cong (0.61 \times \frac{4}{12} \times 22 \times 180) \times 1/2 = 403 \text{ ft}^3$$

Total Volume: $10,831 \text{ ft}^3 \cong 10,800 \text{ ft}^3$

2.7 Startup Boiler Estimate

2.7.1 Latent Heat of Tin⁵⁰--

$$660 \text{ ft}^3 \times (6.5 \times 62.4) \frac{\text{lb}}{\text{ft}^3} = 2.68 \times 10^5 \text{ lb tin}$$

$$\Delta H = 14.2 \frac{\text{Cal}}{\text{g}} \times 454 \frac{\text{g}}{\text{lb}} \times \frac{1}{252} \frac{\text{Btu}}{\text{Cal}} = 25.6 \frac{\text{Btu}}{\text{lb}}$$

$$2.68 \times 10^5 \text{ lb} \times 25.6 \frac{\text{Btu}}{\text{lb}} = 6.86 \times 10^7 \text{ Btu}$$

2.7.2 Sensible Heat of Tin⁵⁰ (Heat from 50°F → 500°F)--

$$\begin{aligned}\Delta H &= 0.053 \text{ Btu/lb } ^\circ\text{F} \times 2.68 \times 10^5 \text{ lb} \times 450^\circ\text{F} \\ &= 6.39 \times 10^6 \text{ Btu}\end{aligned}$$

2.7.3 Sensible Heat of Refractory (50°F → 500°F)--

$$\begin{aligned}10,800 \text{ ft}^3 \times 180 \frac{\text{lb}}{\text{ft}^3} &\cong 1.9 \times 10^6 \text{ lb refractory} \\ 1.9 \times 10^6 \text{ lb} \times 0.3 \frac{\text{Btu}}{\text{lb-}^\circ\text{F}} \times 450^\circ\text{F} &\cong 2.57 \times 10^8 \text{ Btu}\end{aligned}$$

2.7.4 Sensible Heat of Graphite (50°F → 500°F)--

Assume solid blocks and $(0.33 + 0.39) = 0.72 \times$ surface area of slag:

$$\begin{aligned}\text{Wt. graphite} &= \left[\frac{4}{12} \times 24 \times 180 \times 0.72 \right] \text{ ft}^3 \times \rho \\ &= 1,037 \text{ ft}^3 \times 1.7^a \times 62.4 \\ &= 110,000 \text{ lb} \\ \Delta H &= 110,000 \text{ lb} \times 0.254 \frac{\text{Btu}}{\text{lb}^\circ\text{F}} \times 450^\circ\text{F} \\ &= 1.26 \times 10^7 \text{ Btu}\end{aligned}$$

2.7.5 Total Energy Required--

Neglecting other components, total energy required is:

^aSpecific gravity of blocks quoted by Union Carbide.

$$\begin{array}{r}
 0.686 \times 10^8 \\
 0.064 \times 10^8 \\
 2.570 \times 10^8 \\
 \underline{0.126 \times 10^8} \\
 3.5 \times 10^8 \text{ Btu}
 \end{array}$$

Assuming 24-hr startup period,

$$Q = \frac{3.5 \times 10^8 \text{ Btu}}{24 \text{ hr}} = 1.44 \times 10^7 \text{ Btu/hr}$$

2.7.6 Boiler Horsepower--

$$\text{Watt} = \text{Btu/hr} \times 0.2929 = \text{Boiler hp} \times 9.81 \times 10^3$$

$$\text{Boiler horsepower} = 3 \times 10^{-5} \times \text{Btu/hr}$$

$$Q = 430 \text{ Bhp}$$

2.8 Estimate of Tin Loss by Penetration Into Slag

Two estimates are presented based on data on tin penetration into float glass given in References 36 and 51. It is assumed that the tin content of the slag will be essentially the same as that for float glass. This factor has been confirmed experimentally at MRC. In the experiment, tin metal was melted into a button in a carbon crucible and then covered with phosphorus slag in either carbon or alumina crucibles. After 3 minutes at 800°C (1,472°F), the samples were heated for 5 or 10 minutes at about 1,400°C (2,552°F), cooled for 3 minutes at 800°C (1,472°F), and air cooled. Upon cooling, the slag/tin samples were sectioned lengthwise through both the tin and the slag portions. The crosssections were polished and the sections of slag nearest the tin were analyzed for tin using electron probe analysis. In general, the samples prepared in carbon crucibles (reducing atmosphere) showed lower tin concentrations than those found in

samples prepared in alumina crucibles (oxidizing atmosphere). Tin concentrations in carbon crucible samples determined over a distance of 30 μm did not exceed an average of 1.1% by weight. Tin concentrations in alumina crucible samples over the same distance did not exceed an average of 3.4% by weight. These results suggest that slag may behave the same as glass and indicate the importance of using a reducing atmosphere inside the float chamber. Further verification of these results should be attempted in a pilot-scale float chamber.

Visual observations of the slag/tin interface revealed no wetting of slag by tin, confirming another similarity between slag and glass.

2.8.1 Estimate Based on Concentration Profiles in Reference 51--

The concentration of tin oxide in 6 mm (1/4 in.) thick float glass varies from 2% by weight at the (lower) glass surface to zero at a depth of approximately 30 μm (0.03 mm). The concentration profile was linearized as shown schematically in Figure E-11.

Figure E-11. Schematic of approximate concentration profile used to estimate tin entrainment in slag.

The average concentration, $\bar{W}$, of tin oxide over the entire thickness of the glass is computed as follows:

$$\begin{aligned}\bar{W} &= \frac{1}{T} \int_0^T W(x) dx \\ &= \frac{1}{6} \times \frac{1}{2} bh = \frac{bh}{12} \\ \bar{W} &= \frac{0.03 \times 2}{12} = 0.005\%\end{aligned}$$

where T is the thickness of the glass sheet, and b and h are as indicated in Figure E-11.

$$\begin{aligned}\text{Wt. \% tin} &= \text{wt. \% tin oxide} \times \frac{119}{151} \\ &= 0.005 \times \frac{119}{151} \\ &= 3.94 \times 10^{-3} \%\end{aligned}$$

$$\begin{aligned}\text{Tin loss} &= 3.94 \times 10^{-5} \frac{\text{lb tin}}{\text{lb slag}} \times 2.7 \times 10^5 \frac{\text{lb slag}}{\text{hr}} \\ &= 10.64 \frac{\text{lb}}{\text{hr}}\end{aligned}$$

$$\text{Tin loss} \cong 43 \frac{\text{tons}}{\text{yr}} \text{ (for } 8,000 \frac{\text{hr}}{\text{yr}} \text{ operation)}$$

2.8.2 Estimate Based on Reference 36--

The average concentration of tin in float glass is given as 1% by weight over a penetration distance of 20 μm . The mass flow rate of slag in a 20- μm -thick sheet is given by:

$$w = 1.04 \frac{\text{ft}}{\text{s}} \times 20 \text{ ft width} \times \frac{20 \times 10^{-6}}{0.3048} \text{ ft depth} \times 180 \text{ lb/ft}^3$$

$$w = 0.2457 \text{ lb/s}$$

$$\text{Tin loss} = 0.01 \frac{\text{lb tin}}{\text{lb slag}} 0.2457 \frac{\text{lb slag}}{\text{s}} \times 3,600 \text{ s/hr}$$

$$= 8.84 \text{ lb/hr}$$

$$\text{Tin loss} \cong 35 \frac{\text{tons}}{\text{yr}} \text{ (for 8,000 hr/yr operation)}$$

3. FLOAT CHAMBER PROCESS DESIGN SUMMARY

The float chamber process recovers heat from molten slag to produce steam at 3.10 MPa (450 psia), superheated to 316°C (600°F). In the process, the slag is cooled from 1,482°C (2,700°F) to 427°C (800°F) and frozen into a solid sheet. Some 19.4 kg/s (154,000 lb/hr) of steam are produced, given a slag flow rate of 33.9 kg/s (269,000 lb/hr).

This steam generating process has two major components: the float chamber, which functions as a combined boiler and superheater, and the external steam and feedwater handling equipment. These two components are discussed separately in the pages that follow.

3.1 Float Chamber

The float chamber is an elongated rectangular structure, approximately 55.5 m (182 ft) long, 7.3 m (24 ft) wide, and 1.2 m (4 ft) high. A cross-sectional view of this equipment and a sketch of its structural support are shown in Figures E-12 and E-13. Sketches of the entrance and exit configurations are shown in Figure E-14.

In many respects, the chamber is similar to the float tanks used in the float glass industry. Molten slag enters the chamber from the surge vessel via an enclosed runner and flows under a series of two weirs: a gas-sealing weir and a flow-regulating weir.

Figure E-12. Float chamber cross section.

Figure E-13. Proposed float chamber support structure.

Figure E-14. Schematic of the float chamber entrance and exit.

The slag moves down the chamber in a 6.4 mm (1/4 in.) thick layer at a rate of approximately 0.3 m/s (1 ft/s), floating on a 5.1 cm (2 in.) deep pool of molten tin. Solidification occurs in the first 2.4-3.0 m (8-10 ft) of the chamber.

At the exit, a series of two carbon rollers remove the frozen slag ribbon from the bath. The first roller depresses the slag sheet below the molten metal surface to form a liquid seal, and the second lifts it out of the chamber. Leaving the chamber, the slag ribbon is supported by a series of asbestos-covered steel rollers. At least two of these rollers are powered to move the slag over the tin surface and out of the float chamber.

As the slag traverses the length of the chamber, its heat is transferred to overhead steam pipes by radiation and to submerged pipes via conduction through the molten metal layer. Altogether, there are 258 overhead, or radiation tubes, and 212 conduction tubes running widthwise across the chamber. To protect these steel pipes from the corrosive action of molten tin, they are encased in graphite blocks. Each block is 0.10 m (4 in.) by 0.25 m (10 in.) by 1.5 m (60 in.) with three holes machined in it to accommodate standard 1-1/2-in. pipes. Nearly 700 such blocks are required to encase all the pipes in the chamber.

The outer structure of the chamber consists of a 6.4-mm (1/4-in.) steel shell lined with a 0.3 m (1 ft) thick layer of castable refractory such as Kaocrete, manufactured by Babcock & Wilcox. The first 10 ft of the chamber are lined with Kaocrete 28LI, resistant to temperatures up to 1,538°C (2,800°F). The remainder is lined with Kaocrete B, which will withstand temperatures as high as 1,260°C (2,300°F).

The structure is supported by a series of lightweight steel beams, e.g., W8 x 24, spaced as shown in Figure E-13. The top of the vessel consists of thirty-one 1.8 m x 7.3 m (6 ft x

24 ft) sections, any or all of which can be removed when maintenance is required on the chamber interior. A bridge crane and hoist apparatus are provided for this purpose.

The space between the slag layer and the upper radiation collectors is filled with a reducing atmosphere -- 92% N₂, 4% H₂, and 4% CO -- to prevent the tin from oxidizing to tin oxide. Liquid seals are formed at both ends of the chamber to keep this atmosphere from leaking out and to prevent air from entering. In a final design it is important that the entire float chamber be airtight for two reasons:

- Tin oxide is more volatile than tin itself. Thus, relatively larger amounts of tin would vaporize in the system in the presence of oxygen. This vapor would rise to the top of the vessel, condense, and drip back down onto the slag surface. Hence, larger quantities of tin would be lost due to carryover with the slag leaving the chamber. This could significantly increase the makeup tin requirement, which would have a disastrous effect on the operating cost for the system (refer to Appendix F).
- If gas losses are small, the protective atmosphere can be supplied by compressed-gas cylinders at minimal cost. Otherwise, expensive generation equipment, i.e., a cryogenic air separation plant, might be needed to supply makeup nitrogen.

Regardless of the gas supply source, makeup atmosphere will be added to the chamber in the same manner as makeup tin -- through stainless steel tubes in the sides of the vessel, Figure E-15.

3.2 Steam and Feedwater Handling

A flow diagram and material balance for the steam and feedwater handling equipment are shown in Figure E-16 and Table E-12,

Figure E-15. Arrangement for tin and atmosphere makeup.

Figure E-16. Steam and feedwater handling system.

TABLE E-12. MATERIAL BALANCE FOR THE STEAM
AND FEEDWATER HANDLING SYSTEM

Stream No.	Stream name	Quantity of flow, kg/s (lb/hr)	
1	Makeup water	0.38	(3,000)
2	Condensate recycle	19.4	(154,000)
3	Feedwater	19.8	(157,000)
4	Circulating water	129	(1,027,000)
5	Steam-water mixture	129	(1,027,000) ^a
6	Saturated steam	19.4	(154,000) ^b
7	Deaerator steam	nil ^c	
8	Superheated steam	19.4	(154,000) ^d
9	Blowdown	0.38	(3,000)

^a 15% vapor, 85% liquid saturated @ 3.10 MPa (450 psia).

^b 3.10 MPa (450 psia).

^c <0.06 kg/s (500 lb/hr).

^d 3.10 MPa, 316°C (450 psia, 600°F).

respectively. In this system, the float chamber acts as a steam generator with integrated boiler and superheater sections. Of the 470 tubes crossing the bath, 204 may be considered boiling tubes. These are concentrated toward the "hot" end of the chamber in both the radiation and conduction banks. The remaining 266 tubes comprise the superheating section, concentrated at the "cold" end of the chamber. Both the boiling and superheating sections are designed for two-pass operation. Thus, there are 102 "loops" in the boiling section and 133 loops in the superheating section, with 2 tubes comprising a single loop.

Altogether, 129 kg/s (1,027,000 lb/hr) of water are circulated through the boiler tubes. Some 15% of this flow, 19.4 kg/s (154,000 lb/hr), is vaporized per cycle to form a saturated mixture at 3.10 MPa (450 psia). The flow from the boiler tube outlets is collected in a central header (see Figure E-17 for

Figure E-17. End view of header-tube configuration.

approximate configuration) and charged to the steam drum. Here, steam is separated from the mixture and distributed by an outlet header to the superheating tubes. This 19.4 kg/s (154,000 lb/hr) of steam is superheated from the saturation temperature of 236°C (456°F) to 316°C (600°F), collected in a third header, and exported from the system. The remaining 110 kg/s (873,000 lb/hr) of circulating flow is combined with the feedwater in the drum and recirculated to the boiling section through another distribution header.

Feedwater added to the drum includes a 100% condensate recycle stream and a small quantity of makeup water. The makeup water flow is designed to match the blowdown requirement, which is minimal for a system of this nature. For the purposes of this study, blowdown is taken as 2% of the steam export rate, or approximately 0.38 kg/s (3,000 lb/hr).

To maintain water quality in the system, both the makeup water and condensate recycle streams are treated. Makeup water coming into the system is filtered, softened, and deaerated before it is charged to the condensate tank. Because this stream is so small, existing softeners at the plant site should be sufficient to handle the increased flow. Likewise, steam requirements for the

deaerator are negligible [<0.06 kg/s (500 lb/hr)] in comparison to the total quantity of steam produced.

Additional water treatment chemicals are added to the condensate tank and to the drum. These include a condensate corrosion inhibitor, a chelating agent, an antifoam agent, and an oxygen scavenger.

Auxiliary facilities for the system include a startup boiler and a bridge crane (as previously mentioned). The startup boiler is required to melt the initial tin load and to bring the system up to operating temperature. A gas-fired package unit which outputs 2.5 kg/s (20,000 lb/hr) of steam at 6.9 MPa (1,000 psi) is selected for this application. With this unit, startup can be effected in approximately 1 day.

ABBREVIATIONS FOR APPENDIX E

A	-- constant defined by Equation E-16, m^{-1} (ft^{-1})
A	-- heat-transfer surface area, m^2 (ft^2)
$\hat{A}$	-- constant defined by Equation E-25, m^{-2} (ft^{-2})
A_c	-- area required for conductive heat transfer, m^2 (ft^2)
$A_{c, \text{ graphite}}$	-- surface area of graphite blocks required for conductive heat transfer, m^2 (ft^2)
A_ℓ	-- float chamber area required to remove slag latent heat of crystallization, m^2 (ft^2)
A_r	-- area required for radiative heat transfer, m^2 (ft^2)
$A_{r, \text{ graphite}}$	-- surface area of graphite blocks required for radiative heat transfer, m^2 (ft^2)
A_{slag}	-- surface area of slag sheet, m^2 (ft^2)
B	-- constant defined by Equation E-17, $m^{-1} - K^{-3}$ ($ft^{-1} - ^\circ R^{-3}$)
$\hat{B}$	-- constant defined by Equation E-26, $K^{-3} - m^{-2}$ ($^\circ R^{-3} - ft^{-2}$)
B_1	-- constant defined by Equation E-36, m^{-1} (ft^{-1})
B_2	-- constant defined by Equation E-37, $K^{-1} - m^{-1}$ ($^\circ R^{-1} - ft^{-1}$)
B_3	-- constant defined by Equation E-38, $K^{-3} - m^{-1}$ ($^\circ R^{-3} - ft^{-1}$)
B_4	-- constant defined by Equation E-39, K/m ($^\circ R/ft$)
C	-- constant defined by Equation E-18, K/m ($^\circ R/ft$)
$\hat{C}$	-- constant defined by Equation E-27, K/m^2 ($^\circ R/ft^2$)
C_s	-- heat capacity of slag, $J/kg \cdot K$ ($Btu/lb - ^\circ F$)
C_1, C_2	-- constants of integration, $K(^\circ R)$ and $K/m(^\circ R/ft)$
$\hat{C}_1, \hat{C}_2$	-- constants of integration
D	-- depth of tin layer, m (ft)
$\hat{d}$	-- inside pipe diameter, m (in.)
$\frac{dT}{dx}$	-- average temperature gradient in x-direction, K/m ($^\circ R/ft$)
F_ϵ	-- emissivity factor, dimensionless

$\bar{F}_{21}$	-- geometric view factor for radiation between two bodies in an enclosure with nonconducting re-radiating walls, dimensionless
f	-- friction factor, dimensionless
G	-- mass flux, $\text{kg/m}^2 \cdot \text{s}$ ($\text{lb/hr} - \text{ft}^2$)
Gr	-- Grashof number, dimensionless
g	-- gravitational constant, m/s^2 (ft/hr^2)
H	-- specific enthalpy, J/kg (Btu/lb)
ΔH	-- enthalpy difference, J/kg (Btu/lb)
ΔH_f	-- enthalpy change of slag due to fusion J/kg (Btu/lb)
ΔH_{tot}	-- total enthalpy change in going from boiler feed water to superheated steam, J/kg (Btu/lb)
k	-- thermal conductivity of air, $\text{W/m} \cdot \text{K}$ ($\text{Btu/hr-ft} - ^\circ\text{F}$)
k_e	-- "effective" thermal conductivity in empirical relationship for free convection in enclosed air space, $\text{W/m} \cdot \text{K}$ ($\text{Btu/hr-ft} - ^\circ\text{F}$)
k_{gr}	-- thermal conductivity of graphite, $\text{W/m} \cdot \text{K}$ ($\text{Btu/hr-ft} - ^\circ\text{F}$)
k_s	-- thermal conductivity of slag, $\text{W/m} \cdot \text{K}$ ($\text{Btu/hr-ft} - ^\circ\text{F}$)
k_{st}	-- thermal conductivity of steel, $\text{W/m} \cdot \text{K}$ ($\text{Btu/hr-ft} - ^\circ\text{F}$)
k_t	-- thermal conductivity of tin, $\text{W/m} \cdot \text{K}$ ($\text{Btu/hr-ft} - ^\circ\text{F}$)
L	-- length of float chamber, m (ft)
L	-- length of pipe, m (ft)
L_ℓ	-- length of float chamber required to remove slag latent heat of crystallization, m (ft)
L_T	-- total length of float chamber required to remove both latent and sensible heat, m (ft)
m	-- variable in characteristic Equation E-57
m_1, m_2	-- roots of characteristic Equation E-57
N_1	-- $\frac{\rho_s c_s u D}{k_s}$, dimensionless parameter in linear model
N_2	-- $\frac{k_t D}{k_s \delta}$, dimensionless parameter in linear model

Q	-- heat-transfer rate, W(Btu/hr)
Q_c, Q_r	-- cumulative heat transfer rates by conduction and radiation, respectively, over the region from entrance to float chamber to a distance x downstream from the entrance, W(Btu/hr)
Q_{ct}	-- total rate of heat transfer by conduction, W(Btu/hr)
Q_l	-- rate of evolution of slag latent heat of crystallization, W(Btu/hr)
Q_{rt}	-- total rate of heat transfer by radiation, W(Btu/hr)
q_c	-- conductive heat flux through tin, W/m^2 (Btu/hr-ft ²)
q_{cv}	-- free convective heat flux from upper surface of slag sheet, W/m^2 (Btu/hr-ft ²)
q_r	-- radiative heat flux from slag to overhead boiler tubes, W/m^2 (Btu/hr-ft ²)
q_x	-- conductive heat flux in flow direction, W/m^2 (Btu/hr-ft ²)
R	-- combined resistance to heat transfer of slag and tin, $m^2 \cdot K/W$ (hr-ft ² - °R/Btu)
R	-- combined resistance to heat transfer of pipe wall and graphite, K/W (hr - °F/Btu)
Re	-- Reynolds number
R_{slag}	-- resistance to heat transfer of slag, $m^2 \cdot K/W$ (hr-ft ² - °R/Btu)
R_{tin}	-- resistance to heat transfer of tin, $m^2 \cdot K/W$ (hr-ft ² - °R/Btu)
r_0	-- inside radius of pipe, m (ft)
r_1	-- outside radius of pipe, m (ft)
r_2	-- radius of graphite rod, m (ft)
S_g	-- surface tension of glass melt, N/m
S_{gt}	-- tin-glass interfacial tension, N/m
S_t	-- surface tension of tin, N/m
T_o	-- temperature at bottom of tin layer, °C (°F)
T_{ref}	-- arbitrary reference temperature, °C (°F)
T_s	-- temperature of slag, °C (°F)
T_t	-- temperature of tin, K(°R)

T_w	-- overhead boiler tube-wall temperature, K(°R) (overbar on T_w denotes average over width of float chamber)
T_1	-- slag temperature at entrance to float chamber, K(°R)
T_2	-- slag temperature at exit from float chamber, K(°R)
ΔT	-- temperature difference between two planes (see Equation E-81), K(°R)
ΔT	-- overall temperature difference, K, °C (°F)
ΔT_m	-- mean temperature difference, K, °C (°F)
U	-- overall heat-transfer coefficient, W/m ² ·K (Btu/hr-ft ² - °F)
u	-- velocity of slag sheet on tin bath, m/s (ft/s)
$\dot{V}$	-- volumetric flow rate of slag onto float chamber, m ³ /s (ft ³ /hr)
v	-- specific volume, m ³ /kg (ft ³ /lb)
v_{wtd}	-- weighted specific volume of steam-water mixture, m ³ /kg (ft ³ /lb)
W	-- width of slag sheet, m (ft)
w	-- mass flow rate, kg/s (lb/hr)
x	-- distance in flow direction measured from entrance to float chamber, m (ft)
Δx	-- length of differential volume element, m (ft)
y	-- distance in vertical direction measured from bottom of tin layer, m (ft)
z	-- distance in horizontal direction, m (ft)
α	-- coefficient in quadratic temperature profile, K(°R)
β	-- coefficient in quadratic temperature profile, dimensionless
$\hat{\beta}$	-- volume coefficient of thermal expansion, K ⁻¹ (°R ⁻¹)
γ	-- coefficient in quadratic temperature profile, K ⁻¹ (°R ⁻¹)
δ	-- thickness of slag sheet, m (ft)
$\hat{\delta}$	-- distance between two planes, m (ft)
∇^2	-- $\frac{\delta^2}{\delta x^2} + \frac{\delta^2}{\delta y^2} + \frac{\delta^2}{\delta z^2}$ = Laplacian operator, m ⁻² (ft ⁻²)
$\vec{\nabla}$	-- $\hat{i} \frac{\delta}{\delta x} + \hat{j} \frac{\delta}{\delta y} + \hat{k} \frac{\delta}{\delta z}$ = "del" operator, m ⁻¹ (ft ⁻¹)

ϵ -- roughness of pipe surface, m (ft)
 η -- x/D , dimensionless distance in flow direction
 η_2 -- L/D , dimensionless length of float chamber
 θ -- $\frac{T_s - T_o}{T_1 - T_o}$, dimensionless temperature
 θ_2 -- $\frac{T_2 - T_o}{T_1 - T_o}$, dimensionless temperature
 μ -- viscosity, Pa·s (lb/ft - hr)
 ν -- kinematic viscosity of air, m²/s (ft²/hr)
 ρ_g -- density of glass, kg/m³ (lb/ft³)
 ρ_s -- density of slag, kg/m³ (lb/ft³)
 ρ_t -- density of tin, kg/m³ (lb/ft³)
 σ -- Stefan-Boltzmann constant =
 $5.670 \times 10^{-8} \text{ W/m}^2 \cdot \text{K}^4$ (0.1712×10^{-8}
 Btu/hr-ft² - °R⁴)
 T -- equilibrium thickness of slag sheet, m (ft)

APPENDIX F

SLAG WASTE HEAT RECOVERY - FLOAT CHAMBER
Economics

CONTENTS

	<u>Page</u>
1. Capital Cost Summary	147
2. Operating Costs	150
3. Capital Cost Derivations	153

This appendix presents capital and operating costs for the float chamber described in Appendix E. These costs are summarized and discussed in Sections 1 and 2. Section 3 presents detailed cost derivations for the various equipment items, including material requirements and unit cost data where applicable.

1. CAPITAL COST SUMMARY

Capital requirements for the float chamber are summarized below. All costs include installation and are representative of December 1977 prices.

Float chamber:	
Structure and foundations	\$ 110,000
Refractory lining	660,000
Graphite pipe castings	170,000
Tin (initial requirement)	1,560,000
Steam feedwater handling:	
Steam drum	300,000
Deaerator	13,000
Condensate storage tank	14,000
Makeup water pump	1,000
Feedwater pumps	22,000
Circulating pumps	14,000
Chemical feed pumps and mix tanks	19,000
Piping	130,000
Auxiliaries:	
Startup boiler	240,000
Crane	38,000
Instrumentation, controls, and electrical	230,000
Site preparation	7,000
Total direct cost	3,531,000
Engineering, supervision, and construction	390,000
Total direct and indirect costs	3,921,000
Contractor's fee	120,000
Contingency	470,000
Fixed capital investment	4,511,000
Startup	230,000
Working capital	160,000
Total capital investment	<u>\$4,901,000</u>

Costs for the float chamber components are based on the design described in Appendix E. Because they do not reflect an optimum chamber design and operation, the costs are considered preliminary in nature. This optimization is an important factor since the float chamber accounts for 70% of the total direct cost. Chamber size, refractory thickness, and tin depth are particularly important variables because they have a direct bearing on the two largest cost items: tin and refractory. A considerable capital savings would accrue if these material requirements could be reduced without significantly depleting the energy recovery potential of the chamber.

Required tin purity is also an important cost factor. In the glass industry, product quality depends on the purity of tin in the tin pool. By analogy, the present design assumes that primary tin stock will be used. The cost for this material is approaching \$13/kg (\$6.00/lb) in major U.S. markets. However, the requirements for handling slag should be much less stringent. Thus, it may be feasible to replace some of the primary tin stock in the pool with scrap. Scrap is available at approximately one-third of the price for virgin metal.

It may also be possible to substitute large quantities of copper or other metals for tin. One study⁵² suggests that a 60 mol % tin - 40 mol % copper (78 vol % tin - 22 vol % copper) alloy may be suitable for glass industry applications. Copper is available at approximately one-tenth of the price for tin. Thus, an initial savings of \$170,000 or 11% of the metal cost could be realized by adopting this approach. Heat transfer may also be improved since copper is a much better conductor than tin.

Unit costs for the float chamber materials of construction and the steam drum are based on direct industry estimates. Costs for the steam and feedwater handling equipment components are based on

historical data, scaled to current price levels. Estimates for the startup boiler, crane, and site preparation are based on industry estimates.

The remaining fixed capital investment components, exclusive of the cost for tin, were estimated as percentages of the direct costs²⁶ (see below). Tin was considered an "inventory" item rather than part of the equipment, *per se*. Tin is included in the fixed capital investment only because it cannot be recovered at its original value due to contamination from contact with the slag and the great expense involved in removing it from the bath after a shutdown.

Instrumentation, controls, and electrical--13% of the installed equipment cost.

Engineering, supervision, and construction--20% of the direct costs.

Contractor's fee--5% of the direct and indirect costs.

Contingency--20% of the direct and indirect costs.

A large contingency factor is included due to the preliminary nature of the float chamber design. This charge should cover the costs for internal and external float chamber auxiliaries (weirs, rollers, drives, etc.), freight, and other items which could not be directly estimated.

Startup costs and working capital are estimated by the techniques recommended in Reference 25. Factors which add to the startup expenses are:

- (1) Operating labor hired 2 months early with an associated overhead of 25%.

- (2) Two extra operators provided for each shift for the first 3 months.
- (3) One engineer required for 6 months after startup.
- (4) Initial low operating efficiencies at a penalty of 10% of the variable costs for 6 months.
- (5) Ten percent contingency on items 1 through 4.
- (6) Excessive maintenance and minor design alterations during the initial operating period equivalent to 4% of the fixed capital investment (exclusive of the cost for tin).

Working capital is defined as accounts receivable plus inventories less accounts payable. Applicable costs for these categories are summarized below:

Accounts Receivable: 40-day production cost
 Inventories: 2-week supply of raw materials
 3-month supply of maintenance parts
 Accounts Payable: 2-week operating and maintenance labor cost
 1-month raw materials and utilities cost

2. OPERATING COSTS

Annual operating costs for the float chamber follow:

Makeup tin	\$ 466,000
Protective atmosphere	28,000
Treated makeup water	1,000
Labor	122,000
Electric power	18,000
Startup fuel	4,000
Maintenance	180,000

Total direct costs	819,000
--------------------	---------

Depreciation	450,000
Taxes and insurance	90,000
Total fixed costs	540,000
Overhead	106,000
Total operating costs	\$1,465,000

Tin losses due to carryover with the slag leaving the chamber are estimated at 36,000 kg/yr (40 tons/yr) (see Appendix E), based on the carryover rate experienced by the float glass industry. This carryover is an important consideration since makeup tin costs are approximately \$466,000 per year,^a or 51% of the direct operating cost.

The rate at which protective atmosphere escapes from the float chamber is another important assumption. Even though the float chamber is designed with liquid seals at both the entrance and exit, some losses are still anticipated. For the purposes of this evaluation, a 1% per hour escape rate is assumed. At this relatively low rate, cylinders of compressed nitrogen, hydrogen, and carbon monoxide can be used to supply the protective atmosphere makeup at a cost of \$28,000.^b Precautions must be taken, however, to preclude significantly higher loss rates which could require a cryogenic air separation plant for nitrogen generation. This would adversely affect the energy efficiency and capital requirements for the system as well as the operating cost.

Cost of softened water is assumed to be \$9.54/1,000 m³ (\$0.27/1,000 ft³). Feedwater treatment additives (i.e., corrosion inhibitors, chelating agent, antifoam agents, and oxygen scavengers) add another \$85/1,000 m³ (\$2.41/1,000 ft³) to the cost. Thus, the total unit cost is \$95/1,000 m³ (\$2.68/1,000 ft³), or \$0.095/kg

^aAt \$12.9/kg (\$5.83/lb) for tin⁵³ as of December 1977.

^b\$350 for the initial requirement.⁵⁴

(\$0.043/1,000 lb). However, the makeup water requirement is only 0.38 kg/s (3,000 lb/hr), so annual costs for water treatment are virtually nil.

Since the float chamber is highly automated, only two operators will be required per shift. One operator monitors the operation; the other adds water treatment chemicals to the mix tanks, adds makeup tin to the pool, changes gas cylinders and performs other necessary functions. At \$7.00/hr, the total annual cost for labor is \$122,000. Supervision for the operation is provided for in the slag handling and transportation system.

The power requirement for the process is 124 kW (166 hp). The circulating and feedwater pumps account for nearly all of this consumption. Based on drag force calculations, virtually no power is required to move the slag ribbon over the tin surface. Thus, the annual power cost is \$18,000 assuming 8,000 hours of operation per year and a unit cost of 1.8¢/kWh.

Small quantities of natural gas are required to operate the startup boiler. The annual amount allotted for this item is \$4,000.

Maintenance costs are estimated at 6% of the fixed capital investment,²⁶ exclusive of the initial cost for tin. A 50-50 split between material and labor charges is assumed.

Fixed charges to capital are based on 10-year, straight-line depreciation with no salvage values assumed. Actually, the tin may be salable as scrap. However, no net credit is taken due to the expense which would be incurred to recover it after shutdown.

Annual costs for taxes and insurance are estimated at 2% of the fixed capital investment.

Overhead charges are assumed equivalent to 50% of the cost for operating and maintenance labor. This allows for fringe benefits and a small increase in overall plant overhead costs.

3. CAPITAL COST DERIVATIONS

Costs for the various equipment items are derived as follows:

	<u>Material Requirement</u>	<u>Installed Unit Cost</u>	<u>Total Cost</u>
<u>STRUCTURE AND FOUNDATIONS</u>			
W8 x 24 steel beams	5,200 ft; 124,800 lb	\$0.475/lb ⁵⁵	\$ 59,280
1/4 in. steel plate	10,384 ft ² ; 105,570 lb	\$0.475/lb ⁵⁵	50,146
Total	230,370 lb		109,426
Concrete	81 yd ³	\$50/yd ³ (55)	<u>4,044</u>
Total			\$ 113,470
			\$ 110,000
<u>REFRACTORY LINING</u>			
Kaocrete 28LI	643 ft ³ ; 40.5 tons	\$210/ton ⁵⁶	\$ 8,505
Kaocrete B	10,188 ft ³ ; 641.8 tons	\$175/ton ⁵⁶	112,315
Installation	10,831 ft ³	\$50/ft ³ (57)	<u>541,550</u>
Total			\$ 662,370
			\$ 660,000
<u>GRAPHITE PIPE CASINGS</u>			
Graphite blocks	694 blocks; 964 ft ³ ; 102,180 lb	\$0.83/lb ⁵⁸	\$ 84,810
Machining		\$50/block ⁵⁹	34,700
Tools			600 ¹¹
Installation		\$50/ft ³	<u>48,200</u>
			\$ 168,310
			\$ 170,000
<u>TIN (INITIAL REQUIREMENT)</u>			
Tin	134 tons	\$5.83/lb ⁵³	\$1,562,440
			\$1,560,000

STEAM DRUM

Purchased cost	\$260,000 ⁶⁰
Insulation/installation	<u>40,000</u>
Total	\$300,000

DEAERATOR

Purchased cost ^a	\$ 10,000
Installation @ 30%	<u>3,000</u>
Total	\$ 13,000

CONDENSATE STORAGE TANK

The condensate storage tank is sized to handle 62.5% of the hourly feedwater flow, or 12,200 gal.⁶¹ Carbon steel construction is assumed.

Purchased cost	\$ 10,700 ²⁶
Installation @ 30%	<u>3,200</u>
Total	\$ 13,900
	\$ 14,000

MAKEUP WATER PUMP

Pump cost	\$ 740 ²⁶
Motor	180 ²⁶
Installation @ 30%	<u>280</u>
Total	\$ 1,200
	\$ 1,000

NOTE: Power requirement is minimal.

FEEDWATER PUMPS (2 REQUIRED)

Pump	\$ 5,600 ^a
Seal	400 ^a
Motor	2,600 ^a
Installation @ 30%	<u>2,600^a</u>
Total	\$11,200 x 2 = \$22,400
	\$22,000

CIRCULATING PUMPS (2 REQUIRED)

Pump	\$ 4,500 ²⁵
Motor	900 ²⁶
Installation @ 30%	<u>1,600</u>
Total	\$ 7,000 x 2 = \$14,000

^a An industry estimate.

CHEMICAL FEED PUMPS AND MIX TANKS²⁶

Mix tanks, agitators, and drives	\$ 4,500 ²⁶
Pumps	9,600 ²⁶
Motors	250 ²⁶
Installation @ 30%	<u>4,300</u>
Total	\$ 18,650
	\$ 19,000

PIPING^{a,b}

Boiler and superheater tubes	\$ 66,000
Headers	34,000
Other ^c	<u>30,000</u>
Total	\$130,000

STARTUP BOILER

Purchased cost	\$160,000 ⁶²
Installation	<u>80,000⁶²</u>
Total	\$240,000

CRANE

Purchased cost ^d	\$ 30,000 ⁶³
Installation	<u>8,000⁶³</u>
Total	\$ 38,000

SITE PREPARATION

Site preparation involves fine grading a 100 ft by 300 ft plot of land. Other site preparation is included in the capital estimate for the slag handling and transportation system (Appendix C); i.e., ramp construction. Overall site preparation costs are estimated at \$25,000; \$18,000 for ramp construction and \$7,000 for fine grading.²⁴

^a An industry estimate.

^b Cost includes pipe, fittings, insulation, hangers, and installation labor.

^c Includes an allowance for miscellaneous piping which cannot be estimated directly without a detailed drawing.

^d Includes bridge crane (10-ton capacity, 50-ft span), hoist, rail, and runway electrification.

APPENDIX G

SLAG WASTE HEAT RECOVERY - CONTACT TOWER

Technical Discussion

CONTENTS

	<u>Page</u>
1. Related Industrial Processes	158
1.1 Lead Shot Manufacture	159
1.2 Ammonium Nitrate Manufacture	159
1.3 Fiberglass and Mineral Wool Production	159
2. Contact Tower Design Considerations	160
2.1 Particle Generation Techniques	162
2.1.1 Extrusion Through an Orifice	162
2.1.2 Spinning Disk	163
2.2 Spinning Disk Design	166
2.2.1 Disk Size	166
2.2.2 Power Requirements	169
2.3 Heat Transfer in the Contact Tower	172
2.3.1 Radiation Heat Transfer	173
2.3.2 Forced Convection Heat Transfer	174
2.3.2.1 Forced Convection Heat-Transfer Coefficient, h_c	179
2.3.2.2 Characteristic Dimension of the Particle, D_p	181
2.3.2.3 Velocity of the Particle, V_T	184
2.3.2.4 Assumption of the Particle Velocity, V_T , and the Forced Convection Heat-Transfer Coeffi- cient, h_c	187
2.3.3 The Rate of Surface Area Formation, A_T	188
2.3.4 Particle Tower Residence Time, τ_F	189
2.3.5 Calculation of Forced Convection Heat Transfer	190
2.3.6 Center Temperature of Slag Particles	190
2.3.7 Validation of Negligible Radiation Heat Transfer	194

CONTENTS (continued)

	<u>Page</u>
2.4 Contact Tower Diameter, Case with Nonassisted Disk	195
2.4.1 Particle Trajectory Calculation Methodology	196
2.4.2 Trajectories for the Disk of Selected Design Parameters	199
2.4.3 Influence of Disk Operating Conditions on Particle Trajectories and Tower Diameter.	201
2.5 Contact Tower Height, Case with Nonassisted Disk	205
3. Contact Tower Design Summary.	208
Abbreviations for Appendix G.	212

Thermal energy from phosphorus furnace slag can be partially recovered by means of a contact tower.^a Generally, contact towers are used for one or more of the following functions: 1) a means of producing a specific product (e.g., prills and lead shot); 2) a means for containment of a two-phase (solid and gas or liquid and gas) flow; and 3) a means for intimate contact of the two phases.

Contact towers represent more an industrial art than a standard technology. In general, they are used only when other, usually cheaper and more standard, methods of achieving the same purpose have failed.

Background information concerning contact tower technology is presented in Section 1. This is followed by engineering evaluations and design calculations in Section 2, and a process design summary in Section 3; the latter two sections pertain to the application of the contact tower for waste heat recovery from phosphorus slag.

1. RELATED INDUSTRIAL PROCESSES

The processes discussed below show apparent resemblance to and indicate the technical feasibility of slag waste heat recovery via the contact tower. High melting point materials such as glass and slag are processed in towers of industrial significance. In the process, an effective heat transfer between the free-falling material and the surrounding air is created--a factor of considerable importance for slag waste heat recovery.

^aOther names (e.g., prilling tower and lead shot tower) are also used depending on specific tower application. Contact tower is exclusively used in this report except where specific applications are discussed.

1.1 Lead Shot Manufacture

One of the earliest known methods of making small spheres or prills was the use of lead shot towers.⁶⁴ Spherical lead shot was formed from liquid lead when it was allowed to solidify while falling down a shot tower. In 1849, a New York lead manufacturer proposed to employ a tower 15 m (50 ft) high in which "an ascending artificial current of air was contacting a descending current of metal, resulting in the formation of lead shot." For a period of time lead shot towers were used extensively in England, but by 1963 only four remained, indicating the obsolescence of this approach for lead shot manufacture.

1.2 Ammonium Nitrate Manufacture

Prilling towers have been used extensively since the mid-1940's for the production of ammonium nitrate fertilizers.⁶⁵ This application of prilling towers is well developed and documented.^{66,67}

Many commercial ammonium nitrate plants produce prills by spraying a concentrated molten solution from a multiplicity of small-diameter nozzles into a tall tower. Heights of over 30 m (100 ft) are not uncommon. As prills free-fall down the prilling tower countercurrent to the rising air, they cool and solidify.⁶⁸ Typical nozzles with a diameter of 0.8 mm (0.030 in.) produce prills with diameters in the 2.0 mm to 3.3 mm (0.080 in. to 0.129 in.) size range, depending upon the rate of liquid flow, operating pressures, and other operating variables.

1.3 Fiberglass and Mineral Wool Production

The fiberglass and mineral wool industry has extensive technological experience in handling molten mineral materials. Two basic concepts are used commercially: 1) small-diameter fibers are produced by forcing molten material through heated orifices, and

2) fibers are generated from a high speed rotating disk (Figure G-1).^{3, 69-73}

Commercial fiberglass plants use platinum nozzles to form and attenuate the molten glass fibers. A typical nozzle may have up to 400 small orifices through which the fibers are formed. As the glass melt is forced from the furnace through the nozzles (gravity or pressurized flow), the formed fibers are still too large. Further drawing and attenuation is achieved by applying pneumatic and/or thermal energies to the fibers.⁶⁹ Practically, this is done by steam, compressed air, or flame jets. Typical orifice diameters are between 1.0 mm and 2.5 mm (0.040 in. and 0.098 in.). As a result of attenuation, fibers reach micrometer size ranges ($1 \mu\text{m} = 10^{-6} \text{ m}$).^a

Industry uses the high speed rotating disk as the second method of manufacturing fibers from mineral melts.⁷⁴⁻⁷⁹ A large number of patents for the production of fibers or prills from molten glass, rock wool, or phosphorus slag via spinning disk have been issued.⁸⁰⁻⁸⁴ Since the hot melt is poured on a rotating disk through a relatively large opening, this method of fiber making is especially suitable for materials that are not too clean. As the melt reaches the disk it is thrown to the disk's periphery, subsequently disengaging and forming droplets or strings. By means of steam or compressed air jets immediately next to the disk periphery, the droplets and strings are formed into fibers. The fibers accumulate on the bottom of the tower and are removed for further treatment and processing.

2. CONTACT TOWER DESIGN CONSIDERATIONS

The most important factor in the design of contact towers is the selection of a particle (prill, fiber) generation technique. The

^aRefer to conversion tables in Appendix N for conversion of English to metric units.

Figure G-1. Schematic drawing of the insulating wool forming line in a commercial process.

particle generation technique defines the particle characteristics that are used in heat and mass transfer calculations, tower geometry determinations, and auxiliary equipment specification.

Three particle generation techniques are used by industry today. They are spraying, extrusion through orifices, and dispersion by a high speed rotating disk. Spraying is primarily used for pumpable materials that are at low temperatures. Since molten slag meets neither of these criteria, slag spraying is not considered technically feasible. Both remaining techniques are used commonly in high temperature applications and for materials that are very similar to phosphorus slag. Therefore, their further evaluation and discussion is warranted.

2.1 Particle Generation Techniques

2.1.1 Extrusion Through an Orifice--

In order to achieve high rates of heat transfer and corresponding small tower heights (see Section 2.3), small particles must be generated from the phosphorus slag. Using the orifice technique of particle generation, two alternative approaches are plausible: 1) extrusion of slag through small-diameter orifices, and 2) extrusion of slag through a large orifice with subsequent attenuation of the melted stream.

Producing particles via the first technique is dependent upon availability of clean molten material which does not contain suspended particles larger in size than the orifice itself. Molten phosphorus slag is not considered clean material. The slag, as it comes out of the furnace, may contain pieces of broken carbon electrodes and of unreacted and unmelted feedstock. Small-diameter nozzles would quickly become clogged with any of these impurities, causing prohibitive maintenance and operating

problems. It is principally for this reason that this approach to generating prills and fibers was not pursued further.^a

The latter technique, in which molten material is allowed to flow out of a relatively large-diameter orifice, requires supplemental energy. At some distance below the bottom of the orifice, the molten stream is fiberized by impinging on it a jet of air compressed to 791 kPa (100 psig). The jet of air directed perpendicularly to the flow of the melt has a shearing and fiberizing effect on the molten material, breaking it into small individual drops and/or fibers. Because of its apparent similarity to slag dispersion via spinning disk with pneumatic assist, the large orifice method was not considered further.^b

2.1.2 Spinning Disk--

A potential method of generating particles for the contact tower from molten phosphorus slag is via a spinning disk. A variation

^aReasonable tower heights (less than 30 m [100 ft]) require small slag particles. Production of these small particles with small-diameter orifices is considered technically prohibitive; no currently known techniques allow removal of suspended particles and lumps from molten phosphorus slag. One possible alternative is to design the surge vessel for good separation of heavy (sink) and light (float) materials, with a clean slag material being discharged from the vessel at a point of optimum separation. Disregarding the potential difficulties in keeping the discharged slag "debris-free" at all times, the feasibility of this approach has not been demonstrated to date. In order for this program to identify slag waste heat recovery alternatives with minimum technical risks, the concept of clean slag separation was not considered further. However, there were insufficient data to warrant its complete rejection.

^bLarge amounts of energy are needed for the attenuation of slag produced by large orifices. In a U.S. Bureau of Mines laboratory experiment, 25 kg (56 lb) of compressed air were needed to fiberize about a 6-kg (13-lb) batch of molten glass.⁷¹ Thus, the mass ratio of compressed air to the molten glass was approximately 4.3. Projecting these laboratory-obtained ratios to a facility handling 33.9 kg/s (269,000 lb/hr) of slag would imply that approximately 150 kg/s (1.2×10^6 lb/hr) of compressed air would be required. These approximated compressed air requirements are excessive and costly (see Section 3).

of this method utilizes compressed air to further fiberize the materials ejected from the spinning disk. The two variations are illustrated in Figure G-2.

In the approach illustrated in Figure G-2(a), the molten material is fed onto a rapidly spinning disk. As the molten material is deposited on the surface of the disk, it accelerates to the periphery of the disk, is distributed as a thin film on the surface of the disk, and is ejected from the disk in the form of discrete particles. The shape of these particles depends upon many properties of the material dispersed as well as the disk operating conditions.

To reduce the horizontal travel distance of the particles ejected from the disk, the spinning disk technique with pneumatic assist is employed, Figure G-2(b). Here, in addition to the spinning disk, a stream of compressed air is projected downward around the periphery of the spinning disk. This enhances molten material fiberization and deflects the particles in a downward direction. The net result is production of smaller diameter prills or fibers and reduced tower diameter. The penalty is in additional energy requirements for compressed air.

The two alternative methods of slag dispersion involving a high speed disk for the production of slag particles are further discussed below.^a

^aIn its energy requirements, the high speed disk with a pneumatic assist highly resembles the technique involving a large orifice. Consequently, the conclusions drawn concerning the spinning technique, which is evaluated further, and pertaining to energy requirements, should, to a limited degree, also be applicable to molten slag extrusion through a large orifice.

a. SPINNING DISK APPROACH

b. SPINNING DISK WITH PNEUMATIC ASSIST

Figure G-2. Methods of producing prills or fibers from molten slag utilizing spinning disk technique.

2.2 Spinning Disk Design^a

2.2.1 Disk Size--

One of the first steps in designing the contact tower is selection of the spinning disk design. Several assumptions were made governing the disk size. The spinning disk was also assumed to be flat and circular with no guide vanes, slots, or protrusions. This assumption eliminated consideration of disks of other, more complex geometries.

The literature reports very few data on spinning disks operating with molten materials such as phosphorus slag. Calculation of the disk diameter was simplified and based on apparently successful experimental work by the FMC Corporation.⁸⁰ This work resulted in two U.S. patents (3,360,592 and 3,360,593) which describe an application of a spinning disk for production of insulation-type fibers from molten phosphorus slag.^b

^aWhere desirable, refer to conversion tables in Appendix N for conversion of English to metric units.

^bIt should be noted, however, that while FMC's investigation concentrated on making a salable insulation product, this effort places major emphasis on energy recovery and little emphasis on the salable product. This difference in emphasis may have some influence on disk design and operating conditions. For example, the energy consumption in the salable product case is primarily dictated by the product qualities. If the energy costs represent a major operating expense, serious consideration may be given to minimizing energy consumption while maintaining product quality. However, even in the best case, energy consumption may be significant. With major emphasis on energy recovery, all operations, including the disk, are optimized so as to minimize energy consumption. This influences the design and operation of the spinning disk as well as the entire contact tower. Without some experimental demonstration, the significance of this influence could not be determined. Hence, the spinning disk design rests on available experimental data based on product making rather than energy recovery.

In FMC's experimental program, the following disk operating conditions were used:^a

$$n = 23 \text{ s}^{-1} \text{ (1,400 rpm)}$$

$$r = 0.178 \text{ m (0.583 ft)}$$

$$W_s = 6.8 \text{ kg/s (900 lb/min)}$$

$$\rho_s = 2,900 \text{ kg/m}^3 \text{ (182 lb/ft}^3\text{)}$$

where^b n = speed of rotation of the disk
 r = radius of the centrifugal disk
 W_s = mass flow rate of slag
 ρ_s = slag density

The disk tangential velocity, V , may be expressed as

$$V = 2\pi rn \quad (G-1)$$

and the velocity at which the slag leaves the disk, V' , as

$$V' = \frac{Q}{2\pi r t} = \frac{W_s}{2\pi r t \rho_s} \quad (G-2)$$

where Q = volumetric flow rate of molten slag
 t = apparent molten slag thickness at the periphery of the disk

^aThese conditions represent maximum slag loading of the spinning disk. Maximum slag loading implies small spinning disk diameter and resulting lower power consumption (see Section 2.2.2).

^bA complete list of abbreviations may be found at the end of this appendix.

Assuming that the slag is ejected at the disk tangential velocity,^a then

$$V = V' \quad (G-3)$$

and

$$2\pi rn = \frac{W_s}{2\pi r t \rho_s} \quad (G-4)$$

Solving for t , Equation G-4 becomes

$$t = \frac{W_s}{4\pi^2 r^2 n \rho_s} \quad (G-5)$$

Using FMC's experimental data, the film thickness, t , is determined from Equation G-5:

$$t = 80 \text{ } \mu\text{m} \text{ (0.000263 ft)} \quad (G-6)$$

Substituting this value and the slag density value into Equation G-5, a relationship between the slag feed rate, disk size, and disk speed is obtained:

$$r = \sqrt{\frac{W_s}{A n}} \quad (G-7)$$

where $A = 4\pi^2 t \rho_s$ (constant)

^aThis assumption is substantiated partially by the work of R. M. Downey⁸¹ who took high speed photographs of the molten slag ejected from the spinning disk and observed that slag is projected from the disk along straight lines which are tangents to the periphery of the spinning disk.

Equation G-7 was used to calculate disk sizes and disk rotational speeds for various fractions of the total slag feed rate throughout this study (see Figure G-3 and Section 2.4).

2.2.2 Power Requirements--

The net power required to drive the spinning disk was calculated according to the equation developed by S. J. Friedman et al.⁷⁴ This equation neglects windage and friction losses and is based on the derivation of power required to accelerate the liquid to the speed of the rotating disk:

$$P_n = 1.03 \times 10^{-8} W_s r^2 n^2 \quad (G-8)$$

where P_n = net power required for atomization

The net power requirements, P_n , calculated for one value of the slag feed rate of 33.9 kg/s ($W_s = 4,482$ lb/min) and at various disk diameters and rotational speed combinations are presented in Figure G-4. Generally, this figure indicates that disk power requirements increase with increased disk diameter. This effect is more pronounced at high rotational speeds.

A curve depicting disk diameters for constant film thickness as determined from Equation G-7 is also shown to facilitate disk size selection. Any disk on this curve satisfies requirements for slag film thickness equal to the one observed in the FMC study (Equation G-6).

Final selection of the disk size was arbitrary and was based on two practical observations: 1) electric motors for service at 29.2 s^{-1} (1,750 rpm) are readily available (changing motor speed in either direction would require a special motor design and/or speed control, both resulting in an increased complexity and cost of the spinning disk assembly), and 2) at a disk speed of 29.2 s^{-1} (1,750 rpm), the net disk power requirements are at the lower end

Figure G-3. Criteria for spinning disk size selection, constant slag thickness $t = 0.000263 \text{ ft}$ and slag density $\rho_s = 182 \text{ lb/ft}^3$.

Figure G-4. Spinning disk net power requirements.

of the investigated range (e.g., a motor operating at 4,000 rpm would require more than twice the power needed for a 1,750-rpm motor). Hence, the selected disk has the following specifications:

$$\text{disk diameter} = 0.70 \text{ m (2.3 ft)}$$

$$n = 29.2 \text{ s}^{-1} \text{ (1,750 rpm)}$$

$$P_n = 150 \text{ kW (200 hp)}$$

Operation characteristics of disks with specifications other than the above are discussed further in Section 2.4.

2.3 Heat Transfer in the Contact Tower

Results of this investigation have demonstrated that using the spinning disk for dispersion of slag requires a contact tower of such a large diameter that this concept is impractical (see Section 2.4). Downey⁸¹ in his investigation concluded that additional energy input is needed in order to deflect ejected particles from their primary horizontal direction. This additional energy can be supplied with compressed air.

A schematic of the spinning disk approach with pneumatic assist was illustrated in Figure G-2(b). Various versions of this basic concept are used extensively in the production of glass fibers.^{3,80-85} In general, this concept utilizes a ring of compressed air jets, directed downward, located around the periphery of the spinning disk. High velocity air jets form a curtain around the periphery of the disk. Material ejected from the disk encounters this air curtain and is deflected downward, and the contact tower diameter is no longer a function of disk tangential velocity and particle size, but rather a function of the downward force produced by the air jets.

Inherent in the pneumatically assisted disk approach is the assumption that the slag particles can be produced in a stream with a diameter equal to or slightly exceeding the diameter of the spinning disk. Considering the selected disk diameter of 0.70 m (27 in.), the contact tower diameter is no longer limiting to practical tower dimensions. It can be selected based on the desired flow and heat-transfer conditions in the tower. The critical dimension, the tower height, depends on the desired heat transfer. Hence, the following discussions attempt to quantify radiation and convective heat-transfer conditions in the contact tower using a pneumatically assisted disk for slag feeding.

2.3.1 Radiation Heat Transfer--

The magnitude of the radiation component depends upon the radiation areas, the temperature of the falling body, the temperature of the heat absorbing surfaces, emissivities of the surfaces, and a geometric view factor:⁸⁶

$$q_{\text{net}} = \sigma_s A_1 F_a F_b (T_A^4 - T_B^4) \quad (\text{G-9})$$

where q_{net} = rate of heat transfer by radiation

A_1 = area of the surface

σ_s = Stefan-Boltzmann constant

T_A, T_B = absolute temperatures of respective surfaces

F_a = dimensionless geometrical factor

F_b = dimensionless factor based on the emissivities of the two surfaces

For a single particle completely surrounded by a heat absorbing container, F_a may be taken as unity.

This implies that the falling particle "sees" thermal energy absorbing areas from all directions. However, in an actual tower the entire space is filled with fibers. All fibers are radiating

and reradiating thermal energy to each other, and only the fibers closest to the tower walls "see" the heat absorbing surface.

For towers in which only a small fraction of particles "see" the walls, the value of F_a approaches zero. Assuming a 5.5 m (18 ft) diameter tower and a uniform particle concentration, less than 21% of particles will be affected by radiation if the total blinding occurs at 30 cm (12 in.) away from the wall. This is reduced to less than 11% at 15 cm (6 in.) and 3.7% at 5 cm (2 in.) away from the wall. (See Section 2.3.7 for further verification of this assumption.) Consequently, radiation was assumed to play a minor role in contact tower heat transfer and was neglected in further calculations.

2.3.2 Forced Convection Heat Transfer--

The mechanism of forced heat convection in a contact tower with pneumatic assist is extremely complex. Upon leaving the spinning disk, molten slag particles encounter a blast of relatively cold compressed air and are shredded into fibers. In this region extreme air and particle turbulence occur. Once the fiberized particles are out of this highly turbulent region they are randomly positioned and dispersed in a more stabilized air stream.

Both the downward flow of air and the gravitational force are active in moving the fibers to the bottom of the tower. However, due to the random distribution of particle sizes and shapes, the effect of the two forces will depend on an effective particle size as well as particle orientation. Two degrees of freedom are involved in characterization of fibers, the fiber diameter and the fiber length. Due to relatively uncontrolled fiber production, the fiber length and diameter are likely to vary over a wide range of values, making the characterization of particles difficult. In addition, while the dimensional characteristics of formed fibers remain constant, the fiber orientation varies during the entire time the fibers flow to the tower bottom.

While this complex formation and flow of particles takes place, the heat is simultaneously transferred from the particles into the air. The particles are cooled and the air is heated. From a heat-transfer standpoint, two specific regions, characterized by different heat-transfer conditions, can be identified. First, the particles are cooled as they are suspended in the air stream and fall through the tower. This region is referred to as the suspended particle region. Second, additional particle cooling and air heating may be effected when particles are collected as a fixed bed on the tower bottom. This region is referred to as the fixed bed region.^a The two regions are illustrated in Figure G-5.

Figure G-5. Theoretical temperature distribution in a parallel flow heat exchanger.

^aThe contact tower design developed in this appendix does not incorporate a fixed bed region (see Section 3). Therefore, no heat-transfer calculation for this region is presented.

Under the complex conditions stated above, precise calculations of heat transfer are not possible, and some simplifying assumptions are needed. The following methodology to arrive at approximated tower heat transfer and size was applied.

Assuming no heat losses, the heat transfer between the falling slag particles and process air can be determined using the following set of equations:

$$q_n = h_c A_T \tau_F \Delta T_m \quad (G-10)$$

where q_n = rate of thermal energy transfer between slag and air
 h_c = forced convection heat-transfer coefficient
 A_T = the rate of surface area formation
 τ_F = time particles move through the suspended particle region

$$\Delta T_m = \frac{(T_1 - T_3) - (T_2 - T_4)}{\ln \frac{(T_1 - T_3)}{(T_2 - T_4)}} \quad (G-11)$$

where T_1, T_2, T_3, T_4 = temperatures (see Figure G-5)

From the thermal energy balance,

$$q_n = q_s = q_a \quad (G-12)$$

where

$$q_s = W_s C_s (T_1 - T_2) + W_s h_f \quad (G-13)$$

$$q_a = W_a C_a (T_4 - T_3) \quad (G-14)$$

where q_s = rate at which thermal energy is given up by slag
 q_a = rate of thermal energy gain by air
 W_s = slag feed rate
 W_a = air flow rate
 C_s = specific heat of slag
 C_a = specific heat of air
 h_f = heat of fusion of slag

Using Equations G-13 and G-14 and assuming $T_2 = T_4 + 100$ ($^{\circ}\text{F}$), the air temperature T_4 may be determined:

$$T_4 = \frac{W_s (C_s T_1 - 100 C_s + h_f) + W_a C_a T_3}{W_a C_a + W_s C_s} \quad (\text{G-15})$$

Assuming further $T_1 = 1,482^{\circ}\text{C}$ ($2,700^{\circ}\text{F}$) and $T_3 = 21^{\circ}\text{C}$ (70°F), the final slag and air temperatures (T_2 and T_4 , respectively) were calculated as a function of various air flow rates for a constant slag flow rate $W_s = 33.8$ kg/s ($4,482$ lb/min).

The efficiency of heat recovery can be calculated as

$$\eta = \frac{W_a C_a (T_4 - T_3)}{W_s C_s (T_1 - T_3) + W_s h_f} \times 100 \quad (\text{G-16})$$

where η = heat recovery efficiency

The results of the efficiency calculations are illustrated in Figure G-6. The illustrated results depict the range of exit air temperatures, T_4 , exit slag particle temperature, T_2 , and the corresponding tower efficiency as a function of air flow through the contact tower.

For increased efficiency, the tower needs increased air flow. However, this produces lower air exit temperatures. In order for

Figure G-6. Air exit temperature (T_4) and slag exit temperature (T_2) as a function of air flow rate.

the tower to operate according to the conditions illustrated in Figure G-6, the heat-transfer equation (G-10) must also be satisfied. This equation includes several unknown parameters, namely h_c , A_T , and τ_F . To determine these unknown parameters, the following approach was applied.

2.3.2.1 Forced Convection Heat-Transfer Coefficient, h_c --The forced convection heat-transfer coefficient from a falling particle can be calculated according to the empirical equation given by Knudsen and Katz:⁸⁷

$$Nu = 2.0 + 0.6(Pr)^{0.33} (Re)^{0.5} \quad (G-17)$$

$$Nu = \frac{h_c D_p}{k_a} \quad (G-18)$$

$$Pr = \frac{C_a \mu_a}{k_a} \quad (G-19)$$

$$Re = \frac{\rho_a V_T D_p}{\mu_a} \quad (G-20)$$

where Nu = Nusselt number
 Pr = Prandtl number
 Re = Reynolds number
 h_c = forced convection heat-transfer coefficient
 D_p = characteristic dimension of a particle
 k_a = thermal conductivity of air
 C_a = specific heat of air
 μ_a = viscosity of air
 ρ_a = density of air
 V_T = velocity of the particle

Six parameters influence the forced convection heat-transfer coefficient, h_c . Except for the characteristic particle dimension and velocity, parameters vary with temperature. Because the temperature of air in the contact tower varies (see Figure G-5), the forced convection heat-transfer coefficient will also vary. However, it can be shown that the temperature has a rather small influence on the forced convection heat-transfer coefficient. This is illustrated in Table G-1 where the h_c values are displayed for different particle velocities in the temperature range between 21°C and 982°C (70°F and 1,800°F). For a relatively wide range of temperatures, there is a relatively small change in the forced convection heat-transfer coefficient. This effect becomes less significant at higher particle velocities.

TABLE G-1. FORCED CONVECTION HEAT-TRANSFER COEFFICIENT, h_c , AS A FUNCTION OF PARTICLE VELOCITY AND TEMPERATURE

Velocity		Temperature		h_c^a	
m/s	ft/s	°C	°F	W/m ² ·K	Btu/hr-ft ² ·°F
1.5	5	21	70	153	27
		316	600	176	31
		538	1,000	192	24
		982	1,800	222	39
12.2	40	21	70	367	65
		316	600	381	67
		538	1,000	394	69
		982	1,800	422	74
30.5	100	21	70	559	98
		316	600	565	100
		538	1,000	576	101
		982	1,800	603	106
64.0	210	21	70	793	140
		316	600	790	139
		538	1,000	798	141
		982	1,800	823	145

^aCalculated for $D_p = 1.4$ mm (0.056 in.) (refer to Section 2.3.2.2).

The other two influencing parameters, the characteristic particle dimension and velocity, cannot be accurately determined. Some reasonable values need to be assumed for purposes of further calculations. This is discussed below.

2.3.2.2 Characteristic Dimension of the Particle, D_p --Characteristic particle dimension enters both the Nusselt number and the Reynolds number. An approximation of characteristic particle dimension can be obtained from calculated sizes of particles produced by a nonassisted disk which were investigated by Friedman et al.⁷⁴ who comprehensively studied the operating characteristic of many spinning disk designs.^a

Friedman's equations allow calculation of the maximum size of particles generated by a spinning disk operating under given conditions:

$$D_{\max} = 1.2r \left(\frac{r}{\rho_s n r^2} \right)^{0.6} \left(\frac{\mu_s}{r} \right)^{0.3} \left(\frac{\rho_s \delta L}{r^2} \right)^{0.1} \quad (G-21)$$

$$D_{\max} = 3D_{sv} \quad (G-22)$$

$$D_{sv} = \frac{\sum_i \eta_i d_i^3}{\sum_i \eta_i d_i^2} \quad (G-23)$$

^aThe disks studied are commonly used in spray dryers, humidifying equipment, and oil burners. The correlations that Friedman and his coworkers developed were experimentally verified using different materials. All these materials had considerably different properties from molten slag. However, because of the absence of more pertinent data, Friedman's equations were applied in this study. The validity of this assumption is not known and may be questioned. Consequently, the accuracies of size distributions for particles of molten slag produced by spinning disk techniques as well as any subsequent calculations using these size distributions are not absolute and their experimental verification is strongly encouraged and recommended.

where $D_{\max}$ = maximum particle diameter

n = disk rotational speed

r = disk radius

Γ = feed rate of liquid atomized based on wetted periphery

ρ_s = density of liquid feed (slag)

μ_s = viscosity of liquid feed (slag)

δ = surface tension of liquid feed (slag)

L = wetted periphery of disk

D_{sv} = surface per unit volume average particle diameter

d_i = diameter of particle

n_i = number of particles

For a plain centrifugal disk,

$$L = 2\pi r \quad (G-24)$$

and

$$W_s = L\Gamma = 2\pi r\Gamma \quad (G-25)$$

In order to solve Equation G-21, the viscosity and the surface tension of molten phosphorus slag as a function of temperature are needed. Experimentally determined viscosities of phosphorus slag at a SiO_2/CaO weight ratio of 1.01 and for temperatures between 1,302°C and 1,607°C (2,375°F and 2,925°F) are given in Appendix B. At an estimated slag pouring temperature of 1,482°C (2,700°F) in the contact tower, the slag viscosity, μ_s , is about 0.215 Pa·s (2.15 poise). The surface tension of molten slag at the same temperature of 1,482°C (2,700°F) was assumed to be equal to that of molten glass. Glass surface tension of 0.20 N/m (200 dynes/cm) at 1,482°C (2,700°F) was reported by E. B. Shand.²³

Substituting these values into Equations G-21 and G-22 produces the following results for the selected design case at 1,482°C (2,700°F):

$$D_{\max} = 4.2 \text{ mm (0.167 in.)}$$

$$D_{sv} = 1.4 \text{ mm (0.056 in.)}$$

Results indicate that the largest particle which will be produced at the design conditions is 4.2 mm (0.167 in.) in diameter.^a The average particle size based on the statistical surface-per-unit-volume distribution is 1.4 mm (0.056 in.) in diameter. From the definition of the surface-per-unit-volume distribution, D_{sv} signifies that 50% of the particles by number will be below the average size, D_{sv} , and 50% of the particles by number will be above the average size, D_{sv} .

In order to approximate particle size distribution over a broader range of particle sizes, another particle characterization parameter is needed. This parameter is provided by the Hatch dispersion coefficient defined by Friedman⁷⁴ as:

$$\alpha_h = 3.3(D_{sv})^{0.1} \quad (\text{G-26})$$

where α_h = Hatch dispersion coefficient defined as ratio of 50% size to 84.13% size, or of 15.87% size to 50% size taken to give a number greater than 1

Using $D_{sv} = 0.056 \text{ in.}$, the Hatch dispersion coefficient (α_h) is calculated to be 1.92. This coefficient implies that 15.87% of

^aAt design conditions, slag temperature of 1,482°C (2,700°F) was assumed. At the slag temperature of 1,371°C (2,500°F) the maximum particle size increases by about 25%. This indicates the importance of maintaining slag at high temperatures prior to its dispersion in the contact tower.

particles by number will have a diameter less than $D_{sv}/1.92$, or less than 0.74 mm (0.029 in.). On the other end of the particle size distribution, 84.13% of the particles by number will be below $1.92 D_{sv}$ in diameter, or below 2.7 mm (0.108 in.). Plotting the above values of particle sizes and corresponding percentages on logarithmic paper produces a straight line, Figure G-7.

From the above discussion, most of the slag particles will range in size from 0.74 mm (0.029 in.) to 2.7 mm (0.108 in.), with a maximum particle size of 4.2 mm (0.167 in.). Calculated values of the forced convection heat-transfer coefficient at 316°C (600°F) for these spherical particles are listed in Table G-2.

TABLE G-2. THE EFFECT OF PARTICLE SIZE ON THE FORCED CONVECTION HEAT-TRANSFER COEFFICIENT

Particle size D_p		h_c^a	
mm	in.	W/m ² ·K	Btu/hr-ft ² -°F
0.74	0.029	565	100
1.4	0.056	381	67
2.8	0.108	262	46
4.2	0.167	205	36

^a Calculated at 316°C (600°F), 12.2 m/s (40 ft/s) (see Section 2.3.2.4).

2.3.2.3 Velocity of the Particle, V_T --Particle velocity enters into the numerator of the Reynolds number. Consequently, the higher the velocity the higher the Reynolds number, which results in a higher forced convection heat-transfer coefficient. A rather pronounced influence of particle velocity on the value of h_c was indicated in Table G-1.

The particle velocity, V_T , in Equation G-20 can be expressed as a relative particle velocity by the following equation:

$$V_T = |\bar{V} - V_{air}| \quad (G-27)$$

Figure G-7. Calculated size distribution by number for phosphorus slag particles produced by a spinning disk.

where $\bar{V}$ = absolute particle velocity through the tower
 V_{air} = air velocity through the tower

Absolute velocity of particles throughout the contact tower will vary. As particles leave the spinning disk their initial velocity is high; it was calculated at 64.0 m/s (210 ft/s) (see Equation G-1). Then the particles are exposed to a stream of compressed air. In this region of high turbulence, the shape and direction of particles change. Because compressed air forces partially counteract the particle acceleration in the horizontal direction, the particle velocity will also momentarily decrease. Once they leave the turbulent region, particles remain suspended in the air and, due to gravitational forces, accelerate to reach the terminal velocity. In this very complicated environment, particles may experience absolute velocities ranging from 0 m/s to 64.0 m/s (0 ft/s to 210 ft/s).

Velocities of air through the contact tower can be approximated using the data in Figure G-6. In order to achieve reasonably high air temperatures and reasonable tower heat recovery efficiencies, the air flow rate through the tower will need to be between 45 kg/s and 136 kg/s (100 lb/s to 300 lb/s). These flow rates translate into volumetric flow rates of 37.8 m³/s to 113 m³/s (80,000 scfm to 240,00 scfm). With the tower diameter fixed at 5.5 m (18 ft) and known air flow rates, the air velocity through the tower can be calculated.^a

Air flow rates and velocities for a tower 5.5 m (18 ft) in diameter are as follows:

^aIt should be noted that the air velocities through the tower can be varied easily by selecting towers with diameters other than 5.5 m (18 ft).

Air flow rate			
Tower entrance		Tower exit	
m ³ /s	cfm	m ³ /s	cfm
37.8	80,000 ^a	145	307,000 ^b
113	240,000 ^a	278	589,000 ^c

Air velocity, V_{air}			
Tower entrance		Tower exit	
m/s	ft/s	m/s	ft/s
1.6	5	6.1	20
4.8	15	11.8	39

^a21°C (70°F).

^b854°C (1,570°F) (see Figure G-6).

^c449°C (840°F) (see Figure G-6).

Combining the absolute particle velocity range with expected values of air velocities for a 5.5 m (18 ft) diameter tower through Equation G-27, an appreciation for possible ranges of relative particle velocities, V_T , is obtained; minimum relative particle velocity is 0 m/s ($\bar{V} = V_{air}$) and maximum relative particle velocity is 58 m/s (190 ft/s) ($\bar{V} = 210$ ft/s and $V_{air} = 20$ ft/s). The range is too broad to allow a more precise assumption of particle velocities, V_T .

2.3.2.4 Assumption of the Particle Velocity, V_T , and the Forced Convection Heat-Transfer Coefficient, h_c --The findings above indicated the difficulties associated with the determination of particle velocities, V_T , in the contact tower. Particles in the tower may experience a wide range of velocities. In addition, the particles are fiberized, and their shape and characteristic dimension are changed. Due to their fibrous configuration, the particles will change their orientation during their presence in the tower and experience various velocities. It therefore became necessary to assume an average value for particle velocity.

A velocity of 12.2 m/s (40 ft/s) was selected. This value is close to the lower end of the possible particle velocity range and may be regarded as somewhat conservative. However, it is believed that while the particles in the tower are undoubtedly exposed to high velocities, this exposure is for a relatively short time. Particles are likely to move through the tower at lower than their maximum velocity for a larger fraction of their residence time in the tower. Hence, the selection of the average velocity at the lower end of the range.

Considering the most likely range of tower operating conditions ($W_a = 100$ lb/s to 300 lb/s; and tower entrance temperature $T_3 = 70^\circ\text{F}$), an average tower temperature may be assumed at about 316°C (600°F).

With a selected particle velocity, average particle size, and an anticipated average tower temperature, the value of the forced convection heat-transfer coefficient may be determined. This value is $381 \text{ W/m}^2\cdot\text{K}$ ($67 \text{ Btu/hr}\cdot\text{ft}^2\cdot^\circ\text{F}$),^a see Table G-1.

2.3.3 The Rate of Surface Area Formation, A_T

Size distributions for particles produced from a nonassisted disk were presented in Section 2.3.2.2. It was determined that the mean particle size, D_{sv} , for the particles produced at the selected disk operating conditions is 1.4 mm (0.056 in.). As the particles leave the disk, shear forces of highly compressed air fiberize them and change their shape, which changes their surface area. Since the size of fibers produced from a pneumatically assisted disk could not be predicted, characteristic fiber dimensions (diameter and length) for fibers produced from a spherical

^aConservativeness of this assumption is also related to the particle characteristic dimension. In the contact tower, the slag particles are fiberized and become long strings with a small diameter. Due to their large surface area (see Section 2.3.3), these fibers are expected to experience high convection heat transfer.

particle of known diameter were calculated. It was assumed that particles with a mean spherical diameter, D_{sv} , of 1.4 mm (0.056 in.) are stretched into fibers with lengths equal to nD_{sv} , where $n \geq 1$. Assuming further different values for n , the fiber diameters and lengths were determined. These are presented below for different n values for a spherical particle diameter, D_{sv} , of 1.4 mm (0.056 in.).

n	Fiber characteristic dimensions				$\frac{A_{\text{fiber}}}{A_{\text{sphere}}}$	A_T	
	Length		Diameter			m ² /s	ft ² /s
	mm	in.	mm	in.			
1	1.4	0.056	1.2	0.046	1.16	57	610
10	14	0.56	0.36	0.014	2.52	120	1,300
40	57	2.2	0.18	0.0072	5.15	250	2,700
100	142	5.6	0.12	0.0046	8.22	400	4,300
200	284	11	0.08	0.0032	11.4	560	6,000
400	569	22	0.06	0.0023	16.4	810	8,700
800	1,140	45	0.04	0.0016	22.9	1,100	12,000
2,000	2,840	110	0.03	0.0010	35.7	1,800	19,000
80,000	114,000	4,500	0.004	0.00016	229	11,200	121,000

Fibers produced by the fiberglass industry range in size from 0.004 mm to 0.020 mm.⁸⁵ These sizes suggest the technical feasibility of producing fibers with diameters ranging from 0.03 mm to 1.2 mm.

2.3.4 Particle Tower Residence Time, τ_F --

Average particle velocity through the tower, V_T , was assumed at 12.2 m/s (40 ft/s) (see Section 2.3.2.4). With this velocity, the particle tower residence time can be calculated as follows:

$$\tau_F = \frac{H}{V_T} \quad (G-28)$$

where H = tower height

Assuming a tower height of 9.1 m (30 ft), the residence time of particles in the tower becomes 0.75 s.

2.3.5 Calculation of Forced Convection Heat Transfer--

Using Equation G-13 or G-14, the heat-transfer rate $q_s = q_a = q_n$ can be calculated for the most likely tower operating conditions ($W_a = 100$ lb/s to 300 lb/s; 80,000 scfm to 240,000 scfm).

Under the assumption $T_2 = T_4 + 100$ (°F), the term ΔT_m is determined:

$$\Delta T_m = \frac{(2,700 - 70) - 100}{\ln \frac{2,700 - 70}{100}} = 774^\circ\text{F}$$

Except for A_T , all other parameters in Equation G-10 are now known or assumed, and a value of A_T that is needed to facilitate the required heat-transfer rate can be calculated. The results are summarized below:

Air flow		T_4		q_n		A_T	
m^3/s	scfm	°C	°F	MW	Btu/hr	m^2/s	ft^2/s
37.8	80,000	816	1,500	39.2	1.34×10^8	320	3,400
113	240,000	388	730	52.0	1.78×10^8	426	4,600

The calculated A_T values are well below the A_T values practiced by industry today (see Section 2.3.3) and should be easily achievable in a tower of the assumed dimensions (18 ft diameter, 30 ft height).

2.3.6 Center Temperature of Slag Particles--

Previous evaluations dealt with convection heat transfer from slag particles into the surrounding air. This convection heat transfer was shown to be feasible provided the forced convection heat-transfer coefficient of $381 \text{ W/m}^2 \cdot \text{K}$ ($67 \text{ Btu/hr-ft}^2 \cdot ^\circ\text{F}$) is achievable in the contact tower.

In reality, the heat is removed by force convection from the surface of each particle. This reduces the particle surface temperature and establishes a temperature gradient between the surface and the center of the particle. A temperature gradient thus created becomes a driving force for heat transfer from the interior to the surface of the particle.

The resistance to the conductive heat transfer could represent a significant limitation to the overall heat transfer inside the contact tower. Heisler⁸⁸ investigated the rate of heat transfer from bodies of different shapes into the surrounding medium. The results of this investigation, referred to as Heisler charts, Figure G-8, allow determination of temperature at the center of particles as a function of exposure time and ambient temperature.

Figure G-8. Chart for determining temperature history at center of infinitely long cylinder.

The following equations apply.

$$Y_O = \frac{T_a - T_O}{T_a - T_b} \quad (G-29)$$

$$m = \frac{k_s}{r_d h_c} \quad (G-30)$$

$$X = \frac{\alpha \tau}{r_d^2} \quad (G-31)$$

$$\alpha = \frac{k_s}{C_s \rho_s} \quad (G-32)$$

where Y_O = temperature ratio
 T_a = average tower air temperature
 T_b = initial temperature of the slag
 T_O = final center temperature of a particle
 m = relative boundary resistance
 k_s = thermal conductivity of slag
 r_d = radius of a sphere or a cylinder
 h_c = forced convection heat transfer coefficient
 X = dimensionless parameter
 α = thermal diffusivity of slag
 τ = time
 C_s = specific heat of slag
 ρ_s = density of slag

Figure G-8 was used to calculate temperatures at the center of cylindrical fibers of various diameters at various ambient temperatures and at the residence time of 0.75 s. Heat of fusion was neglected in these calculations. The results are summarized in Figure G-9.

Figure G-9. Center temperature of cylindrical particles as a function of particle radius and ambient temperature.

The results indicate that within the expected tower operating conditions (particle residence time of 0.75 s and average air temperature of about 600°F), fibrous particles with diameters of 0.86 mm (0.034 in.) or less will solidify (center temperature of 2,300°F) before reaching the tower bottom. This should aid in maintaining the integrity of discrete particles, prevent particle fusion, and facilitate heat transfer and particle removal from the tower.

Obtaining lower center temperatures in air at 316°C (600°F) requires correspondingly smaller fiber diameters. Particle center temperatures of 443°C (830°F) may be achieved with fibers 0.13 mm (0.005 in.) in diameter. Fibers of these diameters may need to be produced in order to satisfy the assumption $T_2 = T_4 + 100$ (°F) for a tower operating with 113 m³/s (240,000 scfm or 300 lb/s) air feed (see Figure G-6).^a

2.3.7 Validation of Negligible Radiation Heat Transfer--

Previous evaluations assessed heat-transfer conditions inside the contact tower while neglecting the radiation heat transfer (see Section 2.3.1). Also, these evaluations indicated the type of particles needed to facilitate forced convection heat transfer (see Section 2.3.5 and 2.3.6). This information can be used to assess the potential radiation heat transfer.

From data presented in Section 2.3.3, the number of particles generated per unit volume of air inside the contact tower can be calculated. Using the number of particles and their anticipated size and shape, and assuming uniform particle concentration, the area of particles projected on a plane normal to the tower wall may

^aIt should be noted that further cooling of particles may be achieved once particles are solidified and collected on the tower bottom in the form of a fixed bed. Cooling of slag particles in a fixed bed was not considered in this evaluation.

be obtained. This area gives an indication of how well the particles in a given volume shade each other and the tower walls, thus indicating the potential for heat transfer by radiation.

Fibers with a diameter of about 0.12 mm (0.0046 in.) or less are needed to facilitate forced convection heat transfer in a tower of practical dimensions (18 ft in diameter x 30 ft high) (see Section 2.3). Considering fiber particles of this size and smaller, less than 15% of the particles will be affected by radiation heat transfer as indicated below.

Particle diameter, cylindrical particle		Particle projected area, individual particle		Particle projected area per unit volume of 1 cm ³ , cm ²	Distance needed for the particle projected area to be equal to 1 cm ²		Particles affected by radiation heat transfer, %
mm	in.	mm ²	in. ²		cm	in. ^a	
1.2	0.046	1.6	0.003	0.004	231	91	98
0.36	0.014	5.1	0.008	0.010	97	38	58
0.12	0.0046	17	0.026	0.045	22	9	15
0.08	0.0032	23	0.036	0.064	16	6	11
0.06	0.0023	33	0.051	0.090	11	4	8
0.04	0.0016	47	0.072	0.127	8	3	6
0.03	0.0010	74	0.115	0.201	5	2	4

^a These distances are calculated for the tower air flow at 37.8 m³/s. For other flows use the following correction:

$$\text{Distance} \times \frac{\text{New flow (m}^3/\text{s)}}{37.8} = \text{New distance}$$

In addition, only a fraction of the heat carried by the affected particles can be expected to be transferred by radiation. This suggests that only a small fraction of heat transfer in the contact tower will take place by radiation, thus validating the assumption made in Section 2.3.1.

2.4 Contact Tower Diameter, Case with Nonassisted Disk

Engineering evaluations presented above suggested that prohibitively large contact tower diameters are required to contain slag

particles produced by the nonassisted spinning disk. Consequently, the nonassisted disk was eliminated from further consideration. This section presents data to support the above assumption.

Slag particles ejected from the spinning disk may travel a considerable distance in a horizontal direction. This might necessitate large-diameter towers in order to avoid particle collision with the walls.^a An indication of particle horizontal travel can be obtained by calculating particle trajectories.

2.4.1 Particle Trajectory Calculation Methodology--

Particle flow pattern, called a trajectory, through the tower is a complicated one. The slag, fed into the contact tower via a spinning disk, produces a stream of slag particles which are thrown tangentially from the disk toward the tower walls. This

^a Since it is possible for particles to collide with the wall and then fall down to the bottom of the tower, the premise of containing all particles within the tower walls may be questioned. Using the same methodology for calculating heat transfer as that described in Section 2.5, the temperature drop in particle centers during travel from the disk to the tower wall was calculated. Since the particles with the largest diameter, D_{max} , were found to travel the farthest, the calculation was based on particles with the maximum diameter. The calculation suggested an extremely small temperature drop. This means that the large particles may be reaching the tower wall with the bulk of their mass still molten. Formation of a solid crust on the surface of these particles is likely, but without an experiment it could not be ascertained whether the crust would be strong enough to withstand impact with the wall and prevent the buildup of slag on the tower walls. Hence, the above premise was adopted. Also, from the number particle size distribution, Figure G-7, a weight distribution can be approximated. This was done by converting the particle number in arbitrarily selected particle size increments into volume and weight (assuming a mean particle diameter for each increment and spherical particles). The fraction weight for each particle size range was then computed as a percent of the total weight of slag particles in all size ranges. Results of these calculations, presented in Figure G-10, indicate that a large fraction of the particle mass (58%) is concentrated in particles above 2.5 mm (0.1 in.). Therefore, if slag buildup on tower walls were to occur, it would involve a significant fraction of the total slag mass.

Figure G-10. Calculated size distribution by mass for phosphorus slag particles produced by a spinning disk.

stream is generated in a horizontal plane by the disk. Because of gravitational forces, the particles will not retain this horizontal direction but will start to decline. Eventually, due to the opposing drag forces, the horizontal component of the momentum decreases to zero; only the gravity force remains and causes the particles to fall through the tower.

Particles of different sizes will assume different trajectories. Thus, depending on size, some particles reach the vertical flow pattern while other particles are still moving toward the tower walls. The tower diameter is then determined from the trajectory for particles with the largest horizontal component.

Particle trajectories were calculated based on the work of C. E. Lapple et al.⁸⁹ This extensive work takes into account the effects of gravity, aerodynamic drag, air and particle density, air viscosity, and initial ejection velocity. Particles were assumed to be ejected into the air in a horizontal direction at a velocity equal to the tangential disk velocity (Equation G-2). The maximum horizontal distance was defined and calculated as the distance at which the vertical component of particle velocity is much greater than its horizontal component, or $V_v/V_h = 99.9$. This meant that at the calculated time, the particles were moving essentially in a vertical downward direction. The following equations apply:

$$\Delta V_h = - \left(\frac{3\rho_s C V_T V_h}{4\rho_s D} \right) \Delta \tau \quad (G-33)$$

$$\Delta V_v = \left[\frac{g(\rho_s - \rho_a)}{\rho_s} - \left(\frac{3\rho_a C V_T V_v}{4\rho_s D} \right) \right] \Delta \tau \quad (G-34)$$

$$V_T = \sqrt{V_h^2 + V_v^2} \quad (G-35)$$

$$Re = \frac{DV_T \rho_a}{\mu_a} \quad (G-36)$$

$$X_h = \sum_{\tau=0}^{\tau \text{ at } V_v \gg V_h} (V_i - \Delta V_h) \Delta \tau \quad (G-37)$$

where V_T = velocity of the particle
 V = initial ejection velocity (disk tangential velocity)
 V_h = horizontal component of particle velocity
 V_v = vertical (downward) component of particle velocity
 ρ_a = density of air
 C = aerodynamic drag coefficient for sphere (see Reference 89 for values)
 D = particle diameter
 τ = time
 Re = Reynolds number
 g = gravitational constant
 X_h = horizontal distance traveled by ejected particle
 μ_a = viscosity of air

2.4.2 Trajectories for the Disk of Selected Design Parameters--

Assuming the selected disk design parameters (Section 2.2.2), the particle ejection velocity, V , is calculated as follows:

$$V = 2\pi rn = 2\pi \times 1.15 \text{ ft} \times 1,750 \text{ rpm}$$

$$V = 64 \text{ m/s (210 ft/s)}$$

An iterative computer program was used to solve the above set of equations. Calculated results indicate the following maximum horizontal components of trajectories:

Particle size		Horizontal trajectory component, tower radius	
mm	in.	m	ft.
4.2 ^a	0.167	59	195
2.7 ^b	0.108	43	140
1.4 ^b	0.056	24	80

^aValues for $D_{\max}$ (Equation G-21).

^bValues for D_{sv} (Equation G-22).

Results clearly indicate that to contain the trajectory of a maximum size particle ($D_{\max}$) within the walls of a contact tower, a diameter of 118 m (390 ft) is required. To contain particles with a diameter smaller than 2.7 mm (0.108 in.), which represent only 57% of particles by weight (see Figure G-10), requires a tower 86 m (280 ft) across. This means that the trajectories of particles are too long to be contained within a contact tower of practical diameter.

It is evident from the above that the particle diameter has a very strong influence on the particle trajectory. Therefore, generation of smaller slag particles would result in smaller tower diameters. There are several approaches to generating particles with smaller effective diameter via spinning disk techniques:

1. Deflection of particle trajectories in a downward direction using supplemental energy. This approach involves the use of compressed air for particle deflection and has been selected for contact tower operation.

2. Modification of disk operating conditions (slag feeding rate, disk diameter, and rotational speed) to facilitate production of small particles.
3. Utilization of multiple disks. This approach is a modification of the approach above. However, it offers wider variations in disk operating conditions at lower overall disk power consumption.

Approaches 2 and 3 are discussed below.

2.4.3 Influence of Disk Operating Conditions on Particle Trajectories and Tower Diameter--

It is evident from the above discussion that the particles with the largest diameter, $D_{\max}$, result in the longest trajectory. Therefore, further examination focuses on particles with diameter $D_{\max}$ only.

The influence of disk operating conditions on the maximum particle diameter, $D_{\max}$, can be determined from Equation G-21. The results are illustrated in Figure G-11. This figure clearly illustrates the strong influence of disk rotational speed on maximum particle diameter under the condition of constant film thickness. Disregarding the penalty of higher power requirements to operate disks at increased rotational speeds,^a this reduction in the maximum

^aSubstantial reduction in disk power requirements may be realized by distributing slag on several smaller disks. This is illustrated in Figure G-12. Under the condition of constant film thickness, a single disk with a diameter of 0.46 m (1.50 ft) and operating at 66.7 s^{-1} (4,000 rpm) would require about 545 kW (430 hp). Distribution of the same slag feed over 12 disks operating at the same rotational speed and under the condition of constant film thickness would require 0.12 m (0.4 ft) diameter disks and about 30 kW ($12 \times 13.3 = 40$ hp). Practically, it is difficult to envision an operation of a single tower with multiple disks whether on single or multiple shafts due to problems associated with distribution of slag to the multiple disks. Operation of multiple towers each containing a small disk appears prohibitive for economic reasons since even with multiple disks the tower diameters remain large as indicated in Figure G-13.

Figure G-11. Maximum particle diameter for constant disk film thickness conditions.

Figure G-12. Net horsepower for liquid slag dispersion as a function of disk diameter and rpm.

Figure G-13. Maximum horizontal trajectory as a function of maximum particle size at various spinning disk rotational speeds and slag feed rates.

particle diameter, suggesting potential reduction in the contact tower diameter, appears attractive. However, because other parameters (namely, size and initial velocity of particles), a more rigorous analysis is warranted.

Particle trajectories produced by a spinning disk operating at various rotational speeds and slag feeding rates were determined using methodology described in the section above. Results, illustrated in Figure G-13, indicate that maximum particle trajectory does not change substantially as a result of a change in the disk rotational speed. While the maximum particle diameter, $D_{\max}$, may be reduced by a factor of about two when the disk rotational speed increases from 16.7 to 66.7 s^{-1} (1,000 to 4,000 rpm), the maximum particle trajectory decreases by only about 10% to 12.5%.

The reason for this small effect on particle trajectory is that the reduction in particle size at higher rotational speeds (assuming constant slag feeding rate and film thickness) is offset with the associated higher initial velocities of the ejected particles.

Even under the most favorable conditions considered (slag feed rate of 373.5 lb/min, 4,000 rpm), the maximum horizontal particle trajectory remains at about 26 m (85 ft). The contact tower diameter needed to contain these particles would be 52 m (170 ft), too large to be considered practical.

2.5 Contact Tower Height, Case With Nonassisted Disk

Due to surface tension forces, freely suspended liquid particles assume an equilibrium spherical shape. Therefore, it is reasonable to assume that the slag particles produced by a spinning disk with no pneumatic assist will be spherical.

Initially, the particles leave the disk in a horizontal direction at a high velocity. As the particles move through the air their velocity decreases. Simultaneously, due to gravity forces, the particles start to decline from their horizontal direction, eventually move straight down, and are collected at the bottom of the tower.

In order to facilitate heat transfer and removal of slag particles from the tower, it is desirable that the particles be solidified and remain intact before they reach the tower bottom. Using the Heisler chart for spherical particles, Figure G-14, and assuming a forced convection heat-transfer coefficient of $381 \text{ W/m}^2 \cdot \text{K}$ ($67 \text{ Btu/hr-ft}^2 \cdot ^\circ\text{F}$), particle initial temperature of $1,482^\circ\text{C}$ ($2,700^\circ\text{F}$), particle center temperature of $1,260^\circ\text{C}$ ($2,300^\circ\text{F}$) (slag solidification temperature is $2,300^\circ\text{F}$), and particle velocity of 12.2 m/s (40 ft/s), the falling distances necessary to completely solidify particles of different sizes were calculated. The results plotted for the various process air temperatures are summarized in Figure G-15.

The data indicate that a tower about 40 m (130 ft) high is necessary to solidify 4.2-mm (0.167-in.) spherical particles at a temperature of 316°C (600°F). This particle size is representative of maximum-diameter particles (D_{max}) produced at selected disk operating conditions (disk diameter 2.3 ft , $1,750 \text{ rpm}$, $4,482 \text{ lb/min}$ slag feeding rate). As shown in Figure G-10, a significant mass of particles is represented by the larger particle sizes. Hence, significant tower heights are needed to assure that most of the slag mass solidifies. Higher air temperatures increase the tower heights further.

In summary then, dispersion of slag into small particles can be achieved using the spinning disk technique. Disks without a pneumatic assist do not produce particles small enough to be contained in towers of practical diameters. Such disks would also require towers of impractical heights.

Figure G-14. Chart for determining temperature history at center of sphere.⁸⁸

Figure G-15. Contact tower heights necessary to solidify spherical particles of various diameters.

3. CONTACT TOWER DESIGN SUMMARY

The contact tower is designed to recover heat from molten slag. In the process, molten slag at 1,482°C (2,700°F) is dispersed, forming small slag particles. As the slag particles fall down through the tower, they are cooled to 760°C (1,400°F), simultaneously heating the surrounding air from 20°C (70°F) to 704°C (1,300°F). Some 59 kg/s (458,000 lb/hr) of hot air is produced, given a slag flow rate of 33.9 kg/s (269,000 lb/hr).

The contact tower is made of concrete and is 5.5 m (18 ft) in diameter and 15 m (50 ft) high.^a To facilitate gravity feeding of the molten slag, a portion of the tower is underground as sketched in Figure G-16. In many respects, the contact tower resembles existing prilling towers or a large concrete silo.

Molten slag from the surge vessel flows by gravity via an enclosed runner onto the spinning disk at the top of the tower. By means of the spinning disk and the compressed air jets around the disk, slag is fiberized and dispersed in the air inside the tower (see Figure G-2). The air and the slag particles move cocurrently^b through the tower and exchange heat.

^aTransfer of heat from slag particles to air is facilitated in 9 m (30 ft) of the tower height (see Section 2). An additional 6 m (20 ft) were assumed to provide for tower air feed, air removal, and slag feeding, dispersion, and removal.

^bCocurrent tower operation is suggested due to the relatively large amount of compressed air assumed for slag fiberization. Since this air is introduced at the top of the tower, in a countercurrent arrangement it would escape from the tower immediately, having hardly enough time to exchange heat with the slag particles moving to the tower bottom. This is not the case in the cocurrent arrangement where the compressed air flows through the tower and remains in contact with the slag particles. As suggested in Appendix H, compressed air requirements for the contact tower may be substantially reduced. If this reduction is achieved, the compressed air would no longer represent a significant fraction of tower total air throughput. Under such conditions, a countercurrent tower flow arrangement would become plausible, bringing along other advantages such as more efficient heat transfer and higher thermal energy recovery.

Figure G-16. Contact tower slag waste heat recovery, process diagram.

At the bottom, heated air exits the tower while the relatively large slag particles continue to settle into the collection bin. Retaining some slag particles, the heated air passes through a cyclone separator before it is utilized as a source of heat. Accumulated slag is removed from the tower bottom via a screw conveyor.

Dispersion of molten slag is facilitated by a spinning disk surrounded by a series of compressed air nozzles. The spinning disk, made of Hastelloy C, is 69 cm (27 in.) in diameter and 5 cm (2 in.) thick. The disk operates at 29.5 s^{-1} (1,750 rpm) and is driven by a 163-kW (219-hp) electric motor.

The tower needs both the compressed air and ambient air for its operation.

Some 75 kg/s (269,000 lb/hr) of compressed air at 791 kPa (100 psig) enters the tower through the compressed air nozzles surrounding the spinning disk. This air is needed to fiberize the slag and deflect the formed slag particles downwards, thus reducing the tower diameter. The amount and pressure of the compressed air were determined based on past studies on producing insulation from glass and/or slag.^a The energy required to drive a compressor capable of delivering this amount of compressed air was determined to be about 9,000 kW (12,000 hp).⁹⁰

Compressed air alone could lower slag temperature to only about 1,149°C (2,100°F). In order for the contact tower to reduce the slag temperature to 760°C (1,400°F) and produce air at 704°C

^aSeveral investigators studied the amount and pressure of the compressed air necessary to effect fiberization of mineral materials. Rau and Kibbel⁸⁰ used air and/or steam pressure at 825 kPa (105 psig). Porter⁸² used steam pressure of 446 kPa (50 psig). In his fiberizing study Downey³¹ used steam pressures between 584 kPa and 722 kPa (70 and 90 psig). He also recommended steam consumption (weight basis) that is slightly in excess of molten slag throughput.

(1,300°F), an additional 25 kg/s (199,000 lb/hr) of air is needed (see Figure G-6). This amount of air is supplied to the top of the contact tower by a 122-kW (164-hp) forced draft blower operating at 4.0 kPa (16 in. H₂O).

Auxiliary facilities include a screw conveyor and a cyclone separator. The conveyor is required to continuously remove accumulated slag particles from the tower bottom. A 0.44 m (16 in.) diameter, 30 m (100 ft) long screw conveyor of a standard design and capable of handling 34 kg/s (135 ton/hr) of material is considered adequate for this service. The cyclone separator removes residual slag particles from the heated air. A standard cyclone unit operating at about 2.0 kPa (8 in. H₂O) and handling 160 m³/s (339,000 cfm) is proposed for this service.

ABBREVIATIONS FOR APPENDIX G

A	-- $4\pi^2 t \rho_s = \text{constant}$ (see Equation G-7)
A_T	-- the rate of surface area formation, m^2/s (ft^2/min)
A_1	-- area of the surface (see Equation G-9), m^2 (ft^2)
C	-- aerodynamic drag coefficient for sphere (see Reference 89 for values) dimensionless
C_a	-- specific heat of air, $\text{J/kg}\cdot\text{K}$ ($\text{Btu/lb}\cdot^\circ\text{F}$)
C_s	-- specific heat of slag, $\text{J/kg}\cdot\text{K}$ ($\text{Btu/lb}\cdot^\circ\text{F}$)
D	-- particle diameter, m (ft)
d_i	-- diameter of particle, m (ft)
D_{max}	-- maximum particle diameter, m (ft)
D_p	-- characteristic dimension of a particle, m (ft)
D_{sv}	-- surface per unit volume average particle diameter (see Equation G-23), m (ft)
F_a	-- a geometrical factor (see Equation G-9), dimensionless
F_b	-- factor based on the emissivities of the two surfaces (see Equation G-9), dimensionless
g	-- gravitational constant, m/s^2 (ft/s^2)
H	-- tower height, m (ft)
h_c	-- forced convection heat-transfer coefficient, $\text{W/m}^2\cdot\text{K}$ ($\text{Btu/hr}\cdot\text{ft}^2\cdot^\circ\text{F}$)
h_f	-- heat of fusion of slag, J/kg (Btu/lb)
k_a	-- thermal conductivity of air, $\text{W/m}\cdot\text{K}$ ($\text{Btu/hr}\cdot\text{ft}\cdot^\circ\text{F}$)
k_s	-- thermal conductivity of slag, $\text{W/m}\cdot\text{K}$ ($\text{Btu/hr}\cdot\text{ft}\cdot^\circ\text{F}$)
L	-- wetted periphery of disk, m (ft)
m	-- relative boundary resistance, dimensionless
n	-- speed of rotation of the disk, s^{-1} (rpm)
Nu	-- Nusselt number, dimensionless
P_n	-- net power required for atomization, W (hp)
Pr	-- Prandtl number, dimensionless
Q	-- volumetric flow rate of molten slag, m^3/s (ft^3/min)
q_a	-- rate of thermal energy gain by air, W (Btu/min)
q_n	-- rate of thermal energy transfer between slag and air, W (Btu/min)

ABBREVIATIONS (continued)

q_{net}	-- rate of heat transfer by radiation, W, (Btu/min)
q_s	-- rate at which thermal energy is given up by slag, W (Btu/min)
r	-- radius of the centrifugal disk, m (ft)
r_d	-- radius of a sphere or a cylinder, m (ft)
Re	-- Reynolds number, dimensionless
t	-- apparent molten slag thickness at the periphery of the disk, m (ft)
T_A, T_B	-- absolute temperatures of respective surfaces (see Equation G-9), K ($^{\circ}\text{R}$)
T_a	-- average tower air temperature, $^{\circ}\text{C}$ ($^{\circ}\text{F}$)
T_b	-- initial temperature of the slag, $^{\circ}\text{C}$ ($^{\circ}\text{F}$)
ΔT_m	-- mean temperature difference, K, $^{\circ}\text{C}$ ($^{\circ}\text{F}$)
T_o	-- final center temperature of a particle, $^{\circ}\text{C}$ ($^{\circ}\text{F}$)
T_1	-- slag temperature at entrance to contact tower, $^{\circ}\text{C}$ ($^{\circ}\text{F}$)
T_2	-- slag temperature at exit from contact tower (excluding fixed bed region), $^{\circ}\text{C}$ ($^{\circ}\text{F}$)
T_3	-- air temperature at entrance to contact tower, $^{\circ}\text{C}$ ($^{\circ}\text{F}$)
T_4	-- air temperature at exit from contact tower (excluding fixed bed region), $^{\circ}\text{C}$ ($^{\circ}\text{F}$)
V	-- disk tangential velocity, m/s (ft/min)
$\bar{V}$	-- absolute particle velocity through the tower, m/s (ft/s)
V'	-- velocity at which slag leaves the disk, m/s (ft/min)
V_{air}	-- air velocity through the tower, m/s (ft/s)
V_h	-- horizontal component of particle velocity, m/s (ft/s)
V_T	-- velocity of the particle (relative to air stream), m/s (ft/s)
V_v	-- vertical (downward) component of particle velocity, m/s (ft/s)
W_a	-- air flow rate, kg/s (lb/min)
W_s	-- mass flow rate of slag, kg/s (lb/min)
X	-- dimensionless parameter (see Equation G-31)
X_h	-- horizontal distance travelled by ejected particle, m (ft)

ABBREVIATIONS (continued)

Y_o	-- temperature ratio, dimensionless
α	-- thermal diffusivity of slag, m^2/s (ft^2/hr)
α_h	-- Hatch dispersion coefficient defined as ratio of 50% size to 84.13% size, or of 15.87% size to 50% size taken to give a number greater than 1 (see Equation G-26), dimensionless
Γ	-- feed rate of liquid atomized based on wetted periphery $kg/s \cdot m$ ($lb/min \cdot ft$)
δ	-- surface tension of liquid feed (slag), kg/s^2 (lb/min^2)
η	-- contact tower heat recovery efficiency, %
η_i	-- number of particles, dimensionless
μ_a	-- viscosity of air, $Pa \cdot s$ ($lb/s \cdot ft^2$)
μ_s	-- viscosity of liquid feed (slag), $kg/s \cdot m$ ($lb/ft \cdot min$)
ρ_a	-- density of air, kg/m^3 (lb/ft^3)
ρ_s	-- density of liquid feed (slag), kg/m^3 (lb/ft^3)
σ_s	-- Stefan-Boltzmann constant = $5.670 \times 10^{-8} W/m^2 \cdot K^4$ ($0.1712 \times 10^{-8} Btu/hr \cdot ft^2 \cdot ^\circ R^4$)
τ	-- time, s (s)
τ_F	-- time particles move through the suspended particle region (particle tower residence time), s (min)

APPENDIX H

SLAG WASTE HEAT RECOVERY - CONTACT TOWER

Economics

CONTENTS

	<u>Page</u>
1. Capital Cost Summary	216
2. Operating Cost	219
3. Capital Cost Derivations	220

This appendix presents capital and operating costs for energy recovery via the contact tower. These costs are summarized and discussed in Sections 1 and 2. Capital cost derivations are presented in Section 3.

1. CAPITAL COST SUMMARY

Capital requirements for the contact tower are summarized below. All equipment costs include installation and are representative of December 1977 - January 1978 price levels.

Contact tower	\$ 140,000
Fiberizing disk and drive	16,000
Compressors	1,760,000
Blower	39,000
Cyclone(s)	260,000
Screw conveyor	35,000
Instrumentation and controls	135,000
Site preparation	1,000
Total direct costs	2,386,000
Engineering, supervision and construction	480,000
Total direct and indirect costs	2,866,000
Contractor's fee	140,000
Contingency	570,000
Fixed capital investment	3,576,000
Startup	350,000
Working capital	120,000
Total capital investment	<u>\$4,046,000</u>

Due to the preliminary nature of the design, it is difficult to assess the accuracy of the preceding estimates. One major consideration is the compressor discharge pressure and air flow rate needed to fiberize the molten slag particles spun off the disk. To produce insulation-quality mineral fibers, the following requirements must generally be met:^{80,91}

Discharge pressure - 791 kPa (100 psig)

Flow rate - greater than or equal to the mass flow
rate of slag (in this case, 60,000 cfm)

These criteria were used to design the contact tower fiberization system. However, the requirements for an energy recovery system may be much less stringent.^a This is important because the two 14.2 m²/s (30,000 cfm), 791 kPa (100 psig) compressors account for more than 70% of the direct costs.

However, if the fibers produced under these conditions prove to be suitable for insulation, the high capital requirement for the compressors can probably be justified by the additional byproduct credit. This, in turn, assumes that a reasonably large market is available for loose wool fiber insulation. Thus, there are obvious uncertainties associated with the design and capital cost for the slag fiberization.

^aSince extremely large surfaces need to be formed in making fibers of small diameter, the contact tower, which can operate with large size fibers, may require lesser quantities of compressed air. While the amount of compressed air needed for the contact tower operation could not be determined, an indication of potential compressed air reduction may be obtained by comparing the surface areas of fibers formed. The data for making such a comparison are presented in Section 2.3.3, Appendix G. Comparison of the surfaces involved in making fibers 0.04 mm to 0.020 mm in diameter (typical for fiberglass manufacture) with those 0.12 mm in diameter (size acceptable for the contact tower operation) provides an indication of potential reduction in energy consumption. A surface area 4 to 28 times smaller is formed in the case of fibers with larger diameters, suggesting that the contact tower may be able to operate with significantly smaller amounts of energy. Because the fiberization of slag is a rather complex process and many factors may influence the fiber surface formation (e.g., fiberization efficiency, spinning disk/compressed air nozzle geometry, slag temperature and viscosity, etc.), a more accurate estimate of energy requirements for the contact tower could not be made. The contact tower was assumed to require amounts of energy that are typically required for fiberglass and mineral wool manufacture.

Costs for the contact tower and fiberization system are based on direct industry estimates. However, capital requirements for the blower, cyclones, and conveyor are estimated from literature data, scaled to December 1977 costs. This approach is deemed appropriate due to the preliminary nature of the design and the relatively small influence of these costs on the overall capital requirement.

Costs for instrumentation, controls, and indirect items are estimated as percentages of the indirectly identifiable costs. Factors used to estimate these costs are as follows:²⁶
instrumentation and control--6% of the installed equipment cost; engineering, supervision, and construction--20% of the total direct cost; and contractor's fee--5% of the total direct and indirect cost.

A 20% contingency factor is also included in the fixed capital estimate to allow for unforeseen equipment requirements, design uncertainties relating to major equipment items, and freight charges, which are too site specific for direct estimation.

The overall cost for startup includes allowances for the following factors:²⁵

- Operating labor hired 2 months early; the associated overhead rate is 25%.
- One extra operator needed each shift for the first 3 months.
- One engineer required for 6 months.
- Initially low operating efficiency at a penalty of 10% of the variable costs for 6 months.
- 10% contingency on items 1 through 4.
- A "fix-it" charge equivalent to 4% of the fixed capital investment to account for excessive maintenance and minor design alterations during the initial operating period.

Working capital is calculated as accounts receivable plus inventories less accounts payable. Here, accounts receivable are defined as the total hot-air production over a 40-day period. This hot air is valued at the overall cost to produce it. Accounts payable include 2 weeks' operating and maintenance labor cost, 1 month's maintenance materials cost, and 35 days' utilities cost. The only inventory item is a 3-month supply of maintenance parts.

2. OPERATING COST

Contact tower annual operating costs are as follows:

Labor	\$ 122,000
Power	1,330,000
Cooling water	19,000
Maintenance	25,000
Total direct costs	1,686,000
Depreciation	358,000
Taxes and insurance	72,000
Total fixed costs	430,000
Overhead	115,000
Total operating costs	<u>\$2,231,000</u>

Electric power is the major operating expense, accounting for 80% of the direct costs and 60% of the total cost. This cost is based on 8,000 hours per year of operation, a unit charge of \$0.018/kWh, and the following power demands:

Compressors	8,919 kW (12,000 hp)
Disk drive	163 kW (219 hp)
Blower	122 kW (164 hp)
Conveyor	2 kW (3 hp)
Total	9,236 kW (12,386 hp)

As shown, the compressors consume 97% of the total power requirement. Thus, the operating cost for the system is quite sensitive to the size of compressor specified in a final design. This variable is discussed in the preceding section.

It is estimated that two men per shift will be needed to operate the contact tower system. At a wage rate of \$7.00/hr, this results in an annual cost of \$122,000. Since one supervisor is already needed to watch over the slag handling operation, no additional supervision should be required for the energy recovery system itself.

Approximately $5 \times 10^{-2} \text{ m}^3/\text{s}$ (800 gpm) of cooling water are required for the air compressors. This corresponds to an annual cost of \$19,000 assuming a 5¢/1,000 gal charge for cooling water.

Costs for maintenance material and labor are estimated at 6% of the fixed capital investment. A 50-50 split between material and labor cost is assumed.

Fixed charges to capital are calculated by depreciating the fixed investment over 10 years on a straight-line basis. No salvage values are assumed. Taxes and insurance are assumed to cost 2% of the fixed capital investment on an annual basis, and overhead charges are estimated at 50% of the cost for operating and maintenance labor.

3. CAPITAL COST DERIVATIONS

Capital costs presented in Section 1 include the purchase price and installation (i.e., foundations, electrical, paint, and field labor). These costs are itemized in this section for each major equipment requirement.

Contact Tower

Concrete tower	\$ 49,000 ^a
Installation @ 10% ⁹²	4,000 ^b
Insulation	<u>83,400^b</u>
Total	137,300
	\$ 140,000

Spinning Disk and Drive

Disk	\$ 4,700 ⁹³
Motor	5,800 ²⁶
Installation/auxiliary @ 50%	<u>5,250</u>
Total	\$ 15,750
	\$ 16,000

Compressors (2 complete units)

Compressors	\$1,100,000 ⁹⁰
Installation @ 60%	<u>600,000^{26, 92}</u>
Total	\$1,760,000

Blower (42,000 cfm, 16 in. H₂O)

Blower	\$ 20,000 ⁹²
Motor	4,500 ²⁶
Installation/auxiliaries @ 59%	<u>14,500⁹²</u>
Total	\$ 39,000

Cyclones (339,000 cfm)

Cyclones	\$ 153,000 ⁹²
Installation @ 69%	<u>106,000⁹²</u>
Total	\$ 259,000
	\$ 260,000

^a\$36,000 for a 30-ft tower;⁹⁴ cost scaled to \$49,000 for a 50-ft tower by the 0.6 power scaleup factor.

^b4-in. high alumina insulation at \$25/ft².²⁷

Screw Conveyor (16 in. D, 135 TPH, stainless steel, 100 ft)

Conveyor	\$21,900 ⁹²
Installation @ 59%	<u>12,900⁹²</u>
Total	\$34,800
	\$35,000

Site Preparation

700 yd³ excavated for the contact tower at a unit costs of \$1.50 yd³.⁹⁵ Other site preparation costs are included in the capital estimate for the slag handling and transportation system, Appendix C.

APPENDIX I
HEAT RECOVERY BOILERS
Technical Discussion and Economics

CONTENTS

	<u>Page</u>
1. Technical Discussion	224
2. Economics	225
2.1 Capital Costs	225
2.2 Operating Costs	227

1. TECHNICAL DISCUSSION

Hot air from the contact tower may be converted to steam in a heat recovery boiler. This boiler, within certain limits, may be designed to produce different qualities of steam depending on the conditions of available hot air (see Section 2.3.2 and Figure G-8 in Appendix G) and the steam end use.

Two potential steam utilization alternatives have been identified:

- electrical generation via a condensing turbine/electricity generator, and
- electric motor displacement by noncondensing steam-driven turbines.

Considering the two utilization alternatives, final selection of the heat recovery boilers was made based on the combined minimum cost for the boiler and the turbine(s). This selection defined the boiler operating conditions, which are identical for both steam utilization alternatives. The boilers use 59 kg/s (130 lb/s) of air at 704°C (1,300°F).

Desired steam qualities and associated flow rates for these two boilers are summarized below.⁹⁶

Application	Steam flow ^a		Steam pressure		Steam temperature		Stack temperature		Boiler ΔP	
	kg/s	lb/hr	MPa	psig	°C	°F	°C	°F	kPa	in. H ₂ O
1	11.3	89,800	4.24	600	399	750	139	283	1.6	6.3
2	12.1	96,300	1.83	250	232	450	147	297	1.2	4.9

^aFor the electricity generation application, the boiler inlet water temperature is assumed to be 46°C (115°F) due to the 100% condensate recycle. For the electric motor displacement application, however, the steam will not be condensed and recycled. Thus, the inlet water temperature is taken as 21°C (70°F).

2. ECONOMICS

2.1 Capital Costs

Capital requirements for the two heat recovery boilers are summarized below. All costs are in January 1978 dollars.

	Condensing turbine/ electricity generator alternative	Noncondens- ing turbine/ electric motor displacement alternative
Heat recovery boiler (package)	\$ 824,000	\$ 729,000
Installation	410,000	360,000
Total direct cost	1,234,000	1,089,000
Engineering, supervision, and construction	250,000	220,000
Total direct and indirect cost	1,484,000	1,309,000
Contractor's fee	74,000	65,000
Contingency	150,000	130,000
Fixed capital investment	1,708,000	1,504,000
Startup	130,000	120,000
Working capital	52,000	48,000
Total capital investment	<u>\$1,890,000</u>	<u>\$1,672,000</u>

Purchased costs for the two boilers are based on industry estimates.⁹⁶ Installation in each case is assumed at 50% of the purchased cost, a typical amount for packaged systems.⁶² Because these boilers are part of a larger energy recovery system, they are allotted a proportionate share of the indirect construction overhead expense. Costs for engineering, supervision, and construction overhead are estimated at 20% of the direct plant cost. Likewise, a contractor's fee of 5% of the combined direct and indirect costs is assumed.

A 10% contingency allowance is also included in the overall fixed capital investment figure.

Startup costs include the following items:²⁵

- Operating labor hired 2 months early.
- A 25% labor burden prior to startup.
- One extra operator per shift for 6 months.
- 10% penalty in variable costs for 6 months due to low initial operation efficiencies.
- 10% contingency on the above items.
- An initial "fix-it" charge equivalent to 4% of the fixed capital investment. This covers excessive maintenance and minor design alterations over the initial operating period.

Working capital is defined as accounts receivable plus inventories less accounts payable. Items which fall into these categories include:

- Accounts Receivable:
40 days of steam production valued at its total production cost.
- Inventories:
3-month supply of maintenance materials.
- Accounts Payable:
2 weeks' operating and maintenance labor costs, and 1 month's utilities and maintenance materials cost.

2.2 Operating Costs

Annual operating costs for the two heat recovery boilers are summarized below. All costs are in late 1977-early 1978 dollars.

	Condensing turbine/ electricity generator alternative	Noncondens- ing turbine/ electric motor displacement alternative
Boiler feed water	\$ 1,000	\$ 33,000
Labor	122,000	122,000
Maintenance	68,000	60,000
Power	10,000	5,000
Total direct cost	201,000	220,000
Depreciation	171,000	150,000
Taxes and insurance	34,000	30,000
Total fixed cost	205,000	180,000
Overhead	78,000	76,000
Total operating cost	<u>\$484,000</u>	<u>\$476,000</u>

Unit boiler feed water costs for both systems are assumed to be the same as those for the float chamber feed water: \$0.095/1,000 kg (\$0.043/1,000 lb) (see Appendix F). However, there is a large difference in feed water requirements for the two boilers.

With the condensing turbine, 100% of the steam produced in the boiler is condensed and recycled. Blowdown requirements are minimal, approximately 2% of the steam export rate.⁹⁷ Thus, the makeup requirement is only 0.2 kg/s (1,800 lb/hr). Assuming 8,000-hr/yr operation, this corresponds to an annual cost of \$620 (rounded to \$1,000).

The noncondensing turbines used in place of electric motors, however, do not condense the steam. Hence, a once-through steam

generation system is required. This higher feed water demand results in an annual expenditure of \$33,000, assuming 8,000-hr/yr operation and a steam production rate of 12 kg/s (96,300 lb/hr).

To determine labor costs, it is assumed that two men per shift (8,760 hr/yr) are needed to operate either boiler. At \$7.00/hr, this corresponds to an annual labor cost of \$122,000.

Maintenance costs are estimated at 4% of the fixed capital investment for both cases. A 50-50 split between material and labor costs is assumed.

Power requirements for the feed water pumps are estimated at 69 kW (93 hp) for the turbo-generation system and 31 kW (42 hp) for the electric motor displacement system. Assuming 8,000-hr/yr operation and a power cost of 1.8 mills, the annual power costs for the two systems are \$10,000 and \$5,000, respectively. Since both boilers employ natural circulation, no circulating pumps are needed.

To determine fixed charges to capital, 10-year straight-line depreciation is assumed with no allowance for salvage. Costs for taxes and insurance are estimated at 2% of the fixed capital investment. Incremental overhead costs are assumed to be 50% of the combined charges for operating and maintenance labor.

APPENDIX J

CONDENSING TURBINE/ELECTRICITY GENERATOR

Technical Discussion and Economics

CONTENTS

	<u>Page</u>
1. Technical Discussion	230
2. Economics	231
2.1 Capital Costs	231
2.2 Operating Costs	232

1. TECHNICAL DISCUSSION

Both the float chamber and the contact tower heat recovery boiler produce steam which can be used to produce electricity in turbo-generators. The contact tower heat recovery boiler generates 11.3 kg/s (89,800 lb/hr) of steam at 4.24 MPa (600 psig) and 399°C (750°F). The temperature-pressure combination is the minimum required for economical electricity generation.^{98,a}

The float chamber is designed to recover 55.7 MW (1.9×10^8 Btu/hr) of waste heat as 19.4 kg/s (154,000 lb/hr) of steam at 3.10 MPa (450 psig) and 316°C (100°F). However, only minor design modifications would be necessary to produce steam at the required conditions (399°C [750°F]) and 4.24 MPa [600 psig]). These modifications should not have a pronounced effect on the economics of the float chamber. Due to the changed steam conditions, the output from the float chamber is adjusted to 18.4 kg/s (146,000 lb/hr) of steam at 4.24 MPa (600 psig). Here, an inlet water temperature of 46°C (115°F) is assumed. About 98% of the float chamber feed water is condensate recycle.

General Electric Company was consulted for turbine/electricity generator selection for both the float chamber and the contact tower.⁹⁸ Straight-condensing units were recommended in both cases. These units include a 60 s^{-1} (3,600 rpm) steam turbine, a 10.1 kPa (3.0 in. Hg absolute) condenser, and a three-phase, 60 Hz, air-cooled synchronous generator. Peak electrical outputs from the float chamber unit and the contact tower unit are 15,500 kW and 9,400 kW, respectively.

^a400 psig, 600°F steam was also considered for electrical generation and found to be much less economical than the 600 psig alternative.

2. ECONOMICS

2.1 Capital Costs

Capital requirements for the float chamber and the contact tower turbo-electric generators are as follows:

	<u>Float chamber</u>	<u>Contact tower</u>
Turbine-electricity generator (complete unit)	\$1,660,000	\$1,180,000
Installation	750,000	530,000
Total direct cost	2,410,000	1,710,000
Construction expenses	240,000	170,000
Total direct and indirect cost	2,650,000	1,880,000
Contractor's fee	130,000	94,000
Contingency	260,000	190,000
Fixed capital investment	3,040,000	2,164,000
Startup	140,000	100,000
Working capital	72,000	54,000
Total capital investment	<u>\$3,252,000</u>	<u>\$2,318,000</u>

Purchased costs for the turbine-electricity generator packages are based on information provided by General Electric Company.⁹⁸ Installation costs were assumed to be 45% of the purchased costs.

Engineering, supervision, and construction overhead costs for the entire energy recovery systems are estimated at 20% of the direct costs.²⁶ However, engineering design and construction supervision costs are included in the purchase price. Thus construction overhead is the only indirect cost incurred other than a fee for the construction company's services. Construction overhead is estimated at 10% of the direct costs. The contractor's fee is estimated at 5% of the direct and indirect costs. A 10% contingency factor is also included in the fixed capital investment.

Startup costs for the two units are estimated at \$140,000 and \$100,000. This covers the costs for operating labor hired 2 months prior to startup, 25% overhead, 10% contingency on the labor cost, and a "fix-it" charge equivalent to 4% of the fixed capital investment. This charge covers excessive maintenance and minor design alterations over the initial operating period.²⁵

Working capital is calculated as accounts receivable plus inventories less accounts payable. Accounts receivable are equivalent to a 40-day operating cost for the unit, the total cost incurred for 40 days' electrical output. A 3-month supply of maintenance materials is the only inventory associated with the operation. Accounts payable include 2 weeks' operating and maintenance labor costs, and a 30-day supply of maintenance parts.

2.2 Operating Costs

Operating costs for the two turbo-generator units are as follows:

	<u>Float chamber</u>	<u>Contact tower</u>
Labor	\$ 61,000	\$ 61,000
Maintenance	122,000	87,000
Total direct cost	183,000	148,000
Depreciation	304,000	216,000
Taxes and insurance	61,000	43,000
Total fixed costs	365,000	259,000
Overhead	61,000	52,000
Total operating costs	<u>\$609,000</u>	<u>\$459,000</u>

It is assumed that one operator will be required each shift to look over the turbo-generator installation and perform miscellaneous functions throughout the energy recovery plant. Assuming a wage rate of \$7.00 per hour, this results in an operating labor

cost of \$61,000 per year. For both turbo-generator units, annual maintenance charges are estimated at 4% of the fixed capital investment. A 50-50 split between material and labor costs is assumed.

Ten-year straight-line depreciation with no salvage value is used to calculate the fixed charges to capital. Tax and insurance costs are estimated at 2% of the fixed capital investment on a yearly basis.

Overhead charges are estimated at 50% of the annual costs for operating and maintenance labor. This charge covers fringe benefits and other potential increases in overall plant overhead.

APPENDIX K

NONCONDENSING TURBINES/ELECTRIC MOTOR DISPLACEMENT Technical Discussion and Economics

CONTENTS

	<u>Page</u>
1. Technical Discussion	235
2. Economics	235
2.1 Capital Costs	235
2.2 Operating Costs	237

1. TECHNICAL DISCUSSION

Steam produced in the float chamber or the contact tower heat recovery boiler can be used to drive prime movers at the plant site. Steam turbines can be used separately or added to existing systems by installing a dual shaft, permitting either electric or steam operation.

A preliminary economic evaluation was performed to determine optimum steam conditions for single-stage turbine operation. Steam at 1.83 MPa (250 psig) and 232°C (450°F) proved to be the least-cost alternative.⁹⁹

The contact tower heat recovery boiler produces 12.1 kg/s (96,300 lb/hr) of steam at these conditions. Here, an inlet water temperature of 21°C (70°F) is assumed because the single-stage turbines do not condense the steam for recycle.

Water rate for turbines is about 4×10^{-6} kg/s·W (23.6 lb/hr·hp). Thus, the contact tower can generate a total of 3.0 MW (4,080 hp). This is assumed to replace eight 370-kW (500-hp) motors. The float chamber transfers sufficient heat from the slag to produce 20.0 kg/s (159,000 lb/hr) of steam at 1.83 MPa (250 psig) and 232°C (450°F), and can generate a total of 5.0 MW (67,740 hp). This is assumed to replace eleven 370-kW (500-hp) motors and three 300-kW (400-hp) motors.

2. ECONOMICS

2.1 Capital Costs

Capital requirements for the noncondensing turbines are as follows:

	<u>Float chamber</u>	<u>Contact tower</u>
Steam turbines (complete units)	\$364,000	\$208,000
Installation	164,000	94,000
Total direct costs	528,000	302,000
Engineering, supervision, and construction	106,000	60,000
Total direct and indirect costs	634,000	362,000
Contractor's fee	32,000	18,000
Contingency	63,000	36,000
Fixed capital investment	729,000	416,000
Startup	29,000	17,000
Working capital	17,000	9,000
Total capital investment	<u>\$775,000</u>	<u>\$442,000</u>

Purchased costs are the same for the 300-kW (400-hp) and 370-kW (500-hp) turbines because the same components are used. Each unit costs approximately \$26,000.⁹⁹ Installation is assumed at 45% of the purchased cost.

Indirect costs for the overall energy recovery system are estimated as percentages of the direct cost:²⁶ engineering, supervision, and construction overhead--20% of the total direct cost; contractor's fee--5% of the direct and indirect costs. A 10% contingency allowance is also included in the fixed capital estimate.

Startup costs are estimated at 4% of the fixed capital investment.²⁵ This is considered to be a reasonable allowance for minor design changes and excessive maintenance during the initial operating period.^a

^aFluctuating steam outputs, temporary shutdowns, and restartups are potential sources of maintenance problems.

Working capital is calculated as accounts receivable plus inventories less accounts payable. Here, accounts receivable are equivalent to 40 days' energy output, valued at the total operating cost over this period. A 3-month supply of maintenance parts is the only inventory item associated with the operation.

Accounts payable include 2 weeks' maintenance labor cost, a 1-month supply of maintenance materials, and 1 month's utilities cost.²⁵

2.2 Operating Costs

	<u>Float chamber</u>	<u>Contact tower</u>
Incremental feed water consumption	\$ 54,000	\$ --
Maintenance	29,000	17,000
Total direct costs	83,000	17,000
Depreciation	73,000	42,000
Taxes and insurance	15,000	8,000
Total fixed costs	88,000	50,000
Overhead	7,000	4,000
Total operating cost	<u>\$178,000</u>	<u>\$71,000</u>

Operating costs for the float chamber assume a makeup water consumption of 0.38 kg/s (3,000 lb/hr), with condensate recycle comprising the rest of the feed water. If used in conjunction with noncondensing turbines, however, the water consumption will increase to 20.0 kg/s (159,000 lb/hr). Thus, the cost to treat the additional 20.0 kg/s (156,000 lb/hr) of water is included here. Based on 8,000 hr/yr of operation and a unit cost of \$0.095/1,000 kg (\$0.043/1,000 lb) (see Appendix F), this annual charge will total \$54,000.

Maintenance, depreciation, insurance, and teaxes are all calculated as percentages of the fixed capital investment:

Maintenance--4% (2% for labor, 2% for materials)

Depreciation--10% (10-year, straight-line, no salvage)

Taxes and insurance--2%

Overhead costs are estimated at 50% of the cost for maintenance labor. No operating labor is required.

APPENDIX L

ENERGY LOSSES DURING SLAG TRANSPORTATION

A Crude Estimate

Thermal losses will occur during the transportation of slag from the phosphorus furnaces to the surge vessel. A very crude estimate of these losses is made below.^a

An average slag tapping volume is assumed at 3.5 m³ (123 ft³). Also assumed is a cylindrical slag pot 1.8 m in diameter and 1.3 m high (6 ft diameter x 4.4 ft high). The energy losses are assumed to occur from the top free surface (radiation) and from the slag pot walls and the bottom (conduction). Equation L-1 below was used in the calculation; the first term represents conduction losses through the walls, the second term represents conduction losses through the bottom, and the third term represents radiation losses through the top. Emissivity of 1 (black body) was assumed in the radiation term. Also, it was assumed that the cylinder has built up a 0.1-m (4-in.) slag layer with thermal conductance of 3.6 W/m·K (0.64 Btu/hr-ft-°F) around the walls and on the bottom. The thermal resistance of the steel shell was neglected.

$$G_L = 2\pi k_s L_V \frac{(T_S - T_A)}{\ln \frac{r_o}{r_i}} + \frac{k_s A_w (T_S - T_A)}{t_s} + A_w \sigma (T_S^4 - T_A^4) \quad (L-1)$$

^aSee Appendix B, Slag Properties, for values used in this calculation.

where G_L = thermal energy loss
 k_s = thermal conductivity of slag layer
 L_V = height of the vessel
 A_w = area of the top of the vessel
 t_s = thickness of the solidified slag layer
 r_o = outside radius of the vessel
 r_i = inside radius of the vessel
 σ = Stefan-Boltzmann constant (0.174×10^{-8} Btu/hr-ft²-°R⁴)
 T_S = initial temperature of the slag (3, 212°R)
 T_A = ambient temperature (530°R)

Calculation yields a heat loss:

$$G_{L \text{ total}} = 1.7 \times 10^6 \text{ W } (5.8 \times 10^6 \text{ Btu/hr})$$

$$G_{L \text{ radiant}} \approx 1.5 \times 10^6 \text{ W } (5.2 \times 10^6 \text{ Btu/hr})$$

$$G_{L \text{ conduction}} \approx 0.2 \times 10^6 \text{ W } (0.6 \times 10^6 \text{ Btu/hr})$$

Since the recoverable slag heat flow is about 70 MW (2.4×10^8 Btu/hr), in an hour slightly over 2% of the total slag heat may be lost if the slag remains in pots for this period of time. This total heat loss is equivalent to a slag temperature drop of about 22°C (72°F). Most of the loss is through radiation, suggesting the importance of shielding the slag during handling.

This calculation is far from precise. Nevertheless, it suggests that heat loss during slag transportation may be significant. Consequently, transportation should be implemented as quickly as possible.

APPENDIX M

SURGE VESSEL HEAT LOSS

A Crude Estimate

Some heat losses will occur during the residence of the slag in the surge vessel. These losses are crudely estimated below.

Consider a cylindrical surge vessel 3.4 m (11 ft) in diameter and 3.2 m (10.5 ft) high with 0.1 m (4 in.) of insulation on all sides. Assume the thermal conductivity of insulation at 1.1 W/m·K (0.64 Btu/hr-ft-°F), neglect the thermal resistance through the vessel wall, and design the vessel so as to minimize radiation losses. The following equation applies:

$$G_{LS} = \frac{2\pi k_s L_S (T_F - T_A)}{\ln \frac{r_o}{r_i}} + 2\pi r_o^2 k_s \frac{(T_F - T_A)}{t_s} \quad (M-1)$$

where G_{LS} = heat loss from the surge vessel

k_s = thermal conductivity of the insulation layer

L_S = height of the surge vessel

t_s = thickness of the insulation

r_o = outside radius of the surge vessel

r_i = inside radius of the surge vessel

T_A = ambient temperature (70°F)

T_F = slag temperature (2,680°F)

Based on the assumptions above, the heat losses were calculated to be 0.87×10^6 W (2.985×10^6 Btu/hr). This loss is equivalent to a temperature drop of about 6°C (11°F), a relatively minor drop.

APPENDIX N

CONVERSION FACTORS AND METRIC PREFIXES

CONVERSION FACTORS

<u>To convert from</u>	<u>To</u>	<u>Multiply by</u>
Btu/hr	W	2.930711×10^{-1}
Btu/hr·ft·°F	W/m·K	1.730735
Btu/hr·ft ² ·°F	W/m ² ·K	5.678263
Btu/hr·ft ² ·°R ⁴	W/m ² ·K ⁴	3.311564×10^1
Btu/lb·°F	J/kg·K	4.186800×10^3
ft	m	3.048000×10^{-1}
ft ²	m ²	9.290304×10^{-2}
ft ³ /hr	m ³ /s	7.865791×10^{-6}
ft ³ /min	m ³ /s	4.719474×10^{-4}
hp	W	7.460000×10^2
in.	m	2.540000×10^{-2}
in. ²	m ²	6.451600×10^{-4}
lb/hr	kg/s	1.259979×10^{-4}
lb/ft ³	kg/m ³	1.601846×10^1
lb/min	kg/s	7.559873×10^{-3}
psi	Pa	6.894757×10^{-3}

$$t_{\circ C} = (t_{\circ F} - 32)/1.8$$

$$t_K = (t_{\circ F} + 459.67)/1.8$$

$$t_K = t_{\circ F}/1.8$$

$$t_{\circ R} = t_{\circ F} + 459.67$$

METRIC PREFIXES

<u>Prefix</u>	<u>Symbol</u>	<u>Multiplication factor</u>	<u>Example</u>
P	peta	10^{15}	1 PJ = 1×10^{15} J
G	giga	10^9	1 Gg = 1×10^9 g
M	mega	10^6	1 MW = 1×10^6 W
k	kilo	10^3	1 kPa = 1×10^3 Pa
m	milli	10^{-3}	1 mg = 1×10^{-3} g
μ	micro	10^{-6}	1 μg = 1×10^{-6} g