RECOVERY OF COBALT FROM A VIBURNUM TREND LEAD BLAST FURNACE SLAG

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ABSTRACT

The Viburnum trend in south-east Missouri is a polymetallic mineralized district which is mined principally for lead. The lead is smelted and refined in Missouri while the zinc and copper concentrates are sold.

Lead blast furnace slag produced from the lead smelting process contains approximately 0.30% cobalt. The Doe Run Company in Herculaneum, Missouri, is investigating the feasibility of implementing the Mintek Enviroplas process to recover zinc from molten lead blast furnace slag. A molten iron phase is also produced, and cobalt could therefore potentially be recovered as a ferro-cobalt alloy, (which could be further processed to produce a saleable cobalt product).

A steady state simulation model has been developed based on data obtained from a 500 kW pilot scale furnace. The model was developed using Pyrosim simulation software. An economic evaluation of four processing routes has been undertaken (three of which are reported in detail), which includes the costing of a converting stage to upgrade the cobalt-containing alloy from the smelting stage to a higher grade ferro-cobalt alloy.

Keywords: Capital costs, Cobalt recovery, D.C. arc furnace, Doe Run, Economic evaluation, Enviroplas Process, Lead blast furnace slag, Mintek, Operating costs, Process simulation, Pyrosim, Sensitivity analysis, Simulation, Steady-state simulation, Viburnum Trend

Nickel – Cobalt 97

36th Annual Conference of Metallurgists, Sudbury, Canada, August 1997

Mintek Paper No. 8388

INTRODUCTION

The Viburnum trend of south-east Missouri has long been a potential, but to date, unutilized, cobalt resource. The Viburnum trend mineral deposits are found along the flank of the St. Francois Mountains and are mined primarily for lead, with zinc and copper (and some silver) also being recovered. Although they contain only small quantities of cobalt, the large tonnage of ore in the Southeast Missouri district represent the second largest reserve of cobalt in the United States (1).

The principal ore bearing minerals from the Viburnum trend include galena (PbS), sphalerite (ZnS), and chalcopyrite ($CuFeS_2$). The gangue minerals consist of calcite, dolomite, quartz, pyrite, and marcasite. The cobalt- and nickel-bearing minerals found are siegenite $(Ni,Co,Fe)_3S_4$ and bravoite $(Fe,Ni,Co)S_2$ (1). During ore genesis, the minerals were repeatedly deposited which, together with replacement of one sulfide by another, has resulted in a wide variety of complex intergrowth textures (1). Past attempts to recover cobalt (and nickel) from the ores have been uneconomic, owing to mineral processing and metallurgical processing problems (2).

Two main approaches have previously been applied to recover cobalt from the ores. One approach involved separating a cobalt-nickel concentrate from the copper concentrate, while another attempt was to make a cobalt-nickel concentrate prior to separating the lead, zinc, and copper concentrates. Intergrowths between galena, chalcopyrite, and sphalerite, and siegenite and bravoite, led to locked particles reporting to the various concentrates; the separations obtained were unsatisfactory.

As a consequence of zinc, lead, and copper recovery, (after flotation), the following distribution of cobalt results: roughly half of the cobalt reports to the tailings, while the remainder is distributed between the three concentrates. Approximately 20 percent of the original cobalt reports to the copper concentrate. Of the remaining 30 percent, 10 percent of the original cobalt reports to the zinc concentrate (where it, along with nickel, is regarded as an impurity). By difference, 20 percent of the cobalt recovered from the ore reports to the lead concentrate.

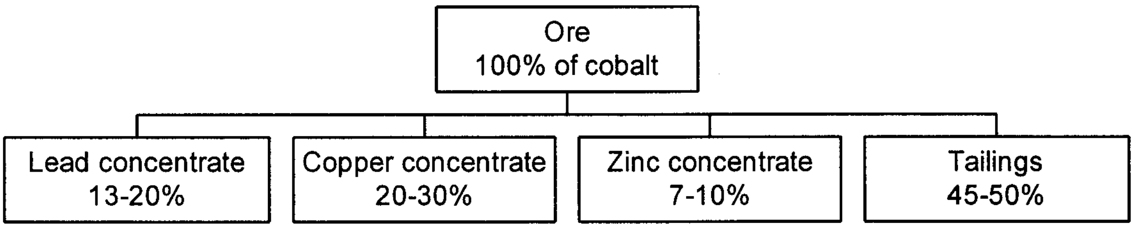


Figure 1 - Distribution of cobalt from the ore to the concentrates (3)

Figure 1 shows the distribution of cobalt to the concentrates. After flotation, the lead concentrate has an assay as is found in Table I (4).

Table I - Lead concentrate assay

Pb	Zn	S	Co	Ni	Fe	SiO_2	CaO
75	1.3	13.5	0.056*	0.075*	1.0	1.0	2.0

* Cobalt and nickel values are average values from composite samples for 1996, taken from three operating mines in the Viburnum Trend

The lead concentrate is fed to a sinter strand to agglomerate and desulfurize the concentrate. Cobalt sulfide (along with the other sulfide minerals) is oxidized to cobalt oxide. The sinter is fed to the blast furnace where coke is added as a fuel and reductant. The slag from the blast furnace is currently stockpiled.

Table II - Lead blast furnace slag assay (5)

ZnO	PbO	Co	CuO	S	FeO	Al_2O_3	SiO_2	CaO	MgO
15.3	2.71	0.30	0.5	2.40	30.6	4.83	22.8	11.0	4.60

Although the cobalt content of the lead concentrate is low, relatively low volumes of slag result from the lead smelting process, thereby leading to a large increase in concentration of cobalt (and other elements) in this stream. The lead blast furnace slag contains approximately 0.30% Co which is sufficiently high to justify further investigation into optimizing the recovery of cobalt as a by-product of zinc recovery. This paper discusses a possible means to recover the cobalt which reports to the lead concentrate, using a pyrometallurgical process (which aims to recover the zinc from the lead blast furnace slag).

ENVIROPLAS PLANT DESCRIPTION

The concentration of zinc in lead blast furnace slags is often sufficiently high to warrant further processing to extract the zinc. This has traditionally been conducted in zinc fuming furnaces (using submerged coal and air injection) (6, 7). Examples of these zinc fuming furnaces are found at Cominco at Trail, in British Columbia, Canada (8), and at BHAS at Port Pirie, Australia. An impure zinc oxide is produced which is further refined to saleable zinc.

The Enviroplas process (9,10) was developed at Mintek, South Africa, to recover the zinc from zinc rich materials, including lead blast furnace slag. The main goal of this process is to condense a zinc rich gas to Prime Western grade zinc in a lead splash condenser. The conditions under which the zinc is fumed are sufficiently reducing to form a metallic (either an alloy or matte) phase. The cobalt distribution to the alloy phase will depend on the degree of reduction achieved in the fuming furnace.

The process consists of three pyrometallurgical operations: a holding furnace; fuming furnace; and a lead splash condenser. A steady stream of molten slag is transferred from the holding furnace via an underflow weir to the fuming furnace in which the zinc is fumed off using coke as a reductant. The zinc rich vapor is condensed in the lead splash condenser to recover the zinc (and any lead) fumed from the slag. Although not considered in this study, the dross and fume from the condenser are recycled to the sinter strand of the blast furnace.

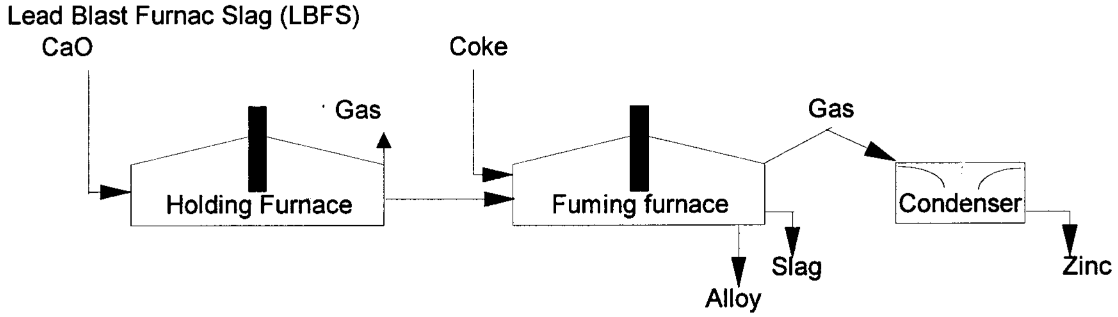


Figure 2 - Schematic diagram of the Enviroplas process (9,10)

The process aims to recover the zinc from the slag, while simultaneously treating the material such that it passes the United States EPA TCLP test for toxicity characterization. Passing this test classifies the material as non-hazardous, allowing it to be cheaply disposed of (or sold). The Enviroplas process is described in detail elsewhere (9,10).

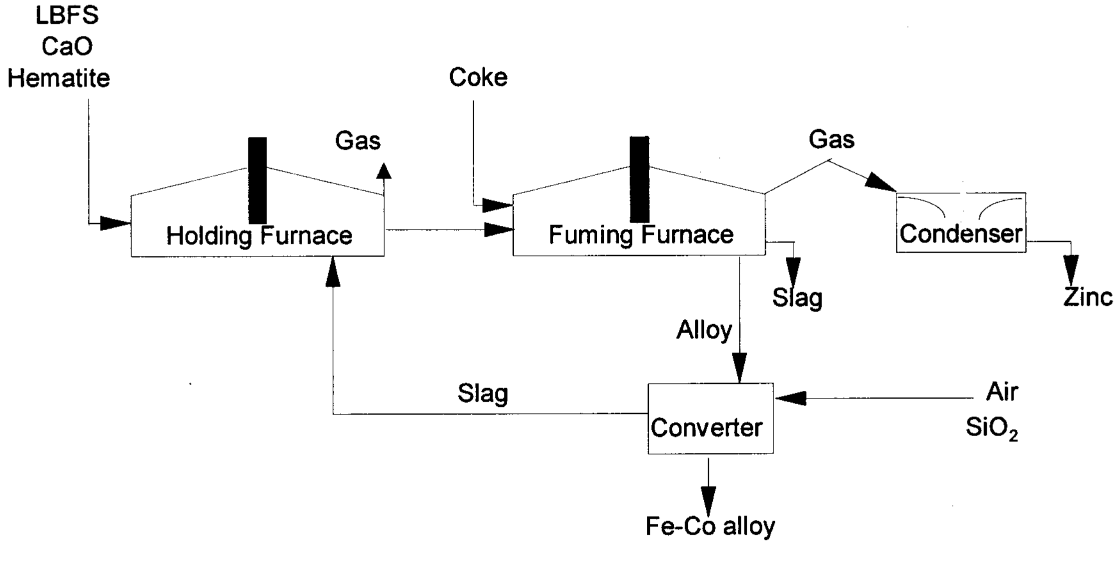


Figure 3 - Schematic diagram of the Enviroplas process with a converter included to upgrade the cobalt rich alloy produced in the fuming furnace

In this paper, variations on the Enviroplas process have been evaluated, with particular emphasis placed on cobalt recovery. A converting stage to upgrade the cobalt rich alloy has been considered as a means to reduce the quantity of iron which is sent for hyrometallurgical treatment. Figure 3 shows the Enviroplas process with the additional converting stage to upgrade the cobalt rich alloy.

The converting stage consists of blowing the alloy with air and silica to form a fayalite slag. This technique for alloy upgrading has not been evaluated in detail. An attempt has been made during this investigation to identify the driving forces at play, and possibly identify areas which may warrant further investigation.

Hydrometallurgical Treatment

The cobalt rich alloy produced in the fuming furnace will need further treatment to recover the cobalt and any other saleable metals. This could be done at an off-site refinery. However, the main deleterious constituents are iron and sulfur (for which a penalty will be incurred). A convenient method to dispose of the iron residue would be to recycle the iron as hematite to the holding furnace (where it will leave the process as iron oxide in the discard slag from the fuming furnace).

It is therefore proposed that an on-site hydrometallurgical treatment be used to treat the cobalt rich alloy. The following stages are suggested (after smelting):

- Water atomization (or granulation followed by milling);
- Pressure leach in an autoclave;
- Filter to remove iron as a hematite residue, (hematite is returned to the holding furnace to avoid disposal problems);
- Open tank for liquid (with addition of sodium carbonate), to precipitate basic copper sulfate;
- Filter to remove basic copper sulfate precipitate;
- Drier to dry the basic copper sulfate;
- Open tank to precipitate Co and Ni carbonate from the remaining solution;
- Filter for Co/Ni carbonate;
- Evaporator for Co/Ni carbonate.

Figure 4 shows a typical Enviroplas flowsheet with the proposed hyrometallurgical treatment stages included. This processing route for treating the alloy has been adopted for the economic analysis.

This process has the advantage of producing only saleable products and a disposable slag. The products would be: Prime Western grade zinc; basic copper sulfate; Co/Ni carbonate crystals; Na_2SO_4 solution.

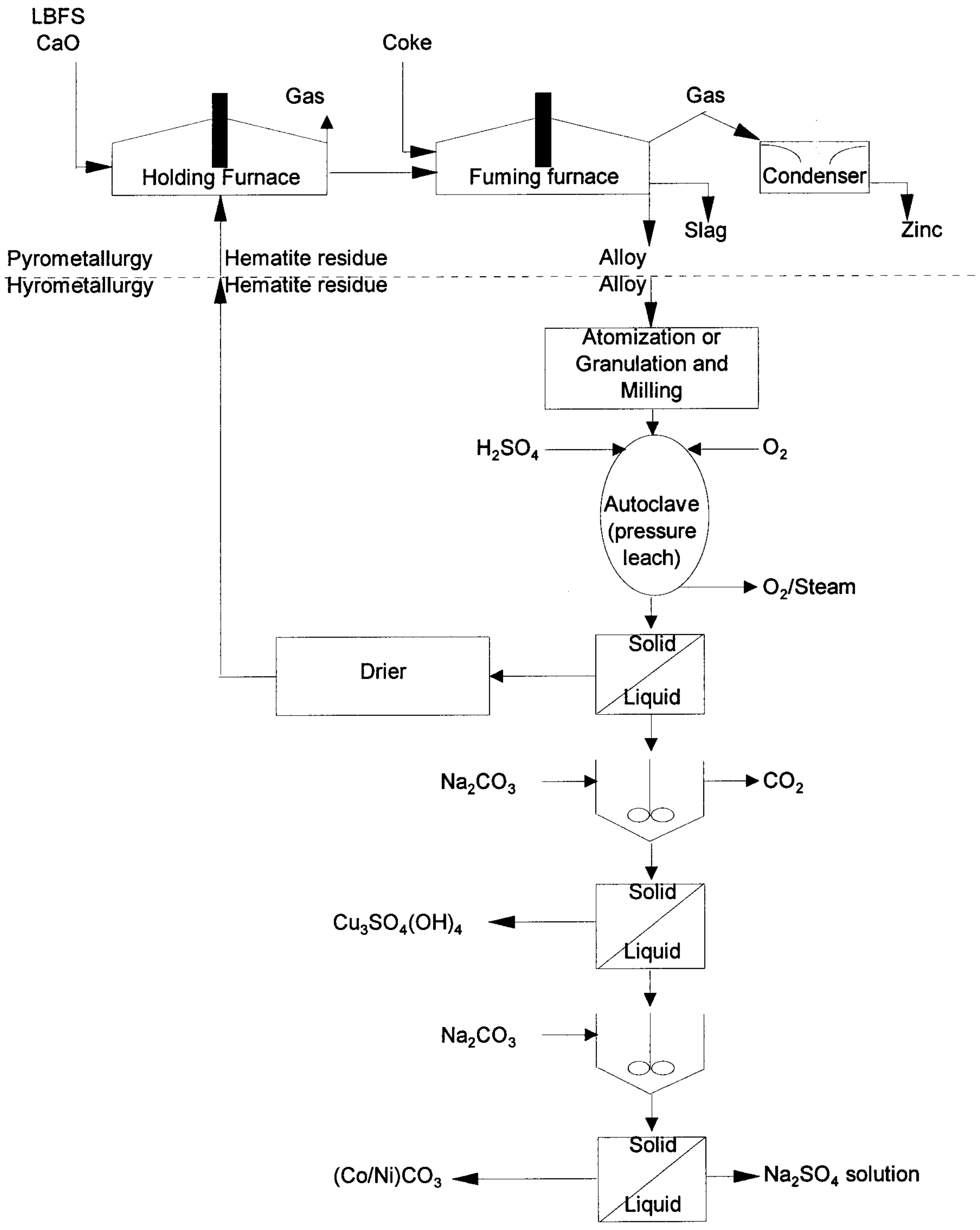


Figure 4 - Schematic diagram of the proposed hydrometallurgical flowsheet (with Option 2 shown as an example of the pyrometallurgical process)

SIMULATION METHOD

Pyrosim (11) simulation software was used to perform the pyrometallurgical process simulation calculations. Pyrosim was developed by Mintek for the calculation of steady state mass and energy balances and has been used routinely for similar studies of other base metal processes, including the cobalt recovery from copper reverberatory furnace slags (12).

The simulations were based on the assumption of chemical equilibrium between gas, slag and alloy. The equilibrium composition was calculated using free-energy minimization, together with the Ideal Mixing of Complex Components solution modeling technique.

The model used here does not attempt to take into account subtle differences in the mode of operation of a furnace, or include physical entrainment of one phase into another. The objective of the model is to provide a consistent means to evaluate the chemistry of each process. In particular, Pyrosim allow for the formulation of a mass and energy balance for each unique scenario, based on the feeds, products (including recycle streams), and selected operating conditions.

The hydrometallurgical simulations were carried out using Aspen Plus simulation software.

Basis

A basis of 1000 short tons per day of hot (1150°C) lead blast furnace slag (LBFS) was selected. This is equivalent to 37500 kg/hour, or about 281000 metric tons per annum.

Options

The following processing routes were selected for simulation and subsequent costing:

- **Option 1**: Zinc is removed to 2.5% ZnO in the discard slag, and the cobalt recovery is merely a consequence of the reducing conditions needed for zinc evolution. The cobalt recovery is low at around 40% of the incoming cobalt reporting to the alloy phase. It is assumed in this case that the cobalt-rich alloy is disposed of at no cost or benefit.

- **Option 2**: The reducing conditions in the furnace are selected (primarily by increasing the coke feed rate) to raise the cobalt recovery to 80% (incoming cobalt reporting to the alloy). The alloy is treated hydrometallurgically to produce a saleable Co/Ni carbonate.

- **Option 3**: This option describes cobalt recovery as in Option 2, but with a converting stage to upgrade the iron cobalt alloy phase, with the objective of reducing the size of the autoclave needed to leach the alloy. Further simulations, not reported on here, explored the effects of blowing to an even greater degree. This resulted in a much greater recycling load, and additional losses of cobalt.

EXPERIMENTAL RESULTS

Pilot plant data was used to verify certain assumptions made while conducting the simulations. The data was obtained from tests performed at Mintek during late 1995 (13).

Pilot-scale test data

Test work was conducted at the 500 kW scale on a DC arc furnace to quantify the evolution of zinc from lead blast furnace slag. In addition to achieving a zinc recovery of 85% from lead blast furnace slag, a cobalt recovery of 46% was achieved (13). The cobalt was recovered as a ferro-cobalt alloy. During these tests, no attempt was made to focus on cobalt recovery, and, on the basis of other test work relating to cobalt recovery from copper reverberatory slags, higher recoveries seem possible (12). The results obtained from the pilot plant tests are comparable with results predicted by Pyrosim for the option to treat lead blast furnace slag to recover 85% of the zinc in the slag, and discard the alloy at zero cost (see Option 1, shown in Figure 7).

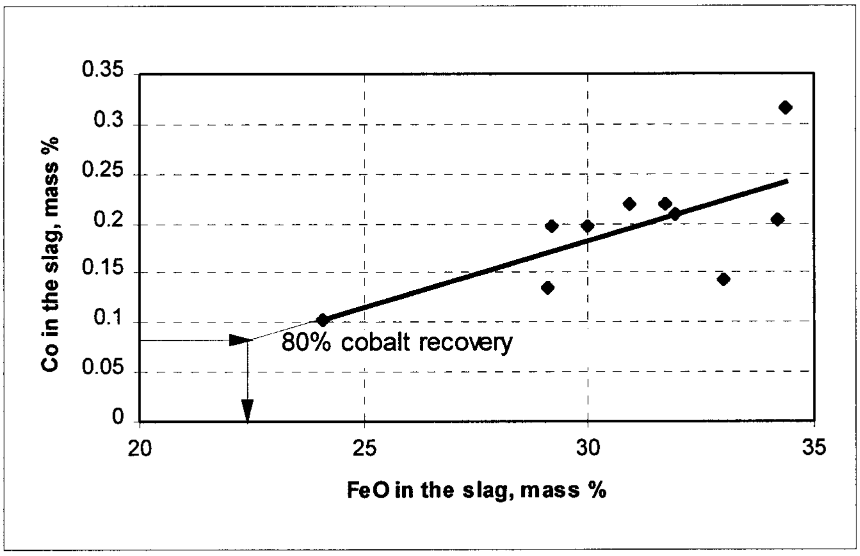


Figure 5 - Cobalt in the slag versus FeO in the slag

A cobalt content in the slag of about 0.08% by mass corresponds to approximately 80% of the cobalt reporting to the alloy.

Temperatures

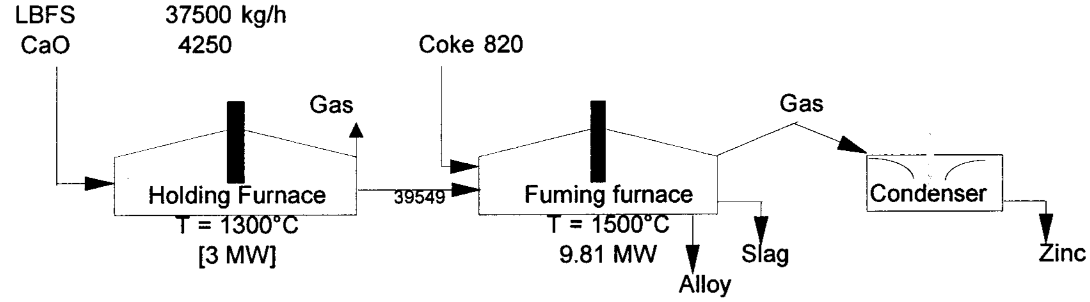
For the purposes of this study, an operating temperature of 1300°C was selected for the holding furnace, in order to maintain a stream of molten slag flowing through an underflow weir to the fuming furnace. The temperature of the fuming furnace was set at

1500°C to ensure sufficiently reducing conditions in this unit. The processes have been evaluated to treat hot blast furnace slag (at 1150°C). This has the advantage of using chemical, rather than electrical, energy to heat the slag from room temperature to 1150°C.

SIMULATION RESULTS

Simulation results for options 1, 2, and 3 are shown in Figures 7, 8, and 9.

Basis: 1000 short tons per day (hot slag at 1150°C), all values on the diagram are given in kg/h



Summary of products:

Slag	Hold. Fce.	Fum. Fce.
kg/h	(39549)	32741
Al_2O_3, mass%	4.58	5.53
CaO	21.18	25.61
Co	0.28	0.20
FeO	29.01	32.90
MgO	4.33	5.24
Pb	2.35	1.03
S	1.79	1.14
SiO_2	21.62	26.13
ZnO	14.48	2.52

Gas	Hold. Fce.	Fum. Fce.
kg/h	459	6568
CO, vol. %	2.10	37.61
CO_2	12.63	10.24
H_2	0.08	1.07
H_2O	1.43	1.19
N_2	3.55	0.26
$Pb_{(g)}$	-	0.59
$PbS_{(g)}$	0.93	1.65
$Zn_{(g)}$	2.28	47.23
SO_2	75.92	-

Overall distribution, %			
	PW zinc	Fe-Co matte	Discard slag
Co	-	41.26	58.74
Fe	-	6.11	93.89
S	-	29.41	41.21
Zn	85.41	-	14.38
Pb	62.74	-	35.68

Alloy & zinc	Furnace	Condenser
kg/h	1063	3939
Zn, mass %	-	98.50
Pb	-	1.50
Co	4.37	-
Cu	17.43	-
Fe	51.32	-
Ni	1.76	-
S	25.09	-

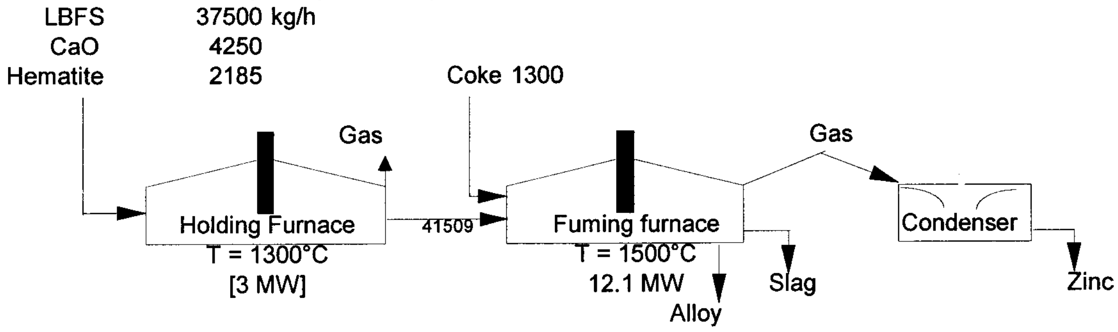
Note: 21.17, and 8.20% of the sulfur in the feed reports to the holding furnace and fuming furnace gases respectively.

Note: Assumed Prime Western grade zinc can be produced in the condenser; all condenser drosses would be recycled.

Note: 1.58% of the incoming lead leaves in the premelter gas, the remainder leaves in the zinc and in the discard slag.

Figure 6 - Option 1: Enviroplas process to recover zinc from lead blast furnace slag; no attempt made to recover cobalt, 2.5% ZnO in discard slag

Basis: 1000 short tons per day (cold slag), all values on the diagram are given in kg/h



Summary of products:

Slag	Hold. Fce.	Fum. Fce.
kg/hr	(41509)	32433
Al_2O_3, mass %	4.36	5.59
CaO	20.18	25.87
Co	0.27	0.07
FeO	32.38	35.02
MgO	4.13	5.29
Pb	2.22	0.49
S	1.45	0.64
SiO_2	20.60	26.38
ZnO	13.79	0.79

Gas	Hold. Fce.	Fum. Fce.
kg/h	685	8110
CO, vol. %	1.29	50.58
CO_2	8.87	5.39
H_2	0.05	1.89
H_2O	1.01	0.82
N_2	2.47	0.30
$Pb_{(g)}$	-	1.35
$PbS_{(g)}$	0.88	0.78
$Zn_{(g)}$	1.93	38.86
SO_2	82.55	-

Overall distribution, %			
	PW zinc	Fe-Co matte	Discard slag
Co	-	80.00	20.00
Fe	-	15.49	84.51
S	-	39.12	22.67
Zn	95.24	-	4.45
Pb	81.06	-	16.78

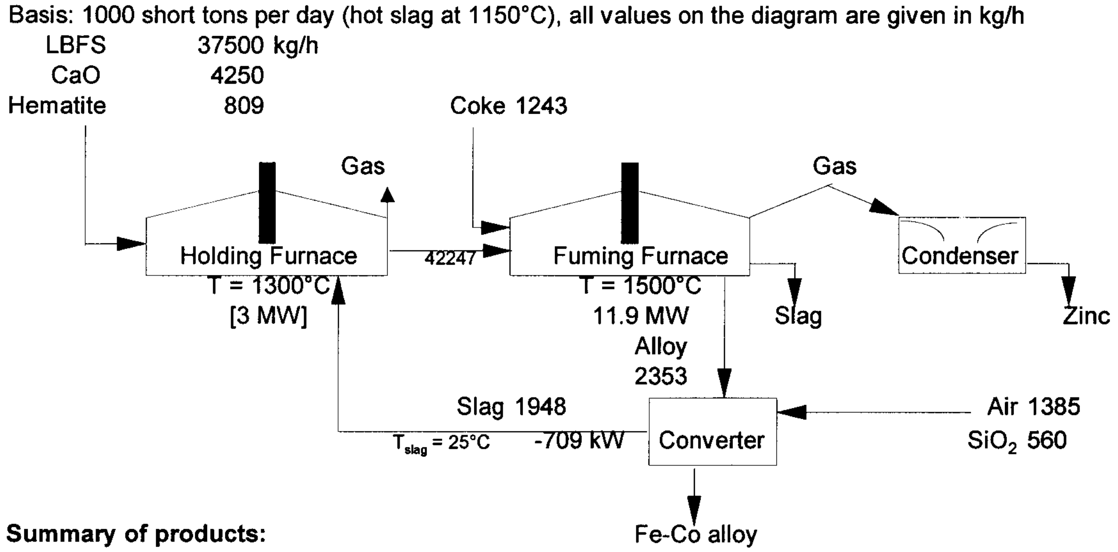
Alloy & zinc	Furnace	Condenser
kg/h	2270	4393
Zn, mass %	-	98.50
Pb	-	1.50
Co	3.97	-
Cu	8.17	-
Fe	71.29	-
Ni	0.83	-
S	15.71	-

Note:33.26, and 4.96% of the sulfur in the feed reports to the holding furnace and fuming furnace gases respectively.

Note: Assumed Prime Western grade zinc can be produced in the condenser; all condenser drosses would be recycled.

Note: 2.16% of the incoming lead leaves in the premelter gas, the remainder leaves in the zinc and in the discard slag.

Figure 7 - Option 2: Enviroplas process to recover zinc from lead blast furnace slag; 80% recovery of cobalt to the alloy/matte, 0.8% ZnO in discard slag,



Summary of products:

Slag	Hold. Fce.	Fum. Fce.	Converter
kg/h	(42247)	33197	1948
Al_2O_3, mass %	4.29	5.46	-
CaO	19.82	25.27	-
Co	0.34	0.09	1.58
FeO	32.07	34.55	69.11
MgO	4.06	5.17	-
Pb	2.20	0.58	-
S	1.61	0.65	-
SiO_2	21.56	27.46	28.75
ZnO	13.56	0.90	-

Gas	Hold. Fce.	Fum. Fce.	Converter
kg/h	518	7944	1091
CO, vol. %	1.86	49.37	-
CO_2	11.32	5.77	-
H_2	0.07	1.80	-
H_2O	1.28	0.85	-
N_2	3.21	0.30	98.78
$Pb_{(g)}$	-	1.20	-
$PbS_{(g)}$	0.86	0.92	-
$Zn_{(g)}$	2.05	39.75	-
SO_2	78.31	-	1.18

Overall distribution, %			
	PW zinc	Fe-Co matte	Discard slag
Co	-	72.45	27.74
Fe	-	7.17	92.83
S	-	44.96	24.12
Zn	94.81	-	5.19
Pb	78.14	-	20.24

Alloy & zinc	Fum. Fce.	Converter	Condenser
kg/h	(2353)	1260	4362
Zn, mass %	-	-	98.50
Pb	-	-	1.50
Co	4.77	6.47	-
Cu	7.88	14.72	-
Fe	68.66	45.20	-
Ni	0.81	1.49	-
S	17.83	32.12	-

Note: 24.4, 1.6, and 5.7% of the sulfur in the feed reports to the holding furnace, converter, and fuming furnace gases respectively.

Note: Assumed Prime Western grade zinc could be produced in the condenser; all condenser drosses would be recycled.

Note: 1.62% of the incoming lead leaves in the holding furnace gas, the remainder leaves in the zinc and in the discard slag.

Figure 8 - Option 3: Enviroplas process to recover zinc from lead blast furnace slag; upgrading the converter alloy to remove iron

Each option has certain perceived advantages and disadvantages which can be resolved only by a detailed costing analysis. For this reason, each process has been costed separately. The following table summarizes the operating details which were used in the costing analysis. Note that a plant availability of just over 85% (7500 hours per year) was assumed for the calculations of annual tonnages.

Table III - Summary of operating details (mass flows in metric tons per annum)

	Option 1	Option 2	Option 3
LBFS, t/a	281250	281250	281250
Coke, t/a	6150	9750	9323
Lime, t/a	31875	31875	31875
Silica, t/a	-	-	4200
Hematite recycle, t/a	-	16388	6068
Fayalite recycle, t/a	-	-	14610
Zinc production, t/a	29543	32948	32715
Slag production, t/a	245558	243248	248978
Alloy production, t/a	7973	17025	9450
%Co	4.37	3.97	6.47
%Cu	17.43	8.17	14.72
%Fe	51.32	71.29	45.20
%Ni	1.76	0.83	1.49
%S	25.09	15.71	32.12
Power rating for holding furnace, MW	3.0	3.0	3.0
Power usage for holding furnace, MW	2.0	2.0	2.0
Power rating for fuming furnace, MW	13.6	16.7	16.5
Power usage for fuming furnace, MW	11.5	14.2	14.0
Total power usage for furnaces, MW	13.5	16.2	16.0
Zn recovery, %	85	95	95
Co recovery, %	0	80	72
Na_2CO_3, t/a	-	5070	4806
Oxygen, t/a	-	5738	5738
Power for hydrometallurgy, MW	-	2.0	2.0
Production of Co as carbonate, t/a	-	676	611
Production of Ni as carbonate, t/a	-	141	141
Production of Cu as sulphate, t/a	-	1391	1391
Production of Na_2SO_4, t/a	-	6794	6441

The distribution between cobalt and iron in the converting stage was identified as an area which needs to be better understood. Experimental data would help to define the partition factors between the cobalt and the fayalite slag. Conservative distribution factors have been assumed in these simulations.

COSTING

The financial performance factors were calculated using the Mintek cash flow model. All numbers are in constant real 1997 US dollars (no difference in escalation between costs and revenue), with equity funding assumed. Financing is assumed to be on a project basis, with equity coming from within the company. The capital cost estimates have an accuracy of around 25 to 35%.

Table IV - Capital cost estimates

	Option 1	Option 2	Option 3
Holding furnace, $m	6.67	6.57	6.65
Fuming furnace, $m	15.95	18.89	18.80
Lead-splash condenser, $m	7.73	8.17	8.12
Converter, $m	-	-	6.05
Atomizer, $m	-	1.52	1.21
Autoclave, $m	-	3.47	2.97
Copper removal, $m	-	2.07	2.06
Product handling, $m	1.17	2.66	2.65
Effluent treatment, $m	-	2.39	1.96
CAPITAL, $m	31.5	45.7	50.5

Table V - Variable and fixed costs

		Option 1	Option 2	Option 3
Electricity costs, $m/a	($50/MWh)	5.08	6.84	6.75
Coke costs, $m/a	($50/t)	0.31	0.49	0.47
Lime costs, $m/a	($50/t)	1.59	1.59	1.59
Silica costs, $m/a	($35/t)	-	-	0.15
Soda ash costs, $m/a	($120/t)	-	0.61	0.58
Oxygen costs, $m/a	($60/t)	-	0.34	0.34
Electrodes, $m/a		1.69	1.88	1.87
Consumables and utilities costs, $m/a		1.63	2.26	2.60
Total Variable costs, $m/a		10.3	14.0	14.4
Total Fixed costs, $m/a		5.9	8.7	9.5

Note: Fixed costs include labor, maintenance, insurance, consumables, overheads.

The metal prices assumed for this study were: 46 cents /lb. for Prime Western grade zinc; 70% of $10/lb for cobalt; and 70% of $3.50/lb for nickel. The cobalt and nickel metal prices were discounted by a factor of 70% to reflect that these products would be sold as a mixed carbonate, instead of as pure metals. For the purposes of this analysis, only the revenue from zinc, cobalt, and nickel was considered. The value of the copper sulfate has been ignored, and the sodium sulfate has been assumed to have no benefit or cost associated with it. The slag is probably able to generate an income of around $10 per metric ton, although this has also been neglected, in order to keep the study on the conservative side. Revenue from the slag is worth about $2.5m per year. It can be seen from Table VI that the production of cobalt contributes about a quarter of the total revenue of Options 2 and 3.

Table VI - Revenue

		Option 1	Option 2	Option 3
Zinc revenue, $m/a	($1020/t or $0.46/lb)	30.1	33.6	33.4
Cobalt revenue, $m/a	(70% of $10/lb)	-	10.4	9.4
Nickel revenue, $m/a	(70% of $3.50/lb)	-	0.8	0.8
REVENUE, $m/a		30.1	44.8	43.6

The internal rate of return (IRR) and net present value (NPV) are calculated after tax, and are based on a tax rate of 35%, and a depreciation of 10% per annum on total fixed capital. Production capacity is assumed to be: 60% in year 1; 85% in year 2; 100% in subsequent years. As shown in Table VII, the IRR calculated over 10 years varied from 18% for Option 3, to 23% for Option 2. Option 2 also has the highest NPV of $31.6 million. Option 2 appears to be the most attractive option to pursue.

Table VII - Operating income statement

	Option 1	Option 2	Option 3
Fixed Capital, $m	31.5	45.7	50.5
Working Capital, $m	2.5	3.7	3.6
Revenue, $m/a	30.1	44.8	43.6
Variable costs, $m/a	10.3	14.0	14.4
Fixed costs, $m/a	5.9	8.7	9.5
Costs (total), $m/a	16.2	22.7	23.9
Gross Profit, $m/a	13.9	22.1	19.7
IRR (10 years)	21%	24%	19%
NPV (10 years, 10%)	$19.4m	$35.5m	$23.4m
ROI	44%	49%	39%

Note: ROI = return on investment before interest and tax

A sensitivity analysis of the IRR of Option 2 is presented in Table VIII. The IRR has been calculated at different levels of capex, opex, volume, and selling price of zinc and cobalt. It is shown that the project is fairly robust in terms of its financial response to fluctuations. It is most sensitive to movements in the selling price of zinc.

Table VIII – IRR sensitivity analysis

% Change	Capital	Manpower	Volume	Zn price	Co price
-40%	39.5	34.6	7.7	-4.5	19.1
-30%	34.4	32.1	12.4	5.6	20.3
-20%	30.3	29.5	16.6	12.7	21.6
-10%	26.9	26.8	20.5	18.7	22.8
0	24.0	24.0	24.0	24.0	24.0
+10%	21.6	21.2	27.4	29.0	25.2
+20%	19.4	18.2	30.6	33.6	26.4
+30%	17.5	15.1	33.7	37.9	27.6
+40%	15.8	11.8	36.7	42.1	28.8

DISCUSSION

The energy requirements for the holding furnace are largely determined by the need to account for a significant energy loss from this unit. The holding furnace is likely to be subjected to large differences in slag level, as it will be fed from ladles. These variations in level will result in large swings in the energy losses through the walls to the atmosphere, and, owing to the large diameter of the furnace, a conservative estimate of 3 MW was made for the transformer specifications; far less energy is required to merely heat the slag from 1150°C to 1300°C.

The use of a separate converting stage to upgrade the cobalt rich alloy results in a lower overall cobalt recovery owing to the fayalite slag recycle which dilutes the cobalt in the fuming furnace slag. The main perceived advantage of a blowing unit of this nature is to reduce the quantity of iron sent for hydrometallurgical processing. The simulations of this unit were conducted without the benefit of consulting experimental data, and should be verified by experimental results.

All drosses and internal recycle streams are assumed to be dealt with as current arisings which are treated on an ongoing basis.

Option 2 appears to be the most attractive option to pursue for the recovery of zinc from lead blast furnace slag. This is mainly attributed to the higher cobalt recovery obtained by operating the furnace under strongly reducing conditions. The flowsheet is also a straight-through operation, with the only recycle being the hematite residue fed to the holding furnace.

CONCLUSIONS

1. Option 2 (reducing conditions in the fuming furnace; no converter) appears to be the most attractive option to pursue for the recovery of zinc from lead blast furnace slag where cobalt is recovered as a by-product.

2. Cobalt recovery significantly improves the economic viability of the process (at the assumed metal prices used in this study)
3. The distribution of cobalt between slag and alloy in the upgrading stage has been identified as an area which should be investigated further, in order to confirm the viability of this unit.

ACKNOWLEDGEMENTS

Acknowledgments are due to: the Doe Run Company for permission to publish the results of tests conducted at Mintek (which were documented by N.C. Machingawuta and M. van Zyl at Mintek); Scott Miller of the Metallurgy Department at UMR for helpful discussions; M.J. Freeman of Mintek's Techno-Economics Division for performing the costing; M.J. Dry of Mintek's Hydrometallurgy Division for invaluable assistance with the selection and simulation of a hydrometallurgical processing route. This paper is published by permission of Mintek.

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