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NEW DEVELOPMENTS AT THE LEACH PLANT AT NCHANGA CONSOLIDATED COPPER MINES LIMITED CHINGOLA DIVISION

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INTRODUCTION

In 1961 a paper on "Leaching of Oxide Concentrates at Nchanga" was submitted for discussion at the technical sessions of the then Northern Rhodesia section of the Seventh Commonwealth Mining and Metallurgical Congress. It was presented by the author E.W. Page. Since 1961 numerous changes have taken place within the A.A.C. Group and within Nchanga itself, and although some duplication of the above-mentioned article will be inevitable, the writer will concentrate on the major new developments:

(a) A low grade oxide treatment plant came into production in 1960.
(b) Two additional tankhouse units were added to the four existing ones in 1962.
(c) A roaster circuit was commissioned in 1964.

These extensions and modifications have increased the producing capacity of the Leach Plant from 6,000 tonnes of cathodes to 9,900 tonnes of cathodes per month. The different types of concentrates treated at the moment through the Leach Plant of the Chingola Division of the Nchanga Consolidated Copper Mines Limited are tabulated in table 1.

The oxides are all leached directly with spent electrolyte; the sulphides are first converted to oxides or sulphates in the roaster circuit, before joining the main Leach Plant circuit. Acid is produced at the Rokana Division of N.C.C.M. Ltd. in two contact plants; the second unit came into production in 1965. Due to the increased demand for sulphuric acid, a third contact acid plant was constructed recently at the Luanshya Division of Roan Consolidated Mines Limited, the second major copper producing group in Zambia. Burned lime is supplied by the Ndola Lime Company at a rate of 3,000 – 4,000 tonnes per month.

TABLE 1
Concentrates treated at Chingola Leach Plant.

Type of Concentrates	% Total Cu.	Monthly Throughput
High grade oxide from Chingola Concentrator	15-18	33,000
High grade oxide slimes from Chingola Concentrator	10-12	15,000
Low grade sulphide from Chingola Concentrator	10-15	11,000
High grade oxides from Konkola Concentrator ex Chingola ore	15-18	11,000

The Rokana Division of N.C.C.M. Ltd. still produces the starting sheets and loops for the tankhouse. In a few years, however, these will be produced at Nchanga in the stripping section of a second tankhouse which is being constructed presently as part of massive extensions taking place at the Chingola Division. The cathodes produced in the tankhouse are railed to the refinery of the Rokana Division for furnacing and casting into shapes.

THE LEACHING PROCESS

A flowsheet of the Chingola Division's Leach Plant operations is presented in fig. 1. The high grade and

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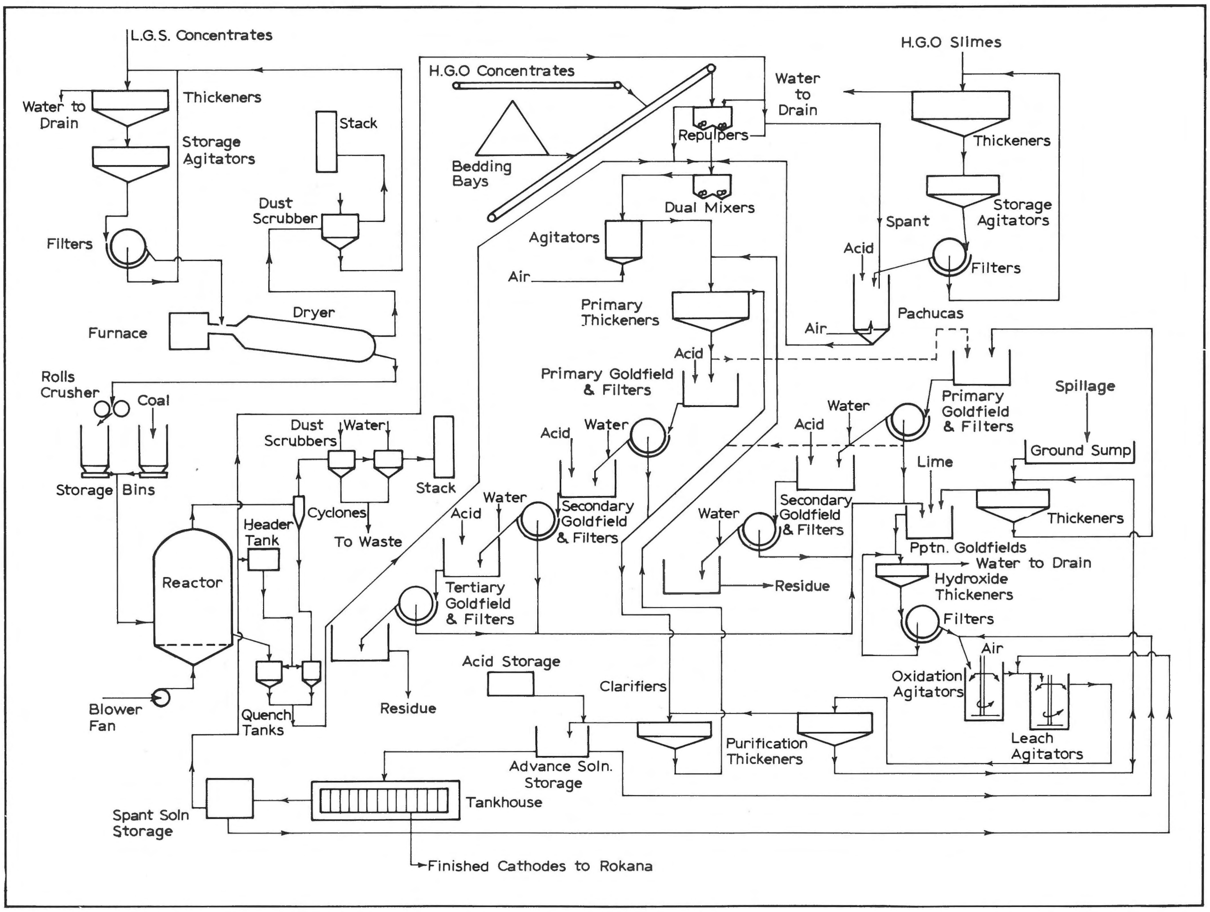

Fig. 1
Flowsheet of Chingola Leach Plant.

low grade concentrates are leached with spent electrolyte from the tankhouse. The low grade sulphide concentrates are first converted to oxides or sulphates in a fluosolids reactor before spent electrolyte is again used as the leaching agent. The partially leached pulps of the three circuits join at the agitators where the final leaching takes place.

The resultant pulp is thickened. The copper sulphate solution is clarified and strong acid is added to make up the required acid balance. The advance solution passes to the electrolytic tankhouse. The underflow of the thickeners is filtered and washed in three stages. The primary filtrate joins the main solution circuit, the secondary and tertiary filtrates are pumped to the precipitation section. The cake of the tertiary filters is again repulped and pumped to the tailings dams or to the recently commissioned Cementation Plant for further treatment.

All dilute copper sulphate solutions are pumped to the precipitation section, where they are mixed with milk of lime. The precipitated copper hydroxide is thickened, filtered and redissolved in advance solution. This pulp is pumped to the purification circuit where under carefully controlled conditions the bulk of the copper is leached out and the undesirable elements are left in the precipitated form. The pulp from the purification section is thickened, the overflow joins the main solution circuit, the underflow is filtered and washed twice before being pumped to the tailings dams.

High Grade Oxide Circuit

High grade oxides, produced by the Chingola Concentrator are transported via belts directly or via bedding bins and then via conveyor belts to the repulpers. The Chingola concentrates from the Konkola Concentrator are received by road and delivered to the bedding bins. The concentrates stockpiled in the three bins, each of which has a capacity of 4,000 dry tonnes, can be reclaimed by Robins Messiters or by front-end loaders through an apron feeder and a ribbon feeder. A typical average assay of the high grade feed to the repulpers is presented in table 2.

TABLE 2
Typical assay of high grade feed to repulpers.

Total Cu	15.0%
Acid Soluble Cu	13.8%
Co	0.06%
Fe	2.4%
S	0.5%
SiO_2	46.5%
Al_2O_3	7.8%
CaO	1.2%
MgO	2.6%
Moisture	18.0%

The concentrates are repulped with spent electrolyte from the tankhouse to a repulp density of 1 250 g.p.l. in two mechanical agitators, one in use, one on standby. The solids intake into the plant is controlled mainly by the flowrate of spent electrolyte and by the free acid content in the solution leaving the dual spindle agitators.

The product from the repulpers passes through $\frac{1}{4}''$ mesh screen, which removes foreign material from the pulp, to three dual spindle agitators where most of the spent electrolyte is added and where the products of the Roaster circuit and the slimes circuit are discharged into the main circuit.

The final leaching stage takes place in five parallel banks of three air agitated vessels in series, each with a diameter of 20′ and a depth of 20′, and a capacity of 28,000 gallons. They are constructed of Californian red wood, lead-lined throughout and bricked with acid resistant bricks at the bottom to minimise wear. The wooden planks are kept together by hoops. Recently some corrosion of the hoops, a slight deterioration in the strength of the wood and collapse of some of the bricks was noticed, which led to the decision to replace the wooden vessels in the future with pachuca type agitators and rubber lined inside and outside. Two units are already installed. Compressed air for agitation is provided by 3 Ingersoll Rand compressors with a capacity of 936 c.f.m. each at 30 p.s.i. In case of local power failure, the compressed air for the agitators is automatically drawn from the high pressure air, generated by the Chingola Division's Power Station. Some relevant data on leaching conditions are given in table 3.

TABLE 3
Leaching data.

Spent electrolyte flowrates:	1000-1800 gallons/min.	
Spent solution assays:	free H_2SO_4	52.0 g.p.l.
	Cu.	28.0 g.p.l.
Agitator discharge assays:	free H_2SO_4	11.0 g.p.l.
	Cu.	50.0 g.p.l.
Leaching efficiency:	98%	
Gangue acid consumption:	800-1000 lb. acid/ton of Cu.	

Low Grade Sulphide Circuit

Introduction. – In 1962 the decision was taken to go ahead with the construction of a Roaster Plant at Nchanga after extensive studies and laboratory tests. The advantages of having a Roaster circuit were:

(a) The grade of sulphide concentrate to the smelter could be improved because now the lower grade fraction of the sulphide flotation circuit, which contains a very high SiO_2 content, could be treated via the Roaster Circuit.
(b) Depending on the roasting conditions, copper sulphide would be converted to copper sulphate, which forms acid on electro-winning and would decrease acid consumption at the Leach Plant.
(c) Better solution storage control could be achieved as concentrates without any moisture would be introduced into the Leach Plant circuit.

The Roaster section was commissioned in 1964 and has been running very successfully since.

Description of Circuit. – A flowsheet of the circuit is presented in fig. 2. A sulphide pulp from the Concentrator, containing 10-15% sulphide copper and 5-6% sulphur, is pumped to two 50′ diameter, 140,000 gallons capacity thickeners and one new 80′

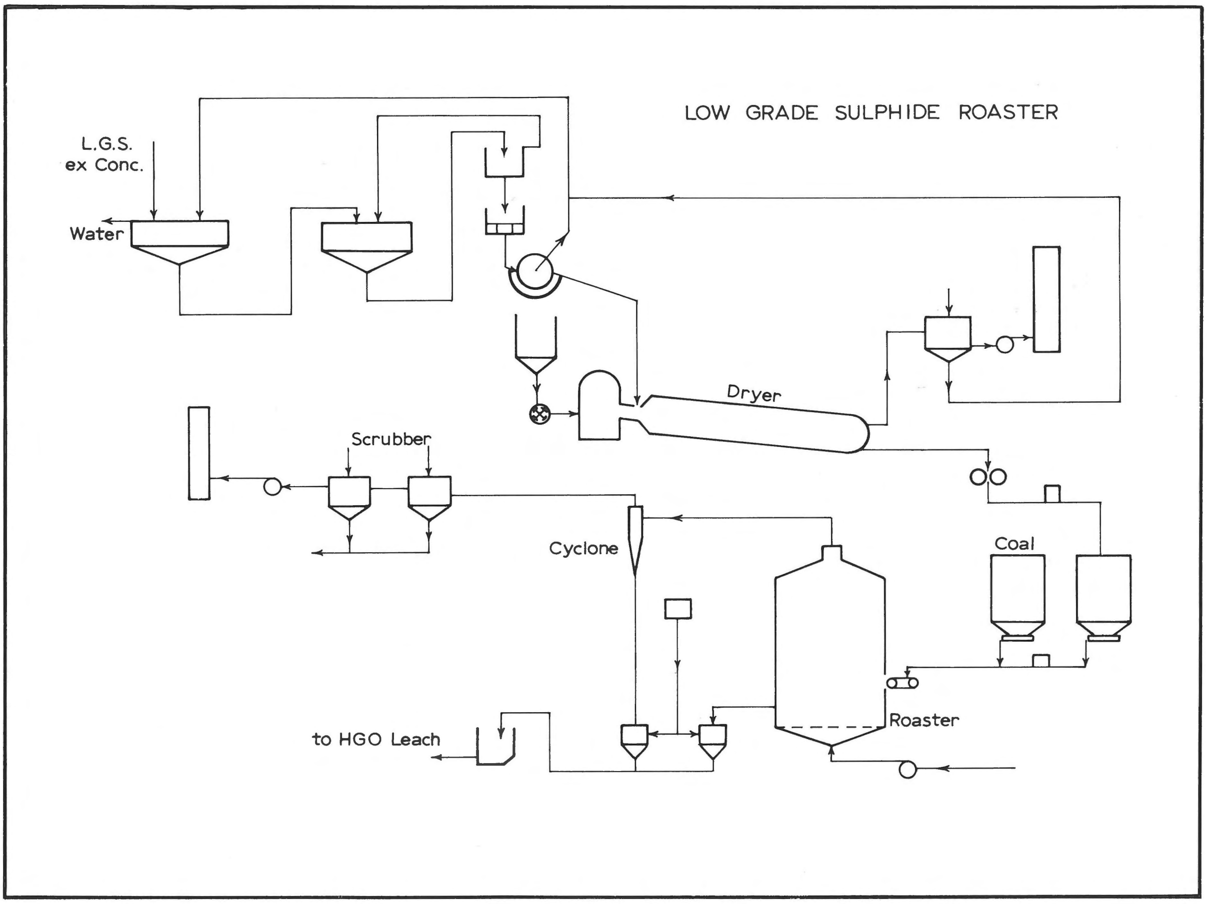

Fig. 2
Low-Grade Sulphide Roaster

diameter, 350,000 gallons capacity thickener at a density of approximately 1 080 g.p.l. The thickeners are equipped with automatic raise and lower mechanisms and with variable speed Dorrco underflow pumps. The underflow density is maintained at 1 600 g.p.l. and pumped to two 50′ 140,000 gallons storage agitators.

From the storage agitators the pulp is pumped to three rotary drum 14′ x 16′ variable speed filters fitted with cotton cloths. The boot level of these filters is automatically controlled. The moisture in the pulp is reduced from 40% to 18% by weight. The filter cake is discharged into a coal fired 86′ long and 9′ diameter rotary drier. The throat temperature is maintained at around 1832°F (1000°C), the exit temperature at about 212°F (100°C). The residence time in the drier is approximately 35 minutes during which time the moisture of the concentrates is reduced to 8%. Before the exhaust gasses are discharged into the air they are scrubbed once by one Elbair scrubber, the 48 sprays of which deliver 80 gallons/min. at a pressure of 80 p.s.i. Elbair effluent returned to thickener. The product from the drier passes via a conveyor belt system through a roll crusher, one in use, one on standby, set at 3/16″, to two storage bins with a capacity of 60 short tonnes each.

Due to the low sulphur content of the concentrates, the temperature in the reactor itself has to be maintained at the required level by the addition of coal. For this purpose a 30 ton pulverised coal storage bin is situated next to the concentrate storage bins. During times of high sulphur in the concentrates when the reaction is autogenous a waterspray is used to control the temperature. Concentrates and pulverised coal are discharged via automatically controlled rotary table feeders into a concentrate flinger, which is a very short, fast running belt, which flings the concentrates in to the middle of the reactor bed. The roaster itself is a Dorr Oliver Fluosolids Reactor, 27′ in diameter, 24′ high of which approximately 4′ is the depth of the bed. During normal operating conditions, 7 500 c.f.m. of air is blown through the reactor, the windbox pressure is 60″ w.g. and the freeboard draught 0.3″ w.g. The temperature is maintained at 650°C. Feed rates can vary between 15 – 30 tonnes/hr. Under these conditions the sulphides are mainly oxidized. Although a maximum sulphating rate is desirable, this would involve lower temperatures and lower air throughput, which makes operating conditions much more difficult. (For a comprehensive study of the chemistry of roasting and of roasting conditions, reference is made to the thesis by W.P.C. Duyvesteyn, written in 1967 in Chingola, to obtain a M.Sc. in Mining and Metallurgy from the University of Delft, Holland.) The air for the reactor is delivered by a Howden-Safanco three-stage turbo blower, one in use, one on standby, with a capacity of 10,000 c.f.m.

The roasted products flow from the bed to the quench tank through a 12″ line. In the past this used to be a complicated fluoseal system which gave so much trouble that the necessity of having a fluoseal bed overflow was questioned. Successful experiments were conducted with a straight overflow pipe which, after a few modifications, was permanently used. It is estimated that approximately two-thirds of the feed to the reactor is discharged directly into the quench tank. The hot calcine is leached with spent electrolyte, flowrate 350 gal/min. and pumped through a 6 x 4 Warman pump to the dual spindle agitators in the Agitator section, where it joins the main circuit.

The exhaust gasses of the reactor containing one-third of the feed to the reactor are passed through four parallel cyclones with a diameter of 5′8″. The cyclone overflow is wet scrubbed in two stages. The scrubbers are again Elbair scrubbers, two primaries, one secondary, each is a 6′ x 4′ two stage unit. Water for the primaries is pumped at a rate of 150 gal/min. at 120 p.s.i., the secondary Elbair water flowrate is 80 gal/min. at 80 p.s.i. Elbair effluent returned to thickener. The underflow of the cyclones is quenched with spent electrolyte, flowrate 90 gal/min. through each of the five quench tanks and is pumped to the dual spindle agitators together with the reactor bed overflow. The scrubbed gas is drawn through a stainless steel Howden-Safanco exhaust fan, one on duty, one on standby, discharging 30,000 c.f.m. into a 150′ high lead lined stack. To maintain a constant suction in the reactor an automatically controlled Court valve is installed just before the exhaust fans. The temperatures at five points in the lower bed and at five points in the upper bed plus the upper and lower mid bed temperatures are recorded continuously by a Kent recorder.

The Low Grade Circuit

Introduction. – As mentioned in Page's paper, the Group was thinking of scavenging mill tailings to produce a low grade oxide concentrate and to maximise electrolytic copper production. This concentrate could be economically treated in a separate circuit in which the oxides from the Concentrator were thickened and filtered. The filter cake was repulped with stripped spent electrolyte to a density of 1 450 g.p.l. The resultant pulp was leached in air agitated pachucas in which strong acid was added to obtain a high leaching efficiency. The discharge of the pachucas was pumped to the newly built filterplant where the cake was washed twice. The primary filtrates were sent to main circuit, the secondary filtrates to the precipitation section. The plant was commissioned in 1960. Due to shortages of acid the separate treatment of low grade oxide was stopped in 1964. The facilities were then used to treat the slimes portion of the oxide concentrates, the treatment of which caused tremendous handling and repulping problems at the Leach Plant. The plant is now split in two sections operationally, one section treating the high grade oxide slimes, the other section providing two stage filtration for a variety of acid pulps.

Description of the H.G.O. Slimes Leaching Circuit. – A flowsheet of the process is given in fig. 3. High grade oxide slimes with 10 – 12% Cu. are pumped from the Concentrator at densities of 1 100 to 1 150 g.p.l. to three 80′ diameter thickeners, capacity

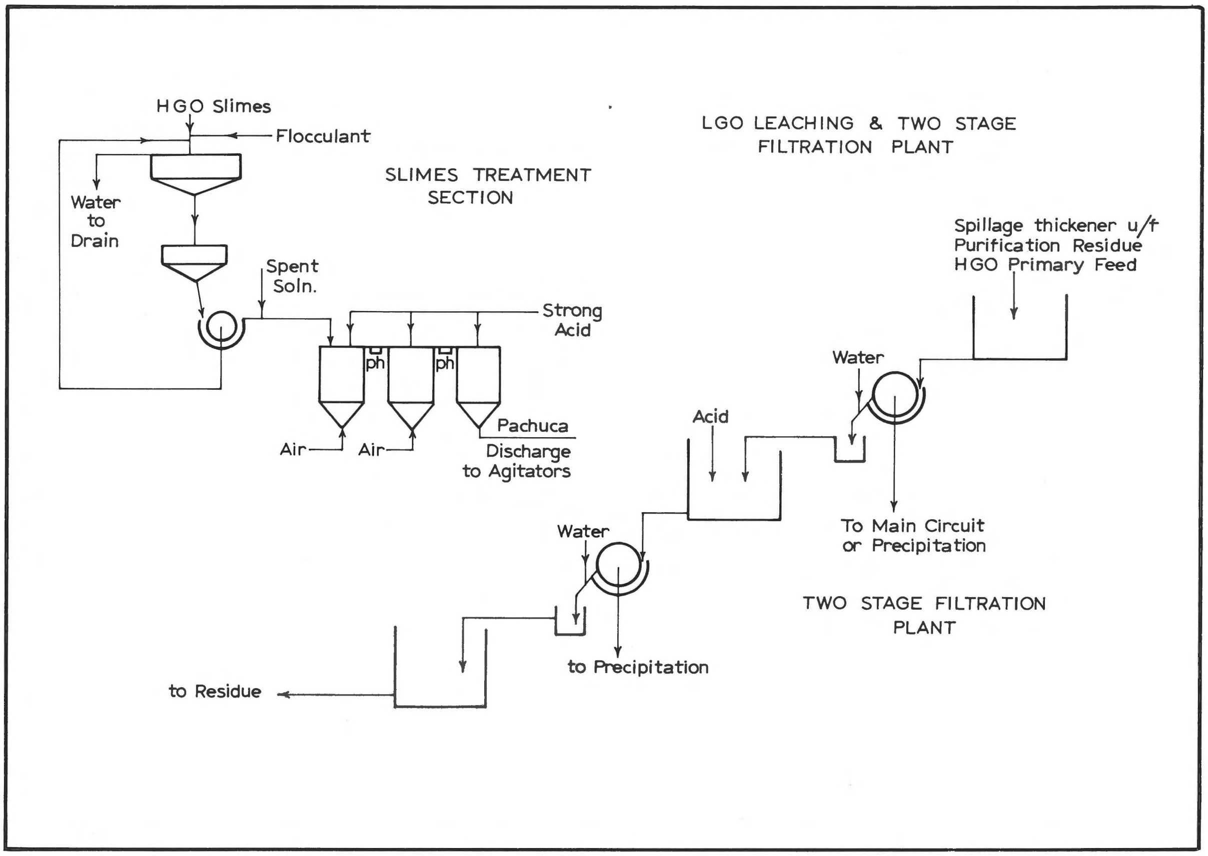

Fig. 3
LGO Leaching and Two Stage Filtration Plant.

370,000 gallons, equipped with Dorrco pumps. The pH of the slurry from the Concentrator is approximately 8.0. Difficulties are experienced with the settling of the solids and the only effective settling reagent up to date has been found to be lime. At a pH of 10.0 or higher, clear overflows are obtained. The pulp is thickened to a density of 1 450 g.p.l. and pumped to two 50′ diameter storage agitators, capacity 150,000 gallons.

The moisture content of the pulp is decreased from 50% to 20% on seven 16′ x 14′ diameter drum filters using cotton cloth. The filter cake is repulped with spent electrolyte, 60 gallons/min. to each filter and the pulp is pumped to three air agitated rubber lined pachucas, each of a capacity of 13,000 gallons. The pH in the first two pachucas can be controlled automatically. Careful control, however, is not necessary as the pachuca discharge is pumped to the leach agitators, where final leaching takes place.

As mentioned earlier, the high-grade oxide circuit, the slimes circuit and the sulphide circuit all discharge their partially leached pulps into the leach agitators, from where onwards there is no difference in the treatment of the various pulps.

The Primary Thickeners. – The agitator discharge gravitates to five 80′ diameter, lead lined primary thickeners. Problems were encountered with the thickening of the material treated. Phenomena, which up to now have not really been explained, caused damage to rakes and rake drive mechanisms, resulting in a lot of down-time. One explanation is the formation of floating "doughnuts" which suddenly collapsed and settled. In 1968 it was decided to equip these thickeners with "Klopper" rakes. The stainless steel rake arms are suspended from a beam, which is fixed onto the rate shaft with 3 stainless steel chains. The rake arm itself is fixed onto the shaft via a pivot. When the density at the bottom of the thickener increases, the forces on the rake arms increase as well, with the result that the rake arms lift themselves automatically from this area with a high pulp density to an area with a lower pulp density. When the density of the pulp falls below a certain level, in Chingola's case 1 450 g.p.l., the rake lowers itself by gravity. These rakes are so effective that it is virtually impossible to trip the rake drive mechanism on overload.

Twice per day a density profile over the depth of all thickeners is taken and graphed, which gives an excellent indication of the solids content of the thickeners. To improve settling of the solids and clarity of the supernatant liquor, tests are continuously carried out in the laboratory and in the plant with various additives. The reagent in use at present is a mixture of 6 : 1 polyacrylamide/guargum, which is added at a rate of 0.04 lbs per tonne of solids. The thickened pulp is withdrawn from each thickener by three 8″ Dorrco pumps. The length of the stroke of these pumps can be controlled to pull forward at the required density.

Clarification – Storage. – The thickener overflow gravitates to a pumpbox from where it is pumped to a header tank and split equally over five 80′ diameter clarifiers, each of a capacity of 370,000 gallons in which the suspended solids content of the solution is brought down. A polyacrylamide NP 10 and a guargum M 61, at a ratio of 3 : 1 and at a rate of 1.5 lbs per ton of copper produced are added in this section to the solution as smoothing agent for the cathodes. Foaming Agent is added to prevent fumes escaping into the air in the tankhouse electrolyte cells, at a rate of 0.3 lbs per tonne of copper produced. Previously a smoothing oil was added, this however made operations untidy and was not as effective. The solids in the clarifiers are pumped back to the primary thickeners when the necessity arises. The solution from the clarifiers passes into five 50′ diameter, 150,000 gallons capacity, storage tanks. In this section the extra acid, lost in the dissolution of gangue materials, in the residues of the filter plants, and in the feed to the precipitation circuit, is added. The other purpose of adding acid is to optimise the acid balance in order to get maximum efficiencies in the electrolytic Tankhouse.

The Filter Plants

The High Grade Filter Plant. – The underflow of the primary thickeners containing 50% solids is pumped to the high grade filter plant where normally it is filtered in three stages. The copper tenor in the entrained solution is reduced from 50 g.p.l. to 0.5 – 1.5 g.p.l. The filters employed are either Dorr Oliver filters, 14′ x 16′ and 16′ x 18′, or converted belt filters again in both sizes. Tests with various cloths continue, presently lainyl is in use. The belt filters are considered to be superior to the drum filters because: (a) No recirculation of cake takes place and (b) It is not necessary to wash the filter cloth each shift to minimise blinding of the cloth. The disadvantages of the belt filter however are the continuous attention a skilled operator has to pay to the alignment of the cloth and the more frequent maintenance requirements.

The cake from the filters, containing 20% moisture, is repulped with water in 4′6″ x 4′6″ Vortex mixers. The pulp gravitates to storage Goldfields, 26′ diameter, 30,000 gallons capacity and is pumped from the agitators to the next filtering stage. The pulp level in the boots is autmatically controlled. Filter vacuum is maintained at 15″ – 20″ w.g. by 22″ x 9″ and by 31″ x 13″ Ingersoll Rand vacuum pumps, which in the near future will be changed to Nash rotary pumps. The blow-off air at 10 p.s.i. is delivered by 23″ x 13″ Ingersoll Rand compressors. To assist filtration by obtaining thicker filter cakes and lower moisture contents, reagents are injected, at a controlled rate, into the feedlines to the filter boots. The reagent presently in use is a mixture of six parts polyacrylamide to one part guargum. The rate of addition is 0.5 lb per tonne of dry solids.

The Low Grade Filter Plant. – At the low grade filter plant there are five non-acid filters, used for the filtration of oxide concentrate slimes, and eleven acid resistant filters, available for various purposes. Three of the acid resistant filters are 14′ x 16′ belt filters, the others are 14′ x16′ drum filters. The belt filters are mostly used for the filtration of purification residue, the drum filters can be used (a) As a fourth and fifth stage filtration of high grade filter plant residue, and (b) As extra primary and secondary filter stages in times of a high solids throughput, when the high grade filter plant cannot cope sufficiently. The vacuum for these filters is maintained at 15″ – 20″ w.g. by five 31″ x 13″ Ingersoll Rand vacuum pumps which again will shortly be replaced by Nash rotary vacuum pumps. The blow-off air is delivered by the high grade filter plant compressors.

The Precipitation Circuit. – Solutions of low copper tenor are sent to the precipitation section for water removal. These solutions are secondary and tertiary filtrates from the high-grade and low-grade filter plants and spillage thickener overflow. The solutions are mixed with milk of lime in four 26′ diameter Dorr Goldfields, each with a capacity of 30,000 gallons. Copper, iron and other hydroxides are precipitated. The pH in the agitators is automatically controlled at 5.6. The hydroxide pulp is thickened in six acid resistant 50′ Dorr thickeners to a density of 1 200 g.p.l. The thickener overflow is pumped to the Concentrator where it is used as a settling aid in the concentrate thickeners. The underflow is filtered on two 16′ x 14′ drum filters and one 12′ x 11′6″ belt filter. The filter cake containing 16% copper is repulped with primary thickener overflow and pumped to the purification section.

The Purification Circuit. – It is imperative to keep the iron tenors in solutions fed to the electro-winning plant as low as possible. After numerous results of testwork were evaluated by the Operational Research Department (reference is made to a report by E.B. Hayes entitled "The Effect of Iron Tenor on Current Efficiency") it was found that an increase in iron tenor of 1 g.p.l. can decrease the current efficiency by as much as 1½%. In order to retain the iron tenor at the required level of below 2.0 g.p.l. part of the main circuit solutions, together with the precipitation discharge, are purified of this unwanted element. It would be advantageous to bleed off spent electrolyte only as approximately 50% of the iron in spent electrolyte is present in the Fe^{3+} state and $Fe(OH)_3$ precipitates at the low pH's employed. However, the acid content of spent electrolyte, is so high that only a relatively small quantity of solution would have to be bled off for the treatment of the constant quantity of hydroxide cake and hence only a limited removal of iron from the main solution would be achieved. It is for this reason that a mixture of primary thickener overflow and spent electrolyte is bled off to the purification section.

There are two banks of four 26′ diameter purification agitators in parallel, each with a capacity of 31,300 gallons. The pH of the first three agitators is automatically controlled at 3.5 and the pulp is violently agitated with excess air in order to oxidize the Fe^{2+} to Fe^{3+}. At a pH of 3.5 $Fe(OH)_3$ is stable in the precipitated form. In the last agitator the pH is decreased to 2.0 by the addition of spent electrolyte. After extensive testwork it was found that at this pH most of the Cu in the hydroxide is leached out, whilst most of the iron and other impurities still remained in the precipitated form. A slightly higher pH would metallurgically be better, however at a pH higher than 2.0 the product cannot be filtered properly.

The leach agitator discharge is thickened in two 80′ diameter thickeners, each with a capacity of 428,000 gallons. The overflow of the thickeners, containing 50 g.p.l. Cu is further clarified before being sent to the main circuit. The underflow is sent to the belt filters in the low grade filter plant. Considerable difficulties are experienced with settling and filtration of the solids in the purification agitator discharge. An uneconomical dosage of the presently known reagents would be necessary to obtain a clear overflow; the filtration problems were reduced by the introduction of belt filters on which this very fine precipitate filters much better than on drum filters. Also the introduction of a fully automatic reagent injection system, injecting the reagents into the feed to the filter boots improved filtration.

The Tankhouse Section. – For a detailed description of the electrical circuit reference is made to the article by E.W. Page. Operations in general did not change in this section since this article was published. The following modifications and additions however took place (1) With the higher copper production through the Leach Plant, the Tankhouse capacity had to be increased. Two more units constructed to the north of the existing units came into production in 1962, increasing the tankhouse capacity to 9,000 tonnes per month, and (2) The Mercury arc rectifiers were changed to Germanium rectifiers which in turn were again replaced by Silicon diode rectifiers in 1969. One Mercury arc rectifier was kept as a standby unit throughout this period. The direct current through each circuit has been increased to 14,000 amps, giving a cathode current density at full load of 18 amps/ft^2.

Rigid control of the iron level in the tankhouse over the last year increased the current efficiencies to over 93%, which is very high for electro-winning. An average composition of the feed to the tankhouse and of the cathodes produced are given in tables 4 and 5.

TABLE 4
Electrolyte to Tankhouse.

Cu	40 – 50 g/l
Cl'	10 – 20 p.p.m.
Co	1.0 – 1.5 g/l
Mg	8 – 13 g/l
Al	2.0 – 5.0 g/l
Mn	3.0 – 5.0 g/l
P	1.5 – 3.0 g/l
F	0,15 – 0,40 g/l
So_4'	120 – 180 g/l
Si	0.50 – 0.80 g/l
Bi	0.9 – 1.30 g/l
Fe	1.0 – 3.0 g/l
Se^{4+}	0.06 – 0.20 p.p.m.

TABLE 5
Composition of Cathodes.

Cu	99.7%
Co	12 p.p.m.
Mg	90 p.p.m.
Al	60 p.p.m.
Mn	30 p.p.m.
P	7 p.p.m.
Ni	0.8 p.p.m.
Pb	20 p.p.m.
Si	0.012%
Bi	0.20 p.p.m.
Sb	1.0 p.p.m.
Fe	70 p.p.m.
Se	2.0 p.p.m.
Loss in H_2	0.25%

The cathodes are shipped to Rokana Division of N.C.C.M. Ltd. for further treatment.

FUTURE DEVELOPMENTS

Two developments of major importance to the Group are taking place presently at Chingola:

1. A £ 3,000,000 plant has been constructed.
   (a) To treat low grade oxide concentrates which have been stockpiled since the acid shortages and the closing of the Low Grade Plant at a rate of 100,000 tonnes per month. containing between 2 and 3% Cu.
   (b) To treat the Leach Plant residue at a rate of 50,000 tonnes per month.

   Treatment of these low grade materials was made possible after the commissioning of a third Acid Plant on the Copperbelt at the Luanshya Division of R.C.M.

   The main feature of the plant is the use of three "Kennecott Cones" in which the copper from low tenor solutions is precipated on to scrap iron in a continuous process. This Cementation Plant was being commissioned at the time of writing and it is envisaged to produce 2,500 tonnes of copper per month from this plant. It was designed and constructed within a period of two years. Capital investment was low, operating costs are high. Immediate cash flow is, however, available before the Solvent Extraction Plant comes on stream, which plant will have a high capital cost but very low working costs.
2. A multi-million Solvent Extraction Plant is being constructed which will treat 1,000,000 tonnes per month of current flotation tailings and reclaimed flotation tailings dams.
   The acid soluble copper will be leached and the pulp will be washed in four 250′ thickeners employing counter current decantation, the pregnant liquor will be enriched in a solvent extraction plant, employing the General Mills product LIX 64N, the advance solution will be pumped to a new tankhouse where another 6,000 – 9,000 tonnes of copper will be deposited per month.
   This plant will come on stream sometime in 1974 and will make use of some of the equipment of the Cementation Plant, which by then will be phased out.

It should be mentioned that with the Cementation Plant coming into production very soon and the Solvent Extraction Plant in the near future, interesting possibilities arise on the integration of the existing Leach Plant and the two new Plants. This subject, however, is wide enough for an article in itself.

UNITS

Although Zambia has metricated its units throughout commerce and industry, the old imperial units were still used as a matter of convenience.

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REFERENCES

Page, E.W. (1961) – Leaching of Oxide Concentrates at Nchanga. 7th Commonwealth Mining and Metallurgical Congress (Northern Rhodesian Section).

Duyvesteyn, W.P.C. (1968) – Conversion efficiency of Low Grade Copper Sulphides and the Acid Gain to the Circuit by Roasting in a Fluosolid Bed. Thesis, Delft.

Hayes, E.B. (1967) – The Effect of Iron Tenor on Current Efficiency. Internal Company Report.