The application of oxygen in a copper smelter

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SYNOPSIS

This paper describes how the problem of increasing the capacity of the Rokana copper smelter was handled, and how oxygen was found to be technically and economically the best solution. The first phase of its implementation is outlined.

SAMEVATTING

Hierdie verhandeling beskryf hoe die probleem om die vermoë van die Rokana-kopersmeltery te verhoog, gehanteer is en hoe daar gevind is dat suurstof tegnies en ekonomies die beste oplossing is. Die eerste fase van die toepassing daarvan word in breë trekke beskryf.

INTRODUCTION

The Zambian Copperbelt is the third-largest copper producer in the world. Of the two copper companies operating in Zambia, Nchanga Consolidated Copper Mines (N.C.C.M.) is the larger, and was formed in 1969 from a merger of the Anglo American Corporation (Central Africa) operations at Chingola, Rokana, and Konkola.

Mixed oxide-sulphide orebodies are mined at Chingola and Konkola, the oxide concentrates being processed in the Chingola leach electrowinning plant. Leach cathodes are refined in the Rokana Refinery.

The Rokana Division has underground mining operations that yield sulphide ores containing both copper and cobalt at economic grades. In the concentration process, the predominantly cobalt-containing concentrate that is produced is treated for the recovery of cobalt. There is also a small open-pit operation at Rokana from which a refractory oxide ore is mined for treatment by the segregation process in the Torco plant.

The Group's predominantly sulphide smelter-grade concentrates from all N.C.C.M's operations are treated at Rokana in what is probably the world's largest copper-producing Rokana smelter.

The Group's current annual production target is 420 000 tons of copper, of which 105 000 will be leach cathodes, the remaining 315 000 tons ex concentrates treatment in the Rokana smelter.

The production of electrolytic cobalt amounts to more than 2000 tons per annum.

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SMELTER PROCESS

Only Rokana produces a wholly sulphide concentrate, and this contains bornite with some chalcopyrite and carrollite (a cobalt-bearing mineral that cannot be totally depressed into the cobalt concentrate).

The greatest concentrate input arises from Chingola. Here the sulphide minerals are predominantly chalcocite and bornite in the ratio of 2:1, but the ratio of sulphide copper to oxide is variable. Depending on the mining plan and the availability of treatment capacities, oxide concentrates containing up to 40 per cent total copper may be produced for smelting. In consequence, the charge to the reverberatory furnaces tends to be sulphur-deficient, and pyrite additions are made to ensure that matte grades are controlled. Mattes having copper contents in excess of 64 per cent are avoided since these result in cold converter operation, which leads to the formation of excessive magnetite and spinel. Rokana has the highest-known matte-grade practice of all copper smelters.

In a comprehensive charge-handling and preparation plant, some 80 000 tons of concentrates per month, dried to less than 8 per cent moisture, along with 6000 tons of limerock and 16 000 tons of aisle reverts and other secondary materials, are blended into a charge for the five, coal-fired reverberatory furnaces.

The central coal-pulverizing plant within this complex has six British Rema grinding units, each able to dry and pulverize coal at the rate of 8 tons per hour to a grind of 90 per cent minus 200 mesh.

Each reverberatory furnace has two waste-heat boilers with the facility to operate on only one but at reduced firing rates (see Fig. 1). The boilers are of the tri-drum vertical bent-tube type with a heating surface of 972 m^2. They were designed for a continual evaporation rate of 36 300 kg per hour at a working pressure of 2,413 MPa. Final steam temperature is 399 °C and feed-water temperature 93 °C. Each boiler has its own tubular air heater, one forced-draught fan, and two induced-draught fans.

Waste gases leaving the boiler are treated in four plate-and-wire type electrostatic precipitators before being vented to the atmosphere through a 76 m high stack.

Furnace slag is skimmed into pots conveyed by trains on a single line that handles all slag and ash traffic from the furnaces and associated boilers. Matte is tapped into two pot bogeys, which are winched into the converter aisle. Five cranes, each having a lifting capacity of 41 tons, service six modified Pierce-Smith converters (9,2 by 4 m) and four casting furnaces, all of which are equipped for coal firing.

Part of the waste converter gas is collected by snorkel devices in the converter mouths at an average concentration of 7,2 per cent sulphur dioxide, and is treated in two contact sulphuric acid plants, each having a nominal capacity of 150 tons of 100 per cent acid per day.

Both 250 kg blister cakes (99,5 per cent Cu) and 285 kg anodes (99,8 per cent Cu) are cast from the casting furnaces on 22-mould Walker wheels.

The oxygen plant, designed and erected by Air Liquide, has a production capacity of 550 tons of oxygen per day at a purity of 96 per cent.

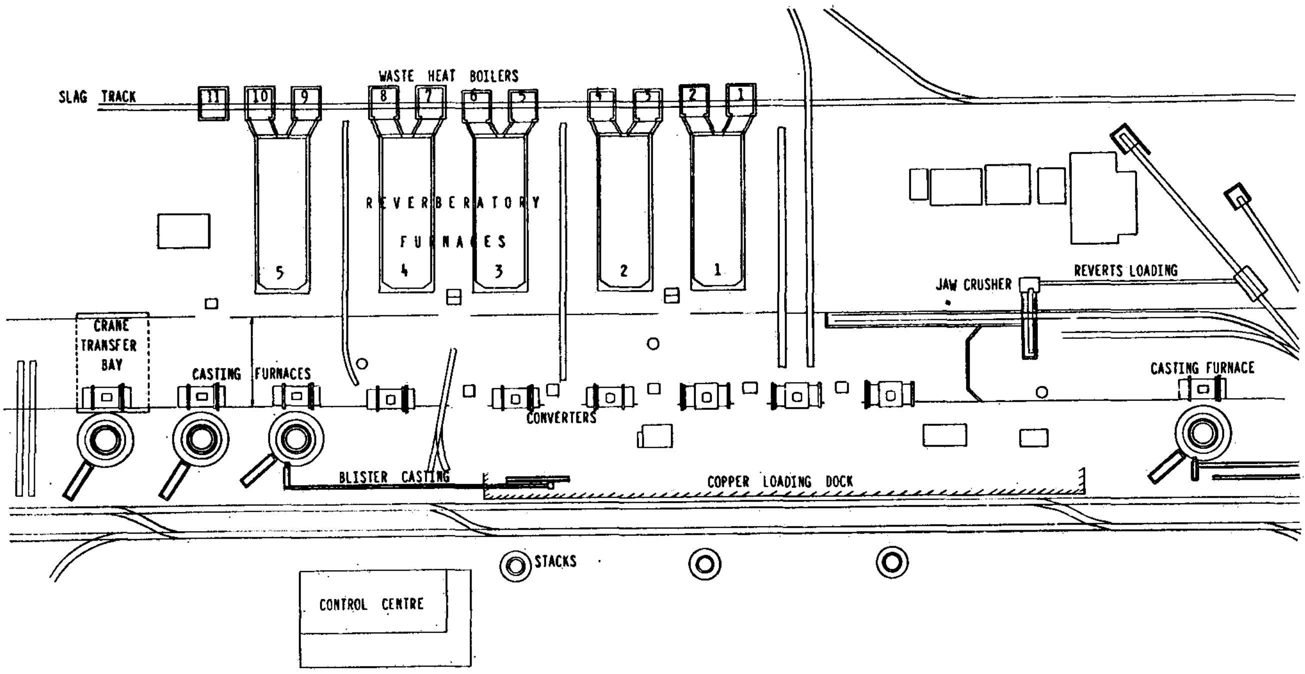

Fig. 1—General layout of the Rokana smelter

The smelter complement is approximately 40 expatriate and 960 local employees.

THE PROBLEM

It was in 1966 that preliminary studies were started into ways and means of increasing smelter output. From earlier comprehensive investigations carried out in the 1950s and the modifications subsequently effected, it was believed that the existing units were being operated at close to their maximum potential[1].

The reason for the studies was that long-range production forecasts had shown that smelting requirements would be well in excess of the known smelter capacity of 300 000, reaching 400 000 tons of copper per annum. A more immediate reason was the production loss resulting from the use of an inferior local coal. While the Zambian authorities had made plans for the systematic development of their own coalfield at Maamba, the declaration of UDI necessitated more urgent measures. As a stop gap, a low-grade, high-ash, bituminous coal was opencasted at Nkandabwe. Smelter fuel assays are shown in Table I.

Not only did the local fuel have a higher ash content than the Wankie coal, which was the industry's standard, but it was harder to grind and the capacity of the pulverizing plant was so reduced that the reverberatory furnaces could not be operated at their rated capacities.

Much worse were the effects of the high silica content of the ash, which readily attacked the basic refractories in the furnaces and reduced campaign lives from a normal three years to less than one year.

In 1968, a multiple regression analysis covering the seven years of operation between 1961 and 1968 to evaluate possible factors affecting refractory consumption revealed that 78 per cent of the variability was due to the ash content of the fuel.

With only a 30 per cent replacement of Wankie fuel by Nkandabwe, the potential monthly copper production was reduced from 25 000 tons to 21 000 tons per month. To maintain an annual copper-production rate of 300 000 tons, it was necessary that all five reverberatory furnaces operated as near continuously as possible on Wankie fuel and that campaigns of more than two years were achieved.

At that time there were five Pierce-Smith converters, and, since one was always out of commission for major overhaul, the operation of the remaining four had to be carefully staggered to ensure maximum availability.

While reverberatory-furnace and

TABLE I
SMELTER FUELS

Type	Coal analysis						Ash analysis				
	Calorific value MJ/kg	% Ash	% Fixed carbon	% Volatiles	% S	% H_2O	% SiO_2	% Al_2O_3	% CaO	% MgO	% FeO
Wankie	30,47	12,3	61,1	26,6	1,7	11,3	38,0	23,7	10,0	0,7	13,4
Nkandabwe	23,26	27,0	46,0	20,0	0,8	2,1'	62,0	22,5	2,7	0,6	5,8
Maamba	27,82	16,4	64,7	18,9	1,0	3,9	—	—	—	—	—

converting capacities were reasonably well matched, the crucial operation was undoubtedly traffic in the converter aisle. A comprehensive study had been made of converter-aisle operations with particular reference to crane utilization. A team of observers studied each operation by means of multiple time activity charts, which were used to assess potential aisle capacity. Proposals on the re-allocation of crane duties and spheres of operation were submitted and introduced.

Using basic times, crane demands and services were determined for matte grades ranging from 58 to 66 per cent copper. The conclusions reached were that two-crane operation was insufficient for a production level of more than 70 ladles of matte per day, or 18 800 tons of copper per month on a matte grade of 58 per cent and 21 400 tons of copper per month on a 66 per cent copper matte. Three-crane operation at 85 per cent availability would just meet a production level of 84 ladles of matte per day. The introduction of a crane-transfer bay increased availability by 11 per cent, which meant that maximum copper production was between 26 300 and 30 000 tons per month, according to the matte grade.

CONSIDERATIONS

It seemed there were four possi. bilities in obtaining a higher pro duction: an increase in the number of installed units, conversion to electric smelting, conversion to flash smelting, or the use of a continuous smelting process.

The first had very few attractions save that it was a road N.C.C.M. knew. To meet the target requirements, an additional two reverberatory furnaces and two converters would have been necessary. These extensions would be capital-intensive, and since one of the main problems was aisle traffic, the situation would be aggravated by the extension of the 275 m-long aisle.

The proposal to convert to electric smelting had many attractions, particularly since there was a coal problem and this type of smelting is ideally suited to that sort of charge where the sulphur balance is so fine. However, the projected operating costs ruled this out.

Flash smelting was considered, but, because the charge was far from autogenous, a considerable amount of auxiliary firing would have been necessary. This type of process is the most capital-intensive of all, and oxygen enrichment would undoubtedly have been required if the targets were to be achieved on a single unit.

The last possibility could not be seriously considered since none of the new processes, however attractive technically, had been proven commercially, and the location was considered unsuitable for a process under development.

It seemed that there were two requirements which, if met, would ideally solve the problem. One was that the capacity of existing units should be increased, and, two, that aisle traffic should not increase.

For these reasons, an oxygen facility that would allow the use of oxygen-enriched air for combustion in the reverberatory furnaces and an oxygen-enriched blast on the converters was thought to be the answer. The latter was probably the more critical since, without the smelting of concentrates in the converters, the handling problem in the converter aisle would still limit smelter output.

There were several other attractions, primarily that capital investment was low, that the scheme could be implemented in a comparatively short space of time, and, equally important, it was a technique that had been used successfully by other copper producers.

It therefore seemed logical to set about learning more about oxygen utilization. Visits were made to oxygen-plant manufacturers, iron and steelmakers, and non-ferrous producers who had experience either with oxygen enrichment or oxygen jetting. Of particular interest and direct application to the situation at Rokana was a visit to the Copper Cliff works of the International Nickel Company of Canada Limited (Inco), which uses oxygen enrichment on reverberatory furnaces and converters for the treatment of both copper and nickel concentrates and mattes[2].

These visits convinced N.C.C.M. that it should go ahead with its plans for oxygen enrichment, and detailed feasibility studies were then undertaken.

STUDIES

Reverberatory Furnaces

Rather than calculate the probable total effects of oxygen enrichment on furnace performance to include such factors as higher flame temperatures and increased heat transfer, it was thought prudent to calculate only those benefits arising from the heat that would otherwise have been carried out of the furnace in the now-excluded nitrogen.

The relevant results are given in Table II.

The common denominator in all three cases is a coal-firing rate of 127 tons of coal per day, which was the maximum coal-firing rate that could be maintained on a furnace with two operating boilers. This is equivalent to a maximum combustion rate of 15,2 kg of coal per cubic metre of combustion volume per hour. Combustion volume here relates only to that of the charging zone, and not to the total furnace atmosphere.

On the assumption that all three cases are physically practicable, it

TABLE II
OXYGEN ENRICHMENT ON REVERBERATORY FURNACES

Case	1	2	3
Oxygen in combustion blast, mass %	23,2	30	40
Coal firing rate, tons/day	127	127	127
Hot air supplied/coal, mass ratio	9,0	—	—
Leakage air/coal, mass ratio	1,5	—	—
Total blast/coal, mass ratio	10,5	8,5	6,7
Furnace waste gas/coal, mass ratio	12,1	10,9	8,5
Hot air temperature, °C	188	188	188
Leakage air temperature, °C	32	32	32
Dry charge, tons/day	622	809	971
Moisture in charge, %	8,2	8,2	8,2
Oxygen supplied, tons/day	—	78	145
Production increase, %	—	30	56

can be calculated that 2 tons of oxygen is equivalent to 1 ton of coal in terms of charge smelted.

The most interesting statistic is the mass ratio of waste gas to coal, which decreases markedly with oxygen enrichment. Since the limitations of the existing furnaces are primarily determined by their gas-handling capacity, the significant increased production potential to be gained from oxygen enrichment can readily be recognized.

A typical heat-balance summary for a reverberatory furnace is given in Table III. It will be noted that, of the 66,5 per cent heat loss to exit gases, nitrogen accounts for no less than 37,1 per cent.

Converters

Table IV gives data that are typical of practice where matte grades are high.

In the converting cycle shown in Table IV, flux ore would be added to all slag blows, and revert material only during the second and third slag blows, when operating temperatures were sufficiently high. Converter blowing time is 71 per cent of the total cycle time.

Table V shows the calculated benefits of oxygen enrichment on converting practice.

Cases 2 and 3 are those where oxygen enrichment is practised. It will be noted that the mass ratios of concentrates to oxygen are 3,4 and 3,3. This ratio would obviously be determined by the type of concentrates charged to the converter.

Of particular interest is the changing sulphur dioxide content of the mouth gas as oxygen concentration is increased in the blast. As the nitrogen content decreases, so the sulphur dioxide increases, the ratio being almost 1:1. This has tremendous implications since production of copper in the Chingola leaching plant can be directly related to acid availability, and consequently there is an incentive to maximize sulphuric acid production at Rokana.

In the reverberatory furnaces, only some 15 per cent of the sulphur entering with the concentrates escapes to the waste gas, which is primarily combustion gas from the burning of coal. Here the sulphur

TABLE V
OXYGEN ENRICHMENT ON COPPER CONVERTERS

Basis

1. All cycles calculated to produce 82 tons of new copper.
2. Concentrates charged were assumed to be high-grade Chingola sulphide concentrates, which assayed:

Cu	55,14%	SiO_2	19,30%
Fe	4,23%	Al_2O_3	2,50%
S	14,51%	MgO	0,60%

Case	1	2	3
Oxygen in blast, % vol.	20,9	30,0	40,0
Cycle data			
Matte charged, tons	132	91	74
Concentrates treated, tons	—	46	66
Concentrate to matte, mass ratio	—	0,51	0,89
Slag to matte, mass ratio	0,394	0,574	0,711
Oxygen consumed, tons	—	13,6	20,0
Mouth gas, % SO_2	17,0	25,5	35,0
Per converter day			
Cycles per day	3,20	4,41	5,07
Matte charged, tons	422	401	375
Concentrates treated, tons	—	203	335
Slag skimmed, tons	166	230	267
New copper, tons	255	352	406
Oxygen consumed, tons	—	60	102
Concentrate to oxygen, mass ratio	—	3,4	3,3
Production increase, %	—	38	59

TABLE III
REVERBERATORY-FURNACE HEAT BALANCE

Heat input	MW	%	Heat output		MW		%
Coal	42 735	94,9	Smelting dry charge		12 298		27,3
Combustion air	2 293	5,1	Heat losses		2 772		6,2
			Exit gases		29 958		66,5
			—nitrogen	16 724		37,1	
			—carbon dioxide	6 559		14,6	
			—oxygen	229		0,5	
			—water vapour	6 446		14,3	
Total	45 028	100,0			45,028		100,0

TABLE IV
TYPICAL CONVERTING CYCLE

Basis

1. Matte charged 132 tons
2. Matte assay Cu 62%, Fe 12,5%, S 22,3%
3. Blowing rate 2,5 minutes per ton of matte
4. Copper production 103 tons
5. Slag assay Cu 5,6%, SiO_2 24,0%, Fe_3O_4 14,2%
6. Flux ore Cu 4,00%, SiO_2 71,0%

	Minutes *Blowing*	Minutes *Outage*
Discharging copper and charging 5 ladles of matte		55
1st slag blow	30	
—skim and charge 2 ladles of matte		10
2nd slag blow	30	
—skim and charge 1 ladle of matte		10
3rd slag blow	30	
—skim		10
4th slag blow	20	
—clean off skim		20
Sub-total	110	105
White-metal stage		
—while adding 20 tons of scrap	120	25
Final blow	90	
Sub-total	210	25
Total cycle time	320	130

dioxide contents are very low, normally less than 1 per cent, and oxygen enrichment does not raise the concentration to any economic level.

Table VI shows that the heat carried out of the converting system by nitrogen is 45,5 per cent, an even greater percentage of the total than in the reverberatory furnace.

ECONOMICS

As a demonstration in practical terms of oxygen enrichment, the 1967 production forecasts were used to show smelter practice with and without oxygen. The results are given in Table VII.

The salient features in this exercise are outlined below.

The capital cost of the oxygen plant was R2,4 million, and this, with the distribution, control system, and instrumentation, made a total cost of R3,1 million.

Savings in operating costs are tremendous. From Table VII it will be noted that the use of 436 tons of oxygen results in a saving of 274 tons of coal each day. With coal costing R11,5 per ton and a forecast oxygen cost of R3,5 per ton, there was an effective saving of some R1600 per day. Saving of the basic monthly charges for two reverberatory furnaces was R20 000, and, with a further saving of one converter operation, this meant a total saving of some R36 000 a month on furnaces.

One of the larger debits—R22 500 per month—was the loss of the power credit from steam generation in the waste-heat boiler. Overall, it seemed that a saving of not less than R70 000 per month could be expected while at the same time existing furnacing capacity would be increased by some 30 per cent.

Reference was made earlier to the problem of co-ordinating converter-aisle operations. With oxygen enrichment, high-grade concentrates can be charged direct to the converters and some crane transfers of matte are saved, although there is a slight increase in the amount of converter slag to be returned to the reverberatory furnaces.

Based on the 1967 production forecasts, calculations showed that the aisle traffic of matte and converter slag would be reduced by some 25 per cent when oxygen-blowing was implemented. Matte transport was reduced from 1266 to 820 tons per day, and converter slag increased from 498 to 502 tons per day.

IMPLEMENTATION

While the calculations, based on the 1967 production forecasts, showed an oxygen requirement of 436 tons per day, a decision was taken to install a tonnage oxygen plant to produce 550 tons per day to allow for the planned increase in production. The plant was designed to produce only gaseous oxygen at a purity of 96 per cent and a pressure of 158 kPa. There is no storage capacity for either gaseous or liquid oxygen, since N.C.C.M. has air-enriched applications all of which can operate without oxygen in the event of an oxygen-plant breakdown but at lower efficiencies.

Since the full advantage of tonnage plants to produce cheap oxygen can be realized only if they operate at a steady rate, the decision was made to allow any surplus oxygen to blow to the atmosphere during those periods of unavoidable restricted demand.

At the converter air main, oxygen pressure is 117 kPa, which is a safe 14 kPa higher than the blast air.

Oxygen for lancing on the reverberatory furnaces is supplied through a small compressor at a pressure of 827 kPa.

The instrumentation logic was considered at great length. The objective was to provide an effective overall control of oxygen distribution so that the total oxygen produced was utilized to maximum advantage. To achieve this, it was necessary to allocate the oxygen on a basis of management priorities, all surplus oxygen that became available as a result of abnormal or planned operational conditions being redistributed on a priority basis.

While maintaining overall control from an oxygen-distribution centre, each operating unit has full supervisory facilities. Special precautions

TABLE VI
CONVERTER HEAT BALANCE

Heat input	kJ per lb of new matte	%	Heat output	kJ per lb of new matte	%
Matte	897,67	25,7	Blister	595,39	17,0
Blast	98,17	2,8	Slag	533,81	15,3
Reactions	2502,78	71,5	Heat losses	105,60	3,0
			Mouth gas		
			—SO_2	568,69	16,3
			—Nitrogen	1593,30	45,5
			—Excess air	101,83	2,9
	3498,62	100,0		3498,62	100,0

TABLE VII
PREDICTED SMELTER OPERATION WITH AND WITHOUT AN OXYGEN FACILITY

Basis 1967 production forecast 2122 tons of concentrates per day
785 tons of molten copper per day

Practice	Normal	Oxygen
Reverberatory furnaces operated	4,2	2
Coal burnt, tons/day	528	254
Oxygen consumed, tons/day	—	252
Oxygen in combustion blast, mass %	—	37
Concentrates smelted, tons/day	2122	1516
Dry charge, tons/day	2588	1850
Converters operated	3,1	2
Oxygen consumed, tons/day	—	184
Oxygen in blast, % vol.	—	37,6
Concentrates treated, tons/day	—	606
Calculated mouth gas, % SO_2	17,0	32,7
TOTAL COAL BURNT, tons/day	528	254
TOTAL OXYGEN CONSUMED, tons/day	—	436

were taken to ensure that all practicable safety measures were provided to guard against instrumentation failures on all plant conditions and possible mal-operation. Electronic instrumentation was specified throughout and was carefully chosen to ensure that standard proven units were used wherever possible. This simplified the problems of maintenance, reducing the quantity and variety of essential spares to an absolute minimum.

All basic distribution and control systems were subject to identical responses in time lags, and this, with pressure and temperature compensation for variable pipeline conditions, gave a highly stable and reliable overall control.

Full advantage was taken of the latest solid-state techniques having in mind a future data-collection system that could be located in the oxygen-distribution centre with the minimum of additional instrumentation and expense. The ultimate objective, of course, was computer control.

The initial work with oxygen enrichment was carried out on the reverberatory furnaces and, more recently, on the converters. A paper by Eastwood, Thixton, and Young[3] outlines the early part of the work on the reverberatory furnaces.

DEVELOPMENT OF OXYGEN TECHNIQUES

Reverberatory-furnace Combustion

While most of the components in the combustion system are common to reverberatory-furnace practice elsewhere, the burner design was developed locally. This consists of a primary port into which only sufficient air is injected with the pulverized fuel to both transport the fuel and to satisfy its volatile content (Fig. 2). Secondary ports located above these burners introduce sufficient additional air to ensure an oxygen content of 0,5 to 1,0 per cent in the furnace exit gas.

This physical layout was believed to have several advantages. Having only sufficient air for the combustion of volatiles in the primary burner ensures a maximum velocity of flame propagation. For the burning of the carbon particles, which is a much slower process, the secondary and leakage air is progressively mixed into the flame and is supplied as required to complete the combustion process. This layer of colder, secondary air above the primary burners to some extent prevents contact between the refractory and the hottest part of the flame, and generally reduces refractory abrasion.

The introduction of the Wankie-Nkandabwe coal mixture had already meant burner modifications since the volatile content of the furnace fuel had been reduced. The diameters of the primary burners were reduced to maintain minimum coal-transport velocities and optimize the air:coal ratios for good ignition.

Introduction of oxygen meant a further change of combustion conditions. As indicated by Inco's experience, all the oxygen was introduced through special ports under

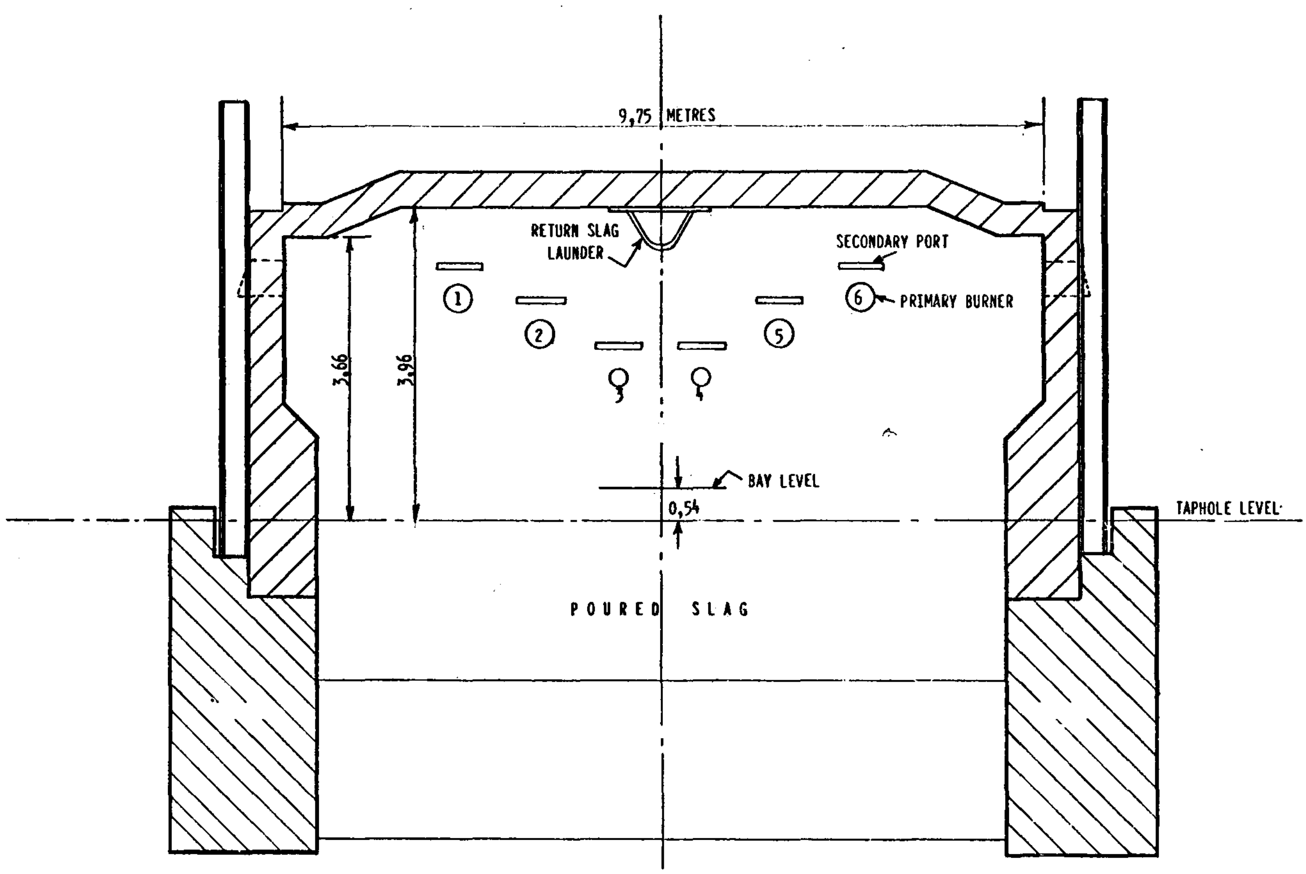

Fig. 2—Section of the burner wall of the reverberatory furnace

the two centre burners. To meet the required analysis of exit gas, it was therefore necessary to reduce secondary air through the top ports. The situation was ultimately reached where no secondary air was being added, thus giving a very inflexible control and imposing a top limit to the level of oxygen enrichment.

Before the introduction of oxygen, all the furnaces were equipped with four outer burners (380 mm diameter) and two inner burners (300 mm diameter). Because of the change of fuel type and the introduction of oxygen enrichment, it was found necessary to change to outer burners of 300 mm diameter and inner burners of 250 mm diameter. With these burners, an impasse was reached when oxygen usage was 95 tons per day. To overcome this barrier, burner diameters were further reduced by 50 mm, which increased oxygen capacity to 110 tons per day.

Although it was hoped to retain certain features of the traditional system, a re-think on burner design has been necessary to optimize combustion with oxygen enrichment. Composite burners, where oxygen or part of it is introduced with the air to the primary burner, are being investigated. A line of further enquiry involves a reduction in the number of burners, with possibly oxygen enrichment only on the inner two, which could maximize smelting rates and minimize refractory wear.

The position of the oxygen ports is a most important factor. Many modifications have been made, and the best results to date have been obtained with the ports located midway between the centre points of burners 2 and 3 and burners 4 and 5. Although different sizes have been tried, ports of 100 mm diameter are favoured.

Bath Conditions

Little difficulty was experienced in the initial stages of commissioning the oxygen supply to the furnaces. However, first observations were that matte and slag temperatures had risen, and that furnace hearths softened at an alarming rate. All indications were that combustion efficiencies had diminished, and heat transfer in the smelting area had been sacrificed to heat transfer in the skimming area (Fig. 3). Exit-gas temperatures, as waste-heat boiler steaming rates, had increased.

There were charge-bank collapses because of excessive heat at the charge-pile toes, the resulting explosions blowing out the slot walls adjacent to the boiler. As experience with oxygen was gained, so normality returned to the operations.

Since the introduction of oxygen, normal slag temperatures have increased to approximately 1270 °C from 1245 °C, and matte temperatures to 1160 °C from 1140 °C, while the hearths are generally some 400 mm above matte-taphole levels.

Uptake-gas temperatures and waste-heat boiler steaming rates are unchanged on former levels, although the coal-firing rates have been increased to 150 tons with an oxygen usage of 76 tons per furnace day. At higher oxygen values, exit-gas temperatures tend to increase with the present burner configuration.

Since the initial difficulties, the rate of refractory wear has not measurably increased with the use of oxygen enrichment; current furnace campaigns are longer than 30 months.

There have been significant changes in the pattern of slag losses since the introduction of oxygen. A normal slag at Rokana might be expected to contain 0,90 to 1,00 per cent copper, but, when oxygen was first introduced, this figure rose to 1,25 per cent on average. When the oxygen-port position was changed from below the centre burners to between burners 2 and 3 and burners 4 and 5, the average copper content in slags decreased to 1,09 per cent.

It had been postulated that, with oxygen enrichment, the increased oxygen partial pressure in the furnace would increase copper values in the slags, but the reasons have not yet been clearly defined.

Since slag temperatures have increased, it has been found possible to reduce the additions of limerock and still maintain fluid slags.

Charging Cycle

When charge pipes are opened and cold wet charge is introduced into the furnace, cold air is inadvertently brought into the system. This upsets combustion since the radiating surface temperature is reduced and the

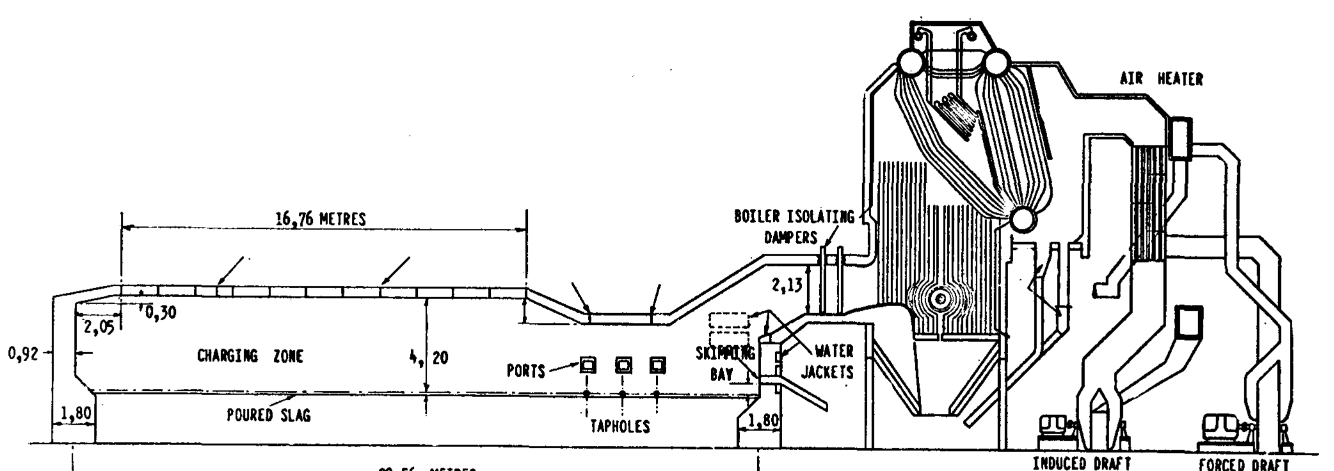

Fig. 3—Longitudinal section of the reverberatory furnace and boilers

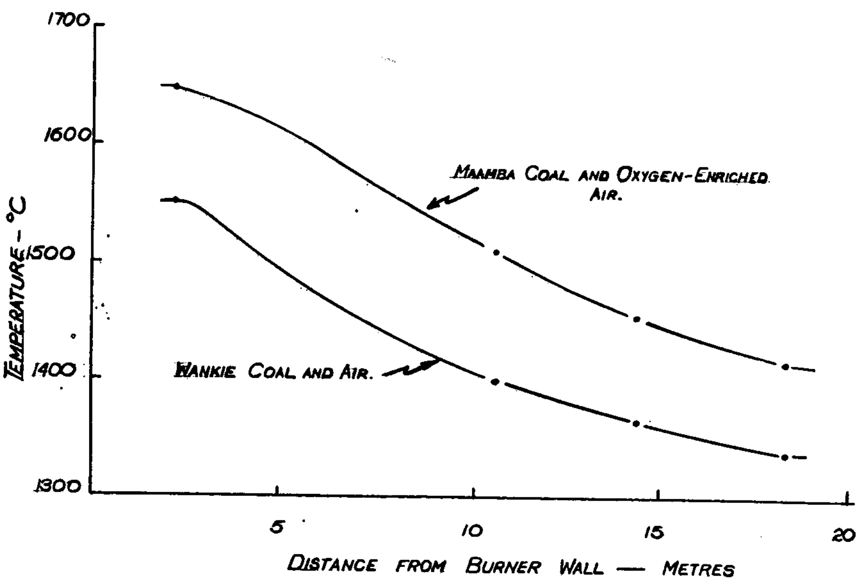

Fig. 4—Flame-temperature profiles for a reverberatory furnace

consequent fall off in flame temperature means less heat radiation to the charge. On average, flame temperatures are reduced by 200 °C and take some 30 minutes to recover.

While it is obviously necessary to maintain the charge-bank profiles to ensure maximum surface area to absorb heat, too frequent charging so interrupts the steady combustion state that smelting rates are seriously affected. However, if there is too infrequent charging, the charge-bank areas remain small and the full smelting potential will not be realized despite high flame temperatures. Not only does the smelting rate fall, but exit-gas temperatures increase and refractory is exposed to excessive wear.

The Rokana reverberatory furnaces originally had 12 charge ports along either side of the charging zone at 1,50 m centres, but now have 22 at 0,90 m centres. Shorter and longer charging cycles have been investigated, but the 3-hour cycle is the present standard. Ninety minutes is allowed for charging, if required, and ninety minutes for skimming and boiler cleaning.

The effect of oxygen enrichment on flame temperatures is shown in Fig. 4.

A typical example of present reverberatory-furnace performance is the smelting of 800 tons of wet charge, 9,4 per cent moisture, when firing 129 tons of Maamba coal with 70 tons of 100 per cent oxygen.

Converters

Development work here has largely been concerned with the use of oxygen during copper blows since finality has not been reached on the method and systems to be used for the introduction of concentrates into the converters.

To avoid overfilling of the converter when oxygen is used on copper blows, it is necessary to vary the amount of matte charged according to the enrichment level used during the copper blow; the greater the enrichment level, the greater the amount of scrap consumed. To maintain safe operating temperatures, the enrichment used is determined solely by the availability of scrap. Since scrap is not always readily available, the oxygen enrichment is programmed to be turned off when bath temperatures reach 1220 °C. With an oxygen-enrichment level of 24 to 25 per cent, some 40 tons of scrap can be charged during the copper blow and, with judicious additions of scrap, through to the blister stage without the converter becoming overheated. Oxygen has never been used through to the anode copper stage because of the danger of increased refractory wear.

The utilization of oxygen enrichment during slag blows posed several problems. Revert material must be kept readily available to ensure that converter operating temperatures do not get out of hand. Slags formed in the early test cycles were found to contain excessive amounts of magnetite. It was thought that, because of the shorter blowing times, the silica had not been completely fluxed, or that, starting with a cold converter, more of the iron was oxidized to magnetite because of the higher oxygen content of the blast. The problem was largely overcome when, on the first slag blow, the converter was blown conventionally up to a bath temperature of 1200 °C and thereafter silica feed and oxygen were switched on simultaneously.

The use of oxygen-enriched air increases the converter capacity for both revert materials and scrap, but initial tests have clearly demonstrated that the treatment of both these materials in the same converting cycle has limitations. When oxygen is used on the copper blow, the amount of matte charged is less than normal, and the subsequent slag-blow times are shorter and hence the amount of reverts that can be treated is minimal. An increase in the enrichment level to produce more heat for the melting of reverts merely shortens the slag blows even further.

Other tests have shown that, if oxygen is used only on the slag blows, the converter can be charged normally. If this practice is followed, oxygen enrichment cannot be used on the whole of the copper blow without the danger of overfilling.

The use of oxygen must therefore depend on the availability of revert material and scrap copper.

It was found that a 1 per cent increase in enrichment level gave an approximate 1 per cent increase in sulphur dioxide content. The maximum gas strength obtained was 22,2 per cent sulphur dioxide at an oxygen-enrichment level of 24 per cent, which confirmed the theoretical calculations.

Experience to date indicates that there is little or no increase in refractory wear at the enrichment levels used.

There have been a few difficulties, not previously experienced, with oxygen enrichment. Individual converter behaviour can still vary occasionally for apparently inex-

TABLE VIII
CONVERTER OXYGEN PRACTICE

	Without oxygen	24% oxygen
Slag blow, minutes	110	46,5
Slag blow time per ladle of matte, minutes	13,7	8,0
Copper blow, minutes	210	128
Oxygen used per slag blow, tons	—	0,794
Oxygen used per copper blow, tons	—	2,114
Blowing time per ton of copper produced, minutes	3,11	1,91

plicable reasons, but no more so than before oxygen practice. Converter performance with its many variables (tuyère angle, mouth position relative to tuyère line, refractory standards, blowing depths) will seldom yield to simple analysis.

In a particular converter campaign having a total of 178 cycles, 120 of these were carried out with oxygen enrichment: 74 used oxygen during both slag and copper blows, and 46 only during copper blows. A total of 300 tons of oxygen was consumed.

Table VIII gives comparative data for an oxygen-enrichment level of 24 per cent and a normal pre-oxygen operation.

It is noteworthy that, even at such a low level of enrichment as 24 per cent, blowing times for a full converting cycle with oxygen are reduced to 55 per cent of their former level and overall cycle times to probably 70 per cent.

A very comprehensive investigation programme is under way to determine the optimum fluxing techniques for various enrichment levels. Studies of all aspects of the addition of revert material and of the operational and economic aspects of charging cold materials, such as cement copper brickettes and pelletized concentrates, are being made.

SUMMARY

The use of tonnage oxygen at the Rokana smelter has proved to be the most expedient and cheapest way not only to maintain but to increase the production of molten copper, despite the disastrous consequences of having to use an inferior coal, which severely limited the coal-pulverizing capacity and hence the production of the reverberatory furnaces.

Since oxygen implementation, the refractory nature of the charge has worsened and higher copper-production levels have been achieved only by the smelting of some oxide concentrates of inferior grade, which would otherwise have been stockpiled as untreatable.

An even greater increase in smelting potential with higher efficiencies and reduced costs will be realized when methods at present under investigation to introduce high-grade sulphide concentrates into the converters are engineered. The preliminary work with oxygen enrichment on the converters has proved rewarding.

ACKNOWLEDGEMENTS

I gratefully acknowledge the permission given by the Management of the Anglo American Corporation (Zambia) to present this paper and the contributions from the staff of N.C.C.M. (Rokana Division).

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Papers of interest

The following recently published papers may be of interest to members.

(1) A method of natural response determination, using pseudo random test signals, by W. Charlton (Senior Lecturer, Wollongong University College, Australia) and A. W. Roberts (Professor of Industrial Engineering The University of Newcastle, Australia).

(2) South West Africa: electricity for the future, by H. R. Hickman.

The above appear in *S. Afr. Mech. Engr.*, March 1974.

(3) ESCOM'S proposed real time computer based control and load despatch system, by F. H. D. Conradie and J. S. Els. *Trans. S. Afr. Inst. Elect. Engrs.*, March 1974