RL 8804 . ANNUAL REPORT.
1989/90 KING RIVER DELTA
PROJECT, MACQUARIE HARBOUR
TASMANIA.

90-3107

000

90·3107

506002

RL 8804 . ANNUAL REPORT . 1989/90

KING RIVER DELTA PROJECT
MACQUARIE HARBOUR TASMANIA

MOVED TO OPEN FILE
19·6·95 AT
DIRECTION OF
REGISTRAR OF MINES

AMG REFERENCE POINTS ADDED

Elisna Pty Ltd, for
Cottesloe Corporation
New Jersey USA

February, 1990

CONTENTS

I   BACKGROUND

Late in 1987 Cottesloe Corporation applied for a retention licence to
cover the northeasterly eight square kilometers of the King River Delta.
At the time the Delta deposit was held under Exploration Licence 2/74 ,
thirteen sq km in extent.

In July 1988 an objection to the application was lodged on grounds that
sampling at the Delta might interfere with operations of a recently-formed
company attempting to rear salmon at a hatchery on the eastern shore of
Macquarie Harbour and at another on its southwestern coast. The case was
brought before the Warden's court at Devonport later in the year but
thrown out when the plaintiff's legal representative failed to appear.
The representative met instead with Mines Department officials concerned,
requesting imposition of stringent limits on sampling. As these
restraints could seriously hamper field work Cottesloe asked that terms
recommended be modified.

Cottesloe's consultants drew attention to the fact that high densities of
suspended matter occur naturally in Harbour waters over extended periods.
Sediment derives principally from flood discharges of the Gordon and King
Rivers and spreads across the entire Harbour when strong westerly winds
combined with spring tides block outflow from it. Also, poisonous Lyell
tailings which over the last ninety years have accumulated at the King
River mouth are remobilized at such times and then invade and pollute all
reaches of the Harbour. They will continue to do so, if left untouched,
for the next century at least, even if Mt Lyell's dumping is stopped
now.

Dredging the Delta and treating all its sediment as Cottesloe proposes
will remove contaminants and deposit the cleaned material in deep water
away from the river mouth within the span of two decades.

Another deterrent to safe culture of edible fish inside the Harbour is
that mud and silt flooring it are poisonous: they are anoxic and contain
concentrations of heavy metals. Rapid, large variations in salinity of
the bay's surface waters constitute yet another barrier to successful
fish farming within its confines. Because of these factors there are
essentially no natural hatcheries there. The Harbour's resident shell-
fish, too, are rare and stunted.

The Department of Mines adopted modified terms suggested by Cottesloe and
late in February, 1989, Retention Licence 8804 was issued.

Cottesloe's last annual report on the King Delta identifies several major errors in a survey of gold carried out by Planet Resources over that sector of the deposit generally having its upper surface above water.

Planet's results have since been studied in detail, permitting the Company's computational mistakes - if not the sampling one - to be quantified. Examining gold distribution in sectional arrangements of samples Planet's contractor collected reveals the existence of four layers: see Figure 1. The four are not represented in every auger hole because holes vary in position and length, and layers vary in attitude and degree of continuity. However, the sequence in which layers occur is constant and the gold grade range of each distinctive.

Layer Y, the uppermost, is encountered in 20 of the 26 holes augered. Where drilled through, it varies in thickness from 1 to 9 metres, averaging 2.5 metres. Gold content determined by fire assays made for Planet at Mines Department laboratories in Launceston averages 125 ppb. Material of this layer is oxidized, partially at least, with iron pan developing from rusting pyrites at numerous points across the Delta's emergent surface. Subsurface material ascribed to this layer probably was washed off its former habitat on the dry top and re-deposited underwater at places previously representing the Delta face. The face advances seaward as the Delta grows.

Layer B underlies Y at 12 augering sites and occurs at the surface at 5 others. Where augered it ranges in thickness from a metre inshore to 8 metres or more offshore. Its top lies at a depth of 0 to 10 metres, its base between 1 and at least 15 metres below sea level. Its average thickness is 3 metres and its average gold content 78 ppb. Constituent material is thought to have accumulated under reducing conditions or under those which, while perhaps oxidising initially, became reducing when the sediment was buried under younger Deltaic material.

Layer G is made up of variable proportions of sediment from Layer B and sub-Deltaic material of Layer P. At 9 of the 13 points where penetrated Layer G is a bare metre thick. Overall it averages 1.4 metres in thickness. The mixture of components is apparent only, in fact representing inclusion within each one-metre sample of fractions from the two bracketing Layers, B and P. Knowing grades of adjacent samples from Layers B and P besides that of the intermediate layer G, the likely ratio of Layer B to Layer P material can be estimated. At a few sites Layer G material is thicker than a metre. These are thought to be points at which, on introduction of Layer B material, sediment of Layer P was stirred up and mixed and deposited with it. Average grade of Layer G material is 41 ppb Au.

Figure 1. Sediment layers, King River Delta

Depth in metres from emergent surface

Planet Resources' Auger Hole	1st	2nd	3rd	4th	5th	6th	7th	8th	9th	10th	11th	12th	13th	14th	15th
1	220	50													
2	170	80	40												
3	70	120	30	10	<10	<10	10	10							
4	130	60	<10	<10	30										
5	150	110	100	90	120										
6	130	30	30	40	30	20	10	10							
7	120	30	10	<10	10	10	10								
8	80	50	10	<10	10	10	10	10							
9	100	40	10	10	10	10	10	20	10						
10	100	60	10	10	10	20	12								
11	80	60	80	100											
12	100	60	100												
13	60	130	80	80	50	100	80	10							
14	130	100	<10	60	80										
15	120	140	(140)	(140)	140	90	100	140	(140)	(90)	90	90	120		
16	130	110	90	70	50	90	140	140	80	100	100	70	100	110	120
17	90	140	130	100	80	50	10								
18	60	60	40	60	40										
19	100	140	140	100	10	60	60	40	<10	[160]	100				
20	80	80	100	100	40	40									
21			10	10	10										
22	90	60	10	10	10	10	10								
23		110	20	10	10	[15]	10								
24	80	70	55	<10	80	80									
25	?	50	50												
26		80													

Layer Y   Layer B   Layer G   Layer P

*ppb Au from TDM fire assay on 100-g cut of each one-metre sample   ( ) interpolated

Table 1. Deltaic and sub-Deltaic Layers

Layer		Holes	Samples	Thick-ness	Area	Volume	Vx1.15	Au Range	Au Avg	Au Total
				m	km^2	Mm^3	Mmt	ppb	ppb	t.o.
Deltaic,	Y	20	46	2.3	1.25	2.875	3.306	60-220	125.2	13,300
	B	17	56	3.3	1.06	3.500	4.025	5-160	77.8	10,000
Intermediate,	G	15	21	1.4	0.94	1.313	1.510	20-60	41.0	2,000
Sub-Deltaic,	P	12	47	3.9	0.75	2.937	3.378	5-220	6.5	700
Total		26	170	10.8						
Layers,	Y & B		102	5.6°	1.14*	6.375	7.331			23,000

° or less where depth ranges of Y & B overlap

* Mm^3/m

4

Layer P comprises King River sediment plus minor quantities of Macquarie Harbour silt laid down beneath the Delta prior to arrival and deposition of Mt Lyell's tailings, spoil which makes up the bulk of the Delta. Twelve auger holes reached Layer P, extending 1 to 7 or an average 3.9 metres into it. Most of these holes lie inshore, north and south of the King River's present channel across the Delta. Grade in 42 of the 47 samples collected from this layer lies below 10 ppb Au, minimum detection limit of the analytical method used. The remaining 5 samples contain between 10 and 30 ppb Au, levels presumably produced by occurrence of isolated fine flakes of the metal derived naturally from concentrations in the King River catchment. Before lower reaches of this river system were clogged by Lyell spoil and the Delta it formed, discharge velocity through the gorge backing the mouth of the King would have been too high to permit anything but a trace of such fine-grained gold to settle and be incorporated into Layer P. Assuming grade of Layer P samples assaying < 10 ppb Au to be at or slightly above the crustal average of 4 or 5 ppb Au, content of Layer P material becomes approximately 6.5 ppb Au.

Table 1 summarizes information above concerning the four layers. It includes other data, derived from Figure 1.

Variation or lack of it areally across the main Deltaic layers, Y and B, is under study. Results concerning Layer Y are available and may be summarised as follows.

Thickness The layer is about 2 metres thick in both NNE and SSW halves of that part of the Delta augered by Planet's contractor. From inshore to offshore the layer thickens though: in the ESE half it averages 1.1 metres in thickness, in the WNW half 3.3 metres.

Grade Gold content, likewise, varies only slightly parallel to shore, being roughly 125 ppb in northern and southern halves of the Delta's augered sector. In the other direction it does change: it is a bit below average offshore and approximately 150 ppb inshore, an increase of some 25%.

Examining the vertical distribution of gold in Layers Y and B shows that - apart from minor inconsistencies - grade in each falls from the surface through the 6th metre of depth. Between 6 and 9 metres trend is indeterminate, with Layer Y registering gain and Layer B loss. From 10 metres through 15 grade in Layer B rises, generally. Layer Y is not represented over that interval. Relevant figures appear at Table 2.

Grade variations observed appear to be part artificial, part real. Drops probably result from progressive loss of auriferous and certain other particles when sample was extracted from auger holes. A sludge pump is said to have been used. Such losses should have risen as the amount of pumping needed to recover samples from progressively greater depth increased. In this context note that the fall in grade evident in both Layer Y and Layer B samples over the first six metres is an identical 31%.

Table 2. Vertical distribution of gold, King River Delta.[°]

Depth*, m	Average	ppb Au Layer Y	Layer B
1	120	130	88
2	101	126	78
3	88	116	69
4	93	135	69
5	79	103	67
6	73	90	61
7	95	120	70
8	113	140	60
9	87	140	60
10			65
11			117
12			87
13			110
14			110
15			120
Summary			
0-6 m	99.7	124.4	99.7
7-9 m	98	132	98
10-15 m			96.5

[°] inshore fifteenth of Delta's total mass
* grade from 15 m to deepest point at 32 m unknown

6

The opposite trend, an irregular increase encountered below 6 metres, we think reflects genuinely higher levels of gold present in older horizons of the Delta. This sediment accumulated when Mt. Lyell's gold recovery circuits were less efficient and spoil consequently contained more of the element. Additionally, grades of ore mined by Lyell in the early days are likely to have been higher.

If the deeper samples have indeed suffered maximum pumping losses, true grade increases at depth may be substantially larger than those shown in Table

One way, albeit indirect, of estimating the extent of losses inferred here is to compare copper contents of individual Planet holes with those of nearby holes drilled by Cities Service. Individually, Cities' holes appear to display little or no vertical grade change, either positive or negative.

Gold grade from the lower limit of Planet's augering at 15 metres to the deepest parts of the Delta Cities found, 32 metres, remains wholly unknown. In light of considerations above however, and the fact that at 15 metres Planet's grade again has attained the average of all their surface samples it is probable that the Delta's lowermost 17 metres of sediment contain at least as much gold as overlying parts.

Cottesloe has emphasized elsewhere that Delta gold grades remain minor in absolute terms, far too low to constitute an attractive target alone. As a potential byproduct though, largely recoverable in processing the deposit's sulphide for its sulphur, iron and copper, the gold is valuable.
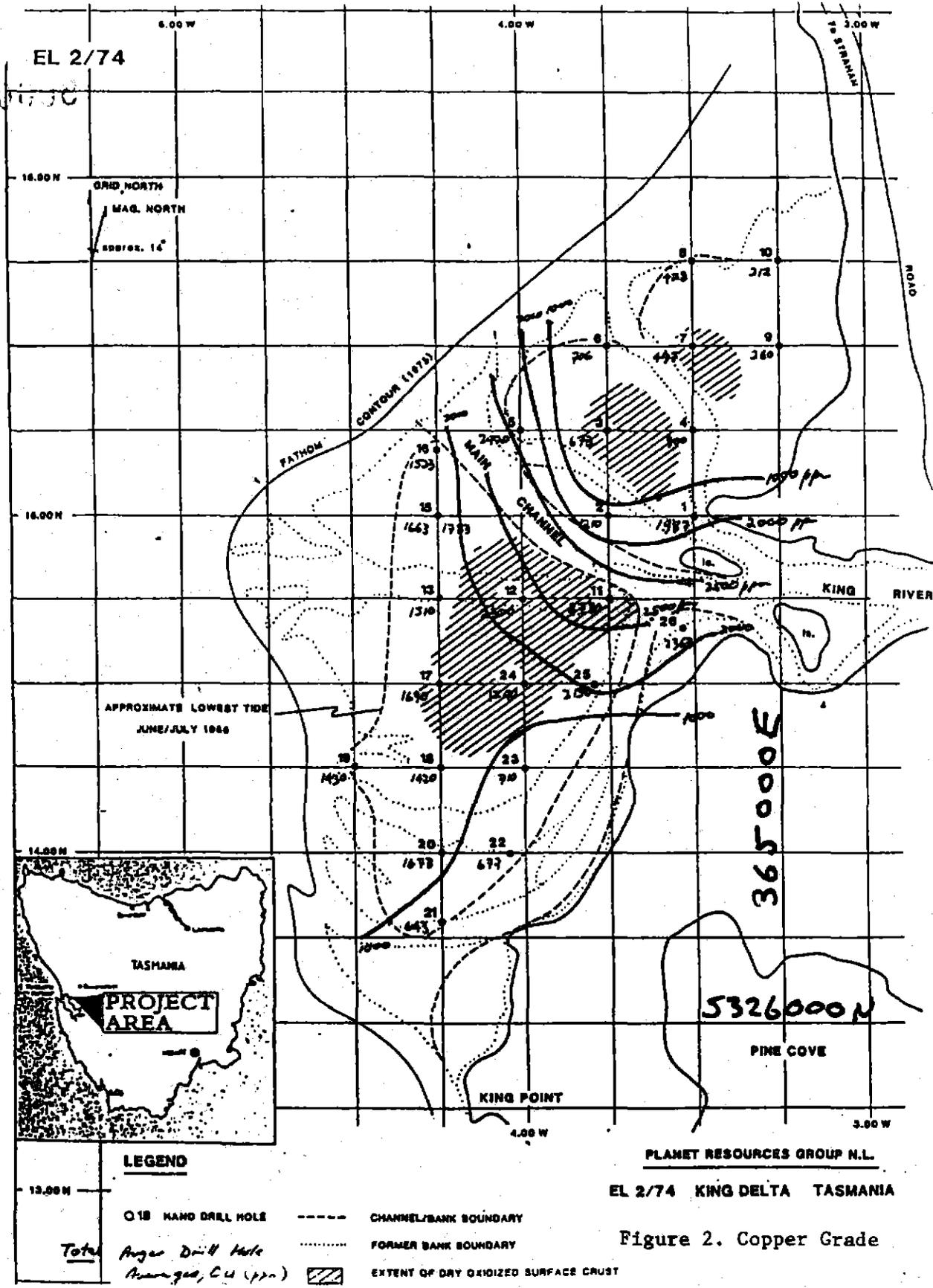
Within a minimum 100,000,000 mt of Delta sediment available for treatment the total quantity of gold present is large - even at the low concentrations reported above.

Little would be recoverable if it were evenly distributed throughout all mineral components of the sediment; but it appears most present in Planet's samples is associated with a sulphide fraction. As the latter is readily separable by inexpensive gravity means recovery of much of the gold is feasible. Such sulphide concentrate should average about 1.8 g/t Au.

Probably most of the Delta's gold occurs as exsolved blebs in chalcopyrite, reputedly its principal mode of occurrence at Mt Lyell. Such association would account for the rise in gold grade towards the Delta's eastern shore, essentially the same distribution pattern as that evinced by the deposit's copper.

In addition to pumping away gold in the smaller sulphide grains Planet's sample recovery method appears to have washed out many particles of free gold. Those of the Delta are believed to be fine-grained and flattened,

Ref. ?

what evidence ?

Figure 2. Copper Grade

LEGEND

O 18   HAND DRILL HOLE
Total Auger Drill Hole
Averages, Cu (ppm)

- - - - -   CHANNEL/BANK BOUNDARY
. . . .   FORMER BANK BOUNDARY
▨   EXTENT OF DRY OXIDIZED SURFACE CRUST

PLANET RESOURCES GROUP N.L.

EL 2/74   KING DELTA   TASMANIA

AMG REFERENCE POINTS ADDED

thus slow to settle. Where sample is pumped up and the water decanted these flakes would be highly prone to loss. Removal of much of the free gold during Planet's sampling may explain, in part at least, why their gold grades are so far belowthose reported previously by Elisna Pty Ltd. Elisna's samples were retrieved whole. These latter are surficial only. Nevertheless, inshore representatives contain over 60% more gold than Planet's samples from the uppermost metre of the same sector - some 210 ppb Au vs. 130 ppb.

For the whole mass of the Delta augered Planet puts gold grade at about 70 ppb, approximately 40% below the content calculable by combining the figures that Cottesloe derives above for the two Deltaic layers. This difference results from Planet's failure to exclude from grade calculations barren material underlying the deposit. For each auger hole Planet had a fraction of every sample collected from it combined to form a composite. Planet's Delta average is derived from the grade of each of the 26 composites, yet over 80% of sample matter recovered from some auger holes represents sub-Deltaic sediment, essentially devoid of mineral concentrations of value. A third of Planet's holes terminate in Deltaic material, the restcontinue for varying distances into barren sand beneath the Delta. In total, 33.8% of all sample retrieved and assayed consists of such underlying, non-Deltaic material.

With the incorrect grades obtained from many of the auger holes Planet then compounded error by producing inaccurate, misleading grade contours. Maps were drawn showing contours for copper and barium distribution as well as those for gold: see Figure 3.

None of the resultant distribution patterns is valid. Broadly speaking they show grades of all three elements increasing towards the King River's present channel across the Delta, and lower contents in emergent flats to the north and south.

What these apparent grade changes really reflect are the differing proportions of barren sub-Deltaic material included in sediment augered at various sites. As inshore sectors of the Delta bracketing the King's main channel are thin, holes augered in these sectors contain high percentages of barren, sub-Deltaic sediment and the sectors therefore seem impoverished in gold and other valuable components. Holes near the channel were augered exclusively or mostly in Delta material proper and thus exhibit grades which - pumping losses aside - are correct, or closer to being so.

Real changes of grade generally trend east/west or thereabouts, practically at right angles to the direction Planet claims. Besides eastward - that is, shoreward - increase of gold described earlier in this report and detected previously by Elisna Pty Ltd, similar changes were found by Cities Service, affectingcopper and other metals. Cities' plot of copper

Figure 3. Copper grade, King River Delta, Tasmania

Contour interval, 200 ppm Cu

1,000 ppm

1,000 ppm

0   1

km

5 cm
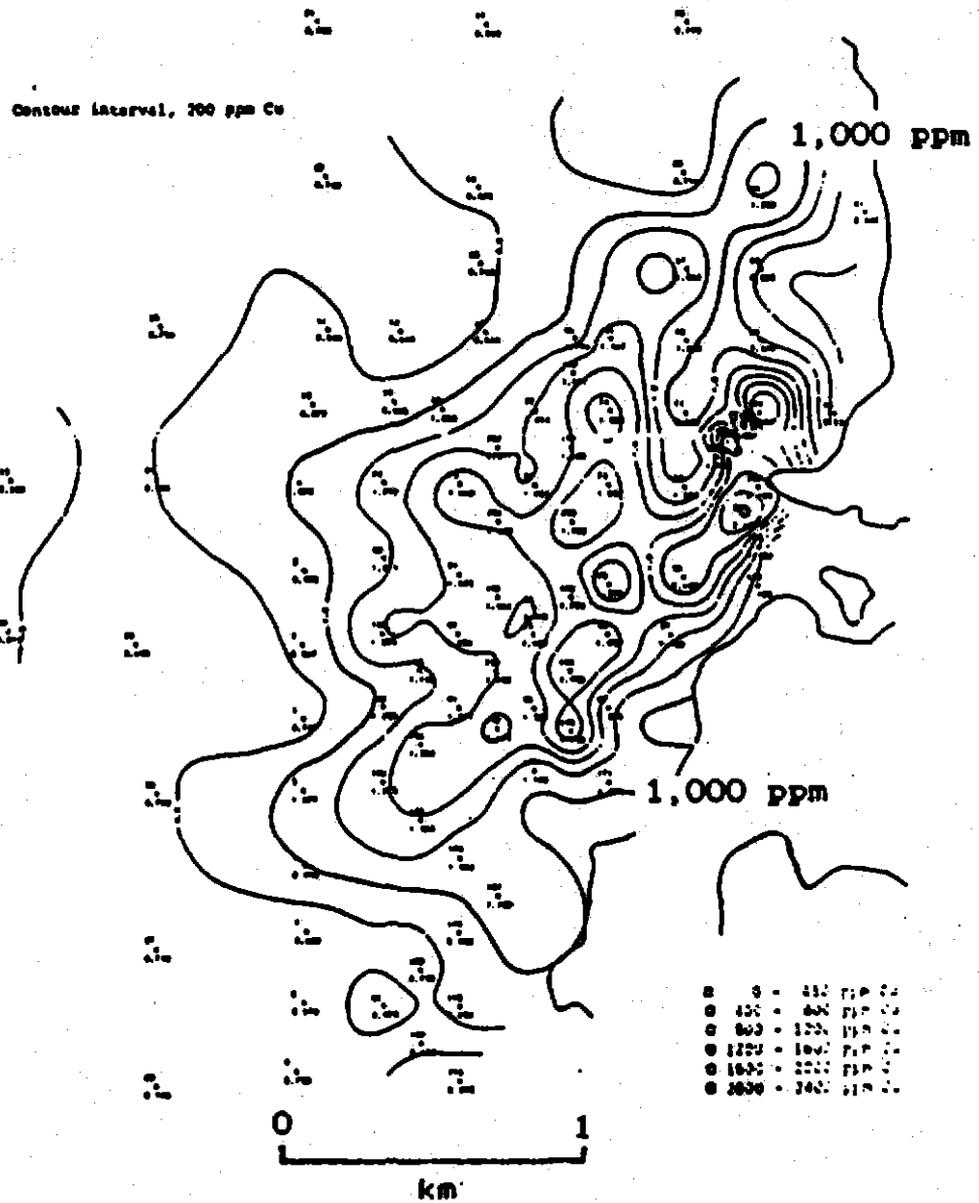
distribution, based on several hundred assays representing scores of drillholes, appears in Cottesloe's RL application report and is reproduced here as Figure 4.

Planet's survey was conducted conscientiously, both by its own junior staff and the contractors'. It is a pity it was marred, within each organization – apparently by inadequate direction. If senior officers had visited the Delta and supervised field operations and subsequent office studies more closely, mistakes in sampling procedure and in grade calculations and interpretation could have been avoided.

III   TESTING

Work in progress is designed to show where and to what extent the Delta's
sulphur, iron, copper, cobalt, gold and barium are concentrated. The
second objective is to determine how best to upgrade and extract them.

Between 1974 and 1986 numerous studies of this nature were made. How-
ever, it now emerges that Cities Service' bulk sample, the material
examined in nearly all those studies, is far from representative of the
Delta. Instead of being composed of cuts of each two-metre sample from
every hole they drilled there, as Cities planned, the bulk sample proves
to have come from a single pit dug in the uppermost metre of the Delta's
dry surface. Material from this layer is exidised, thus substantially
different from most sediment of the Delta. Within the deposit, reducing
conditions prevail. In addition, the bulk sample site lies close to the
river mouth, so sediment collected at it is considerably coarser than the
Delta average. Wind and periodic flooding have heightened this second
difference by removing much of that finer-grained material which did
occur at the site. Such skewing adversely affects results of many past
tests, those concerning percentages of useful components which can be
concentrated successfully at low cost by mechanical methods, to give an
instance. In the comparatively coarse material tested previously, copper
sulphides, for example, are largely locked. The fairly well-liberated
particles of these minerals, being smaller, occur in finer-grained
sediment offshore, and below the Delta surface – particularly in its
thickest, volumetrically most significant zone some distance beyond the
river mouth.

Cities' assays of every drillhole sample they collected do provide a
reliable picture of Delta-wide distribution and thus overall grade of
each of the six elements investigated, Cu, Pb, Zn, Co, Mo & Ag. But
those results bear little on questions of mineralogy and extractability.
Nor, as indicated, do they cover gold, an element since found to be
concentrated in all Delta sediment examined so far.

These considerations underlie the need to re-run and expand many of the
earlier tests, now using material available from Planet's auger holes.
Because of pumping losses referred to in preceding pages this sample,
also, is imperfect. Still, it appears genuinely Deltaic fractions of
it represent reasonably well all but the finest-grained components of
the inshore fifteenth of the deposit's total sediment mass.

Sample preparation

Nearly all the estimated 2.3 mt sediment retrieved from holes sunk in the
King Delta by Planet Resources' Melbourne contractors was lodged at
Mines Department laboratories in Launceston. Missing fractions of the

original total comprise small quantities of each sample or composite abstracted for chemical analysis. When Planet withdrew from the project Cottesloe Corporation inherited the sample material.

After evaluating Planet data as described in Section II of this report Cottesloe arranged for parts of all samples to be retained, parts of some to be discarded,and fractions of many to be mixed to provide the following.

1. Reference set: this comprises a 200-g portion of every successive metre or, where less was penetrated, part metre of sediment recovered from each auger hole. In a few cases material representing a single metre or part of it consists of two or more samples mixed. The field and corresponding lab numbers of the 168 specimens forming this reference collection appear in Appendix . Designations Y, B, G or P in the list indicate the category of Delta material to which Cottesloe has assigned each sample.

2. Composite KDY consists of 400-g portions of the 50 samples representing the Delta's uppermost or Y layer. Twenty kg of this material have been prepared.

3. Composite KDB consists of 400-g portions of all 53 samples representing the Delta's main layer, that commonly bracketed by Y and G layers. After mixing, 20 kg of this material was set aside for tests. The remaining 3 kg were discarded.

4. Composite KDG consists of 400-g portions of all 21 samples representing the Delta's basal or G layer. Each of these samples contains Delta and sub-Delta material, the ratio of which varies from one sample to another. Five kilograms of this material has been retained.

5. Composite KDP consists of 400-g portions of all 44 samples representing sub-Delta material. Five kilograms of it are being held.

6. Composite KDS is made up of whatever fraction remained of each of the 50 Y-layer samples after 200-g and 400-g portions were abstracted. It appears there were substantial quantities of some components of KDS, medium to small amounts of others, and nothing remaining in the case of a few. The weight of each component of KDS has been recorded.

Tests

Experiments are under way on cuts of KDY and KDB. Composites KDG and KDP in representing levels at which natural concentrations of useful components peter out are of scant interest but are being retained in case checks should be needed later. Composite KDS is held in reserve: it may be drawn upon for future tests requiring bulk sample or, for example, as a source from which to prepare several kilograms of sulphide concentrate.

Concentrations of S, Fe, Cu, Co, Au and Ba are being determined in KDY and KDB and in fifteen sub-samples derived from them. Four sub-samples of each composite represent products of sizing at 180, 75, and 10 microns, two the high- and low-density fractions resulting from a TBE separation, three more the tail, mid and con fractions derived by tabling, two the buoyant and non-buoyant products of sulphide flotation, a pair the liquor and residue of a cyanide leach, and a couple more the same products of a thiourea leach. Besides separations specified the programme involves some 150 chemical analyses.

Work is being conducted at Mines Department laboratories in Launceston and is expected to be completed early in 1990.

IV   TECHNOLOGY

More information was collected on the three principal methods, identified
last year, possibly applicable in producing elemental sulphur from King
Delta pyritic concentrate. They are the Noranda, Orkla and Outokumpu.
Each has seen use at commercial scale.

The Noranda process which until the present inventigation ranked as a
candidate proves not to warrant further consideration. It was used from
1954 to 1959 by Noranda Mines' Sulphur-Iron Division in plant near Port
Robinson, Ontario, but encountered problems and remained uneconomic.
Only a fifth of the total sulphur in pyrite was recovered in elemental
state and this material was frequently contaminated by trace metals. It
is said that the co-product, ferruginous sinter, generally was of saleable
quality (personal communication, Noranda Minerals).

The second method above was developed and used successfully by Orkla
Grube-Aktiebolag at Thamshavn, western Norway, from 1931 onwards to
treat cuprous pyrite mined at Lökken, close by. Operations ceased in
1962 when reserves at Lökken – under continuous exploitation for several
centuries, primarily as a copper source – finally ran out. The Orkla
facility could handle up to 330,000 mt annually and for many years was
yielding between 80,000 and 115,000 mt elemental sulphur per annum. The
furnace charge requires, besides pyrite, about 12% quartz, 10% coke and
2% limestone. The process recovers approximately 85% of total sulphur in
the ore, and a like percentage of its total copper. Steam from waste-
heat boilers permits co-generation of electric power. The iron component
of pyrite is converted to a silicate, of no commercial value. The process
is outlined in an article reproduced at Appendix B.

Orkla Grube-AB held rights to the process. We have not established yet
if those entered the public domain when Orkla Grube disbanded, or if they
are vested in a successor or other entity. Orkla Exolon SA of Orkanger,
near Lökken, possibly retains title to them.

The third candidate method, that devised and employed profitably by
Outokumpu Oy, is described in two additional articles attached at
Appendix B. Outokumpu used the process at its Kokkola plant to handle
a pyritic flotation concentrate produced from complex sulphides mined
nearby at Pyhäsalmi in central Finland. The process yields elemental
sulphur, sulphur dioxide gas, clean high-grade iron cinder (67% Fe, 0.3% S)
and steam to drive turbines, thence generators, for power production.
Recovery of elemental sulphur amounts of 47% of the sulphur content of
pyrite processed.

The Kokkola plant ran for about fifteen years, following start-up in 1962.
It yielded some 1,355,000 mt sulphur, and 3,765,000 mt iron cinder (ratio,
1:2.78). The smelter ultimately had a charge capacity of 480,000 mtpa
and at peak operation in 1968 produced 125,000 tons of sulphur. Produc-
tion of this elemental form was stopped in 1977 because of rising
availability of relatively low-cost Polish brimstone.

Kokkola production figures appear in Table 3, together with those
depicting the final seven years of Orkla Grube's Norwegian output.

Rights to the Outokumpu process outside Finland were held by the Lummus
Company and presumably passed to CE-Lummus when Combustion Engineering
and Lummus joined forces.

At this stage it is too early to say which of the two Scandinavian meth-
ods referred to in preceding paragraphs would prove superior in treating
King Delta pyrite, though superficially the Outokumpu appears best. If
100 mt Pyhäsalmi pyrite concentrate were processed by the Orkla method,
value of sulphur obtained at the stated recovery rate of 84% and a sulphur
price of US$125/mt, landed, would be approximately $5,300. Employing the
Outokumpu method with its 47% sulphur recovery rate and output of 2.78
times as much iron calcine as sulphur generated - the average ratio for
fifteen years' production -- amounts of these two products deriving from
an equivalent furnace charge of 100 mt would be worth $5,750, that is,
8 1/2% more than the figure resulting from use of the Orkla process.
The Outokumpu method yields SO_2 as well, containing 94% as much sulphur
as that produced in elemental form.

We lack details concerning power output associated with the Orkla process
so are unable to compare it with the rated 42 megawatts the Outokumpu
plant could generate, running at full capacity. Nor have we yet been
able to determine relative costs of reagents consumed at Lökken and
Kokkola, though, per unit, silica, coke and limestone of the Norwegian
process are likely to be cheaper in Tasmania than fuel oil required by
the Finnish one.

The Outokumpu method is the more suitable technically because it is
designed to handle small particles of pyrite, like those of the King
Delta, rather than lump pyrite with which Orkla's furnaces were charged.

For fuller evaluation of respective merits of the two processes and for
rough estimates of likely construction and running costs each may involve
our data have seen turned over to industrial chemists associated with a
major European plant constructor.

Papers of the third set included at Appendix B describe a cobalt recovery
system developed at Noranda Mines' research centre for possible applica-
tion in treatment of ore from a deposit at Blackbird, Idaho, to which the

Table 3. Production of sulphur from pyrites at two Scandinavian plants

Year	Outokumpu[1]: Finland Elemental Sulphur 1 mg tons	Iron Cinder tons	Year[3]	Orkla[2]: Norway Elemental Sulphur tons
1962	5,630	3,645	1956	95,382
1963	38,214	152,301	1957	95,149
1964	68,139	204,276	1958	89,126
1965	73,771	227,087	1959	77,111
1966	73,641	324,364	1960	71,254
1967	101,413	229,480	1961	61,156
1968	125,249	340,748	1962	45,175
1969	111,841	296,094	1963	0
1970	114,822	274,801		
1971	101,456	214,979		
1972	119,221	279,094		
1973	122,715	164,348		
1974	99,589	235,517		
1975	84,409	280,145		
1976	85,730	328,365		
1977	29,126	140,064		

[1]Source: Personal communication, Outokumpu Metallurgy, 1-VI-89

[2]Source: Personal communication, US Bu Mines, Branch of Industrial
        Minerals, 30-I-89

[3]Final seven years

company then held title. We procured the information as Blackbird ore
is a cobaltiferous pyrite and thus in one sense akin to sulphide of the
King Delta. The latter, pyrite chiefly, assays about 0.1% Co only,
however, against the 0.7% Co of Blackbird's sulphide. Some reagents of
this Noranda system such as hydrogen sulphide and sulphuric acid
possibly could be obtained from Delta pyrite destined for elemental
sulphur production by diverting a fraction of it to Klaus plant.

The fourth excerpt at Appendix B discusses recovery of cobalt from
copper leach solutions. The method may find use should it be decided to
treat Delta sulphide by hydrometallurgical means.

Information concerning the five processes above has been obtained through
or from the following: US Geological Survey; Idaho Geological Survey;
Noranda Ltd; Norwegian Consulate, New York; Norwegian Trade Commission,
Washington, DC; Orkla Exolon SA; Norwegian Geological Survey; Finnish
Embassy's Office of Science and Technology; Mines Department, Helsinki;
Outokumpu Oy; SA Chemical Institute.

V   SULPHUR

To assess the full potential of sulphur produceable from King Delta
pyrites Cottesloe Corporation is investigating output, export and import
of the commodity and its derivatives by Australia and neighbouring
countries.

The Commonwealth imports between 350,000 and 550,000 tonnes elemental
sulphur each year: all but 1% comes from Canada. Canadian sulphur is
produced from natural gas high in hydrogen sulphide. The fields yielding
this sour gas lie in Alberta.

Australia buys Canadian sulphur from four oil companies there. Of the
534,000 tonnes shipped to the Nation in 1988 Shell supplied 28%,
Cansulex 27%, Petrosul 26% and Mobil 20%.

Principal importers are

.   The Phosphate Co-operative Australia Ltd
    350 Bourke St Melbourne Victoria 3000

.   Incitec Ltd
    PO Box 140 Morningside Queensland 4170

.   Adelaide and Wallaroo Fertiliser Ltd
    25 Peel St Adelaide South Australia 5000

.   CPSB and Farmers Ltd
    133 St Georges Terrace Perth Western Australia 6000

Imported material accounts for some 60% of the Commonwealth's yearly
sulphur consumption.

Nearly all of the other 40% is obtained in the form of sulphuric acid
produced at plants treating domestically-mined sulphides of zinc and of
lead. The remainder is produced as elemental sulphur, in refining
imported crude oil. Twenty plants produce sulphuric acid, and six others
recover sulphur at oil refineries. Seventeen of the twenty-six are
situated in southeastern Australia with six in New South Wales, five in
South Australia, four in Victoria, two at Brisbane in Queensland and one
in Tasmania. Refiners involved are Amoco, Shell, Australian Oil
Refiners, and Petroleum Refineries Pty Ltd.

Ninety-five percent of the sulphur Australia uses is turned into sulphuric
acid. Approximately 1,800,000 monotonnes are produced per year. The
sulphur component comprises 32.6% of this total. Roughly two-thirds of

acid is employed in fertiliser manufacture, the rest in making other chemicals, refining petroleum, mining and processive metals, and in producing plastics, paint, rubber, paper, soap, poisons, and medicine.

Sulphuric acid also is imported, from Japan. However, the quantity never has reached 2% of Australia's annual requirement and is unlikely to increase. Sulphur in elemental form is far easier to handle and transport than acid, and on a weight basis over three times as economical.

Saudi Arabia and Iran each produce sulphur in quantity but are as distant from SE Australia as Vancouver, the port from which the Nation's present supplies of this substance are shipped. There are no closer sources.

Cottesloe's information on sulphur has been obtained from the US Bureau of Mines; The Sulphur Institute, Washington, DC; Shell Company of Canada; Cansulex, Vancouver BC; NZ Statistics; NZ Board of Trade; BMR and Australian Bureau of Statistics, Canberra.

VI   ECONOMICS

Both Orkla and Outokumpu methods of producing elemental sulphur from pyrites were used commercially in Scandinavia, as we have said. Resultant production met significant proportions of western Europe's sulphur demand.

The 31- and 15-year operational lives of the two sulphur-extraction plants concerned and the fact that some years after start-up the capacity of each was expanded testify to their profitability.

Cottesloe's research indicates that for Australia, New Zealand and most countries in the Far East, all far removed from sources of native sulphur and of elemental sulphur derived cheaply from natural gas, application of either the Orkla or Outokumpu process would be economic today, especially when savings in shipping costs are taken into account. Freight charges and associated insurance raise the price which Australia and New Zealand pay for Canadian sulphur 26 to 28%. The latter two nations and several Asian ones importing sulphur all have negative trade balances, another consideration favouring domestic production of sulphur over its purchase abroad.

A number of the countries, Australia particularly, have adequate feed-stock for such plants in the form of pyritic waste from base metal mining and processing.

At present this resource is under-utilised.

Some pyritic sulphur is used, being converted to sulphuric acid: but difficulties and hazards of conveying the chemical long distances to consumers from the logical production points at or near sulphide mines and smelters restricts the amount worth generating.

Producing elemental sulphur instead of acid would largely overcome the problem and is feasible at sites such as the King River Delta where pyritic tailings are abundant and can be concentrated fast, inexpensively.

Cottesloe has collected relevant commodity prices in effect when the Orkla and later the Outokumpu plants were running. Comparing those figures with corresponding prices of recent date shows that increases in fuel cost have remained essentially in balance with increases in value of sulphur and likely co-products. Results are tabulated below. Data were obtained from U.S. Departments of Energy and of the Interior: see Appendix C.

Table 4. Prices of sulphur and other commodities in 1958, 1968 & 1988.

Year	Sulphur US$/1t	Iron Oxide US$/1t	Copper US¢/1b	Cobalt US¢/1b	Gold US$/to	Oil US$/bbl
1958	23.00	11.70^{1960}	26.31	2.00	35.00	3.01
1968	39.00	11.05^{1970}	41.17	1.85	39.66	2.94
1988	85.95	32.53^{1987}	120.51	7.49^{1987}	438.31	12.57
			change x 1958 figure			
1968	1.56	0.94	1.56	0.93	1.13	0.98
1988	3.74	2.78	4.58	3.75	12.52	4.18

In avoiding trans-Pacific freight charges and associated insurance costs
elemental sulphur produced from pyrite in Australia could be sold at a
price 14 to 15% above that of North American sour gas and Frasch sulphur
yet still save local buyers 12 to 13% of the amount they otherwise would
have paid. And Australia's foreign spending would be cut by A$70,000,000
to $100,000,000 each year.

The King Delta deposit alone could yield some 1,250,000 mt sulphur. At
a 20-year mining rate this is sufficient to meet 11 to 18% of the amount
imported annually in recent years. A central West Coast plant drawing
on pyritic tailings from other Tasmanian sulphide mines probably could
produce twice as much, reducing the Commonwealth's present import of
sulphur by a fifth to a third.

VII   PROGRAMME

Testwork

Nature and extent of additional lab studies needed will be determined
once results of those in progress are received.

Two recently-developed means of producing concentrate perhaps worth
testing are the Kelsey separator, and the K-process, a chemical tech-
nique. The former is a type of jig-cum-table, allegedly superior to
existing equipment in recovering fine-grained gold and other minerals.
The K-process uses a bromine-base reagent to leach gold. Unlike cyanide
this reagent is not poisonous thus not an environmental or other hazard.
In some applications it is said to have cost advantages over cyanide.

Currently Cottesloe is looking into the possibility of having small
quantities of sulphide concentrate from the Delta flash-smelted by an
overseas group which developed a high-temperature furnace and operates
a bench-scale version of it.

If more sulphide concentrate is required a dozen new holes may be drilled
at the Delta to provide raw material. Ideally, holes should be bored
with a vibrocorer. Details of one available in Australia appear at
Appendix D. Maximum core length retrievable with this apparatus is only
6 metres: however there is seldom any core loss so re-sampling Planet's
auger holes Numbers 5, 6, 13, 14, 17 and 20 with such equipment would
allow amounts of gold Planet washed away to be determined. The extent
of similar losses from the uppermost 6 metres of Planet's holes 15, 16
and 19 also could be established.

Sinking a few vibrocorer holes into the Delta face would allow subsurface
distribution of gold in upper parts of that zone, also, to be studied.
As mentioned elsewhere Elisna's gold sampling of the Delta, exposed and
submerged, was surficial, and Cities' samples - though from holes pene-
trating the deposit fully - were not analysed for gold.

Participation

Following withdrawal of an Australian company which had contracted to
join in, Cottesloe is actively seeking a partner with the technical skills
and market position needed in developing the King Delta deposit. Several
agents have been appointed to assist the Corporation in this renewed hunt.

A North American oil and gas group was located wishing to secure an
option, but after investigation was judged to lack the necessary capital.
Two other companies declined to participate. Talks with a couple more
are under way.

Three categories of company are of special interest. Chief are companies

- importing elemental sulphur
- producing elemental sulphur domestically
- smelting sulphides, particularly those currently generating by-product sulphuric acid

Another target category comprises certain Australian and overseas organizations mining alluvial deposits: they have extractive skills and concentrator capability applicable to the Delta. Several such companies — notably those dredging tin in SE Asia - have suitable equipment too, some now idle.

At general level groups already endeavouring to find large mineral deposits in Tasmania are of interest, especially those searching for copper and gold.

SRMH 2/90

026

APPENDICES

506029

Auger hole samples, King River Delta

TDM Register No		Planet Resources Description	
861920	Y	KDA	001
861921	G	KDA	002
861922	Y	KDA	003
861923	B	KDA	004
861924	G	KDA	005
861925	Y	KDA	006
861926	Y	KDA	007
861927	G	KDA	008
861928		KDA	009
861929		KDA	010
861930		KDA	011
861931		KDA	012
861932		KDA	013
861933	Y	KDA	014
861934	G	KDA	015
861935		KDA	016
861936		KDA	017
861937		KDA	018
861938	Y	KDA	019
861939	Y	KDA	020

A1

Reg. No		Planet Resources
861940	Y	KDA 021
861941	Y	KDA 022
861942	Y	KDA 023 & KDA 024 Composite
861943	Y	KDA 025
861944	G	KDA 026
861945	G	KDA 027
861946	G	KDA 028
861947	G	KDA 029
861948	G	KDA 030
861949	P	KDA 031
861950	P	KDA 032
861951	Y	KDA 033
861952	G	KDA 034
861953		KDA 035
861954		KDA 036
861955		KDA 037
861956		KDA 038
861957		KDA 039 & KDA 040 Composite
861958		KDA 041
861959	G	KDA 042
861960		KDA 043
861961		KDA 044
861962		KDA 045
861963		KDA 046
861964		KDA 047
861965	B	KDA 048
861966	G	KDA 049
961967		KDA 050
861968		KDA 051
861969		KDA 052
861970		KDA 053
861971		KDA 054
861972		KDA 055
861973		KDA 056
861974	B	KDA 057
861975	G	KDA 058
861976		KDA 059
861977		KDA 060

Reg. No		Planet Resources
861978	P	KDA 061
861979	P	KDA 062
861980	R	KDA 063
861981	Y	KDA 064
861982	B	KDA 065
861983	B	KDA 066
861984	B	KDA 067, KDA 068, KDA 069 & KDA 070 Composite
861985	Y	KDA 071
861986	B	KDA 072
861987	B	KDA 073 & KDA 074 Composite
861988	Y	KDA 075
861989	Y	KDA 076
861990	B	KDA 077
861991	B	KDA 078
861992	B	KDA 079
861993	B	KDA 080
861994	B	KDA 081
861995	B	KDA 082, KDA 083, & KDA 084 Composite
861996	Y	KDA 085
861997	B	KDA 086
861998	B	KDA 087
861999	B	KDA 088
862000	B	KDA 089
862001	Y	KDA 090
862002	Y	KDA 091
862003	Y	KDA 094
862004	Y	KDA 095
862005	Y	KDA 096
862006	Y	KDA 097
862007	B	KDA 100
862008	B	KDA 101
862009	B	KDA 102 & KDA 103 Composite
862010	Y	KDA 104
862011	Y	KDA 105
862012	Y	KDA 106
862013	Y	KDA 107
862014	Y	KDA 108 & KDA 109 Composite

506032

Reg. No		Planet Resources
862015	Y	. KDA 110
862016	Y	KDA 111
862017	Y	KDA 112
862018	B	KDA 113 & KDA 114 Composite
862019	B	KDA 115
862020	B	KDA 116 & KDA 117 Composite
862021	B	KDA 118 & KDA 119 Composite
862022	B	KDA 120 & KDA 121 Composite
862023	B	KDA 122, KDA 123, KDA 124, KDA 125, & KDA 126 Composite.
862024	B	KDA 127, KDA 128, KDA 129, & KDA 130 Composite.
862025	Y	KDA 133
862026	B	KDA 134
862027	B	KDA 135
862028	G	KDA 136
862029	E	KDA 137 & KDA 138 Composite
862031	Y	KDA 140
862032	B	KDA 141
862033	B	KDA 142
862034	B	KDA 143 & KDA 144 Composite
862035	B	KDA 145 & KDA 146 Composite
862036	G	KDA 147 & KDA 148 Composite
862037	Y	KDA 149
862038	Y	KDA 150
862039	E	KDA 153
862040	E	KDA 154
862041	B	KDA 155
862042	E	KDA 156 & KDA 157 Composite
862043	E	KDA 158 & KDA 159 Composite
862044	E	KDA 160
862045	E	KDA 161, KDA 162, & KDA 163 Composite
862046	B	KDA 164, KDA 165, KDA 166, & KDA 167 Composite
862047	Y	KDA 168
862048	Y	KDA 169
862049	Y	KDA 170
862050	Y	KDA 171
862051	G	KDA 172

A4

031

Reg. No		Planet Resources
862052	G	KDA 173
862053	Y	KDA 174
862054	Y	KDA 175
862055	E	KDA 176
862056	E	KDA 177
862057	B	KDA 178
862058	G	KDA 179
862059		KDA 180
862060		KDA 181
862061		KDA 182
862062		KDA 183
862063	Y	KDA 184
862064	B	KDA 185
862065	G	KDA 186
862066		KDA 187
862067		KDA 188
862068		KDA 189
862069		KDA 190
862070	B	KDA 191
862071	B	KDA 192
862072	B	KDA 193
862073	B	KDA 194
862074	B	KDA 195 & KDA 196 Composite
862075	B ?	KDA 197
862076	G	KDA 198
862077	G	KDA 199 & KDA 200 Composite
872078	Y	KDA 201
862079	B	KDA 202, KDA 203 & KDA 204 Composite
861400	Y	KDA 92
861401	Y	KDA 93
861402	Y	KDA 98
861403	B	KDA 99
861405	B	KDA 120
861406	Y	KDA 131
861407	Y	KDA 132
861408	Y	KDA 151
861409	Y	KDA 152

Lee p.?

Lee p.

A5

THE ORE
Smelter charge of copper-bearing pyrites is mixed in this tunnel .... conveyed to the tops of furnaces, ...
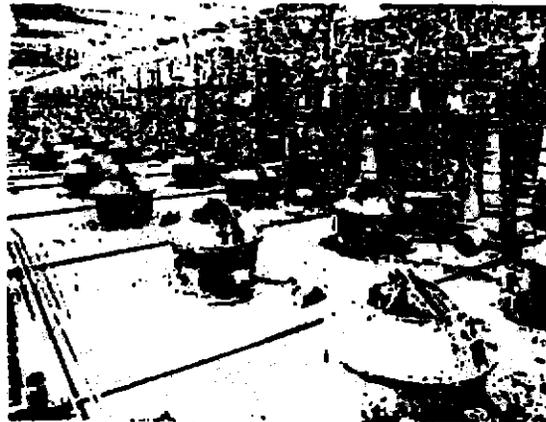
THE WASTE GASES
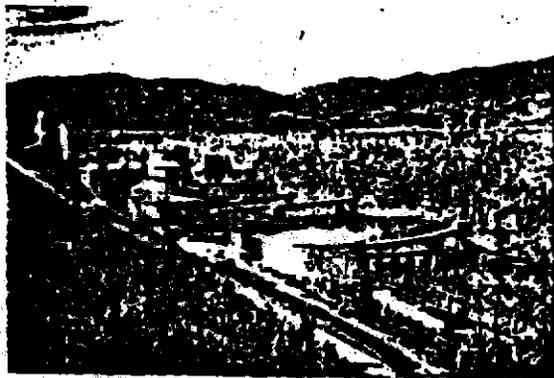Sulphur-bearing gases go through Cottrells, catalyzers ...... to be cooled in waste heat boilers ....

Smelter Gases Yield Elemental Sulphur

THORRY KIAER, Managing Director
Orkla Grube-Aktiebolag, Løkken Verk, Norway

Orkla Smelting Plant at Thamshavn which produces by-product sulphur from copper-bearing pyrite concentrates.

ELEMENTAL SULPHUR is produced as a by-product from copper-bearing pyrites at the metallurgical plant of Orkla Grube-Aktiebolag, Løkken, Norway. Average analysis of the Løkken pyrites and other metallurgical details of the process are shown in the tables on page 90. Note that sulphur content is 41%.

In brief the process operates as follows: Lump pyrite is mixed with coke and the fluxing agents, silica and limestone. This charge is smelted in water-jacketed furnaces. Part of the contained sulphur is distilled off in the upper part of the furnace, while FeS is oxidized to form FeO and SO_2 in the smelting zone of the furnace. The SO_2 in the smelting zone is reduced to sulphur and CO_2 in the furnace shaft with the aid of coke which is used only as a reducing agent, and not a fuel.

FeO reacts with the silica and limestone to form a slag containing approximately 80% $2(FeO) \cdot SiO_2$ and about 15% $CaO \cdot FeO \cdot 2(SiO_2)$. The copper plus some of the FeS forms a matte running about 12% copper. This slag-matte melt is separated in forehearths illustrated in the accompanying sketch. The slag contains about 42% Fe, mainly as silicates.
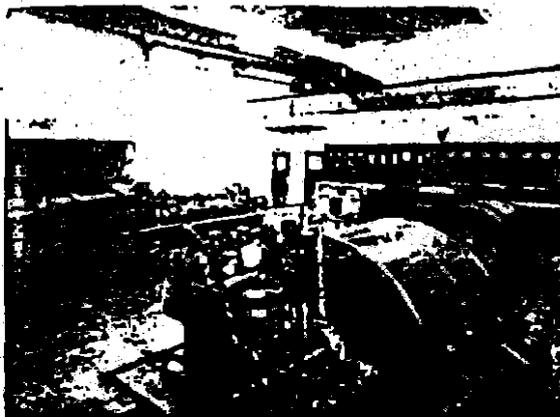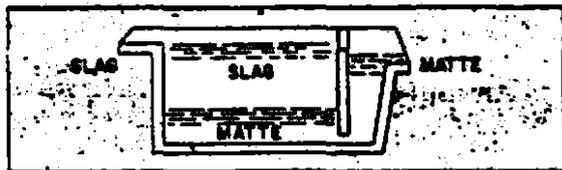
88

... charged through these bells, ... and smelted in this battery of furnaces.


... yielding power in these turbo-generators, ... and sulphur which is granulated and stored here.

at Orkla-Grube Plant in Norway

Sketch of forehearth used to separate slag and matte

Gases leaving the top of the furnace contain about 300g sulphur per cu m (temperature 450 deg C), CO_2, and various sulphur compounds in the form of SO_2, H_2S, COS, and CS_2. These gases are cleaned in dust flues and then passed through a catalyzer which promotes the following reactions:

$$SO_2 + 2H_2S \rightarrow 2H_2O + 3S$$
$$CS_2 + SO_2 \rightarrow CO_2 + 3S$$
$$2COS + SO_2 \rightarrow 2CO_2 + 3S$$
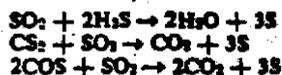
Gases leaving the catalyzer are cooled from 450 deg C to 130 deg in waste-heat boilers. Most of the sulphur is

condensed in the boilers. Then the gases are washed the injection of liquid sulphur in scrubbers, reheated in heat exchangers to about 300 deg C, and passed through a second catalyzer. Sulphur formed in the second catalyzer is precipitated in cooling towers, and scrubbed with liquid sulphur. SO_2 contained in the tail gases is absorbed in towers filled with limestone, and sprayed with sea water to prevent damage to surrounding vegetation. In a final step the sulphur is washed with milk of lime to remove arsenic, and then granulated or molded.

Matte produced in the furnace is upgraded to about 40% copper by remelting with coke, silica, and limestone. Conventional converters and electrolytic circuits are used to produce refined copper. In the overall process, copper recovery is about 84 to 85% and sulphur, about 85 to 86%.

Sulphur production at Orkla now totals 115,000 tons per year; matte production, 15,000; and copper production, 6,000. The company's first plant was built in 1931 at Thamshavn, about 30 km from Løkken. In 1936 plant capacity was doubled, and it now is able to process 330,000 tons of ore per year. Similar plants are licensed in Spain, Portugal, and Russia. (Please turn page)

Table I. Composition of Furnace Charge and Intermediate Products

	Tons	Cu %	Fe %	S %	SiO₂ %	Zn %
Pyrites	29 980	1.866	38 149	41 018	12 95	1 922
Quartz	4 826				93 00	
Limestone (95% CaCO₃)	1 837					
Coke (11% ash, 16% H₂O)	4 144					
Lean matte	4 591	11 355	54 00	24 00		
PRODUCTS:						
Rich matte	1 364	37 44		24 24	0 04	
Slag from matte furnace	4 426	0 50		2 13	29 53	
Slag from pyrite furnace	21 555	0.27		2 67	34 33	
Dust from gas mixer	105			21 06		
Dust from Cottrells	132			15 22		

Table II. Exit Gases From Furnaces

		g per cu m		
Gases	Percent by Vol.	g per cu m	C	S
S		300		
CO₂	13 2		71 0	
CO	0 2		1 1	
O₂	0 5			
SO₂				24 92
H₂S				11 42
CS₂		24 8	7 1	21 70
COS		22 3	4 0	14 83

Table III. Determination of the Volume of Gas Per Ton of Pyrites Smelted

	Kg of Carbon per ton Pyrites
Carbon charged to the furnaces as coke	100 9
Carbon charged in limestone	7 0
Total carbon charged to furnaces	107 9

Carbon in Exit Gases	g Carbon per cu m
CO₂, 13.2 per cent by volume	71 0
CO, 0.2 per cent by volume	1 1
COS	7 1
CS₂	4 0
Total	83.2

The volume of gas per ton pyrites is:
$$\frac{107.9 \times 1000}{83.2} = 1300 \text{ cu m}$$

Average Analysis of Lökken Ores In Percent

S	41.0
Cu	1.5 - 2.5
Fe	38.0
Zn	1.9
Co	0.1
SiO₂	12 - 16
Al₂O₃	0.3
CaO	0.8
MgO	0.3
As	0.05
Se	0.005
Ag	20. g/ton
Au	0.2 g/ton

Table IV. Sulphur Balance

	Kg per ton pyrites
Charged sulphur	410
Sulphur combined with Cu, Zn, etc.	−15
Sulphur in FeS₂	= 395
Volatile sulphur (42% in FeS₂)	− 166
Sulphur in FeS	= 229
Sulphur combined with Cu, Zn, etc.	+ 15
Fixed sulphur entering smelting zone	= 244

Sulphur in matte $\dfrac{1.364 \times 24.24}{29.980 \times 1000} = 11.2$

Sulphur in slag $\dfrac{669.800}{29.980} = 22.3$

Sulphur in dust $\dfrac{62.100}{29.980} = 2.1$ → − 35.5

Sulphur oxidized to SO₂ = 208.5

Sulphur as SO₂ not reduced in the gases from the furnaces: $\dfrac{24.92 \times 1.300}{1.000} = -32.4$

Sulphur produced by reduction of SO₂ = 176.1

Slag and Matte Analyses In Percent

	Slag	Matte
Cu	0.3	38.50
Fe	42.0	32.46
S	2.3	25.33
SiO₂	33.5	0.14
Zn	−1.6	---
CaO	4.5	---
Al₂O₃	1.4	---
MgO	0.13	---
Co	---	0.11

Major Equipment Installed in the Orkla Plant

8 water-jacketed furnaces
19 Cottrells 60,000 volt
8 first catalyzer-chambers
4 boilers
6 gas scrubbers (with liquid sulphur)
2 heat exchangers (with oil burners)
8 second catalyzer-chambers
8 gas cooling towers (circulating sulphur)
2 gas scrubbers filled with Raschig rings
2 Cooling towers (seawater injection)
9 towers for limestone
Steam turbine station with superheater, turbines, and generators.

The above tables pertain to the feed, products, by-products, and intermediary products involved in the production of elemental sulphur at Orkla Grube-Aktiebolag's smelter Thamshavn, Norway, pages 88 and 89.

FLOW DIAGRAM of elemental sulphur recovery at the Kokkola smelter. The numbers refer to processes as follows:

1. Drying of flotation pyrite concentrate.
2. Combustion air preheater.
3. Flash smelting furnace (reaction No. 1).
4. Waste heat boiler.
5. Electrostatic dust precipitator.
6. Gas reheater.
7. Catalyzer No. I (reaction No. 2).
8. Gas cooler.
9. Catalyzer No. II (reaction No. 3).
10. Sulphur scrubber.
11. Liquid sulphur handling and cooling.
12. Autoclaves for arsenic removal.
13. Sulphur prilling and storage tower.
14. Liquid sulphur tank.
15. Gas scrubber.
16. Matte granulator.

$$2 FeS_2 + \Delta = 2FeS + S_2$$
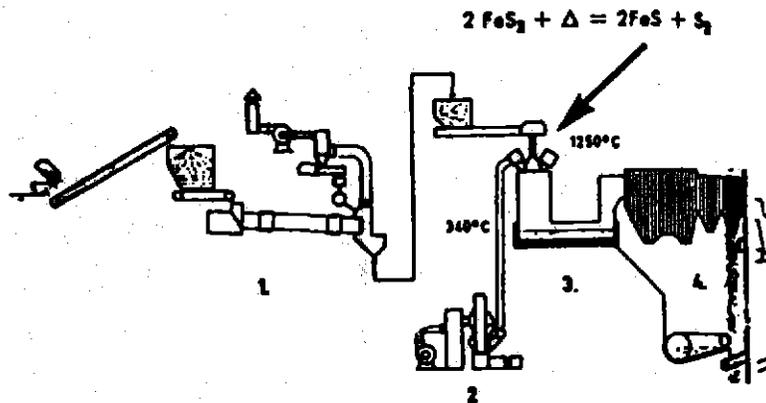
OUTOKUMPU adds second catalyzer to raise

by George O. Argall, Jr.
Editor
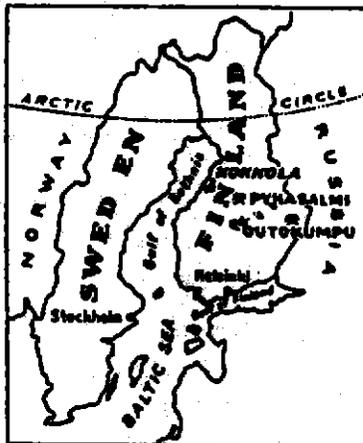
PLACE-FIX map showing location of the Kokkola sulphur smelter and the Pyhasalmi mine, source of pyrite feed.

With the continuing high demand for elemental sulphur and the currently higher price ($27.00-$32.00 per ton Gulf Coast mine, United States, and $47.00 per ton in Scandinavia), Outokumpu Oy's new pyrite to elemental sulphur process is of ever growing interest.

Outokumpu Oy, Finland's largest mining and metallurgical company and long a world leader in pyrometallurgical smelting developments has operated its Kokkola smelter since August 1962.

The smelter has been an outstanding success—both technically and financially. Designed and built to furnace 360,000 metric tons per year of pyrite flotation concentrate, feed capacity is now 480,000 annual tons with only very minor modifications to original equipment and little addition of new equipment. Even more important has been the increase in total sulphur recovery (elemental sulphur and SO$_2$ gas). The new smelter has been in operation using the revised flowsheet only two months, during which time the sulphur recovery has been 91 percent. When every part of the equipment is made to operate at optimum conditions, the sulphur recovery is expected to increase.

The virtual doubling in sulphur price in Finland during the smelter's operation has created a very high cash flow to the company. Only a few additional men are employed, despite the 30 percent increase in furnace charge. It must be pointed out that the smelter proved itself commercially well before the increase in sulphur prices.

What is the Kokkola Process?

Flash smelting of 480,000 tons per year of pyrite flotation concentrate

18

$$4H_2S + 2SO_2 = S_6 + 4H_2O$$

$$2CO + SO_2 = \tfrac{1}{2}S_2 + 2CO_2$$

$$COS + H_2O = H_2S + CO_2$$
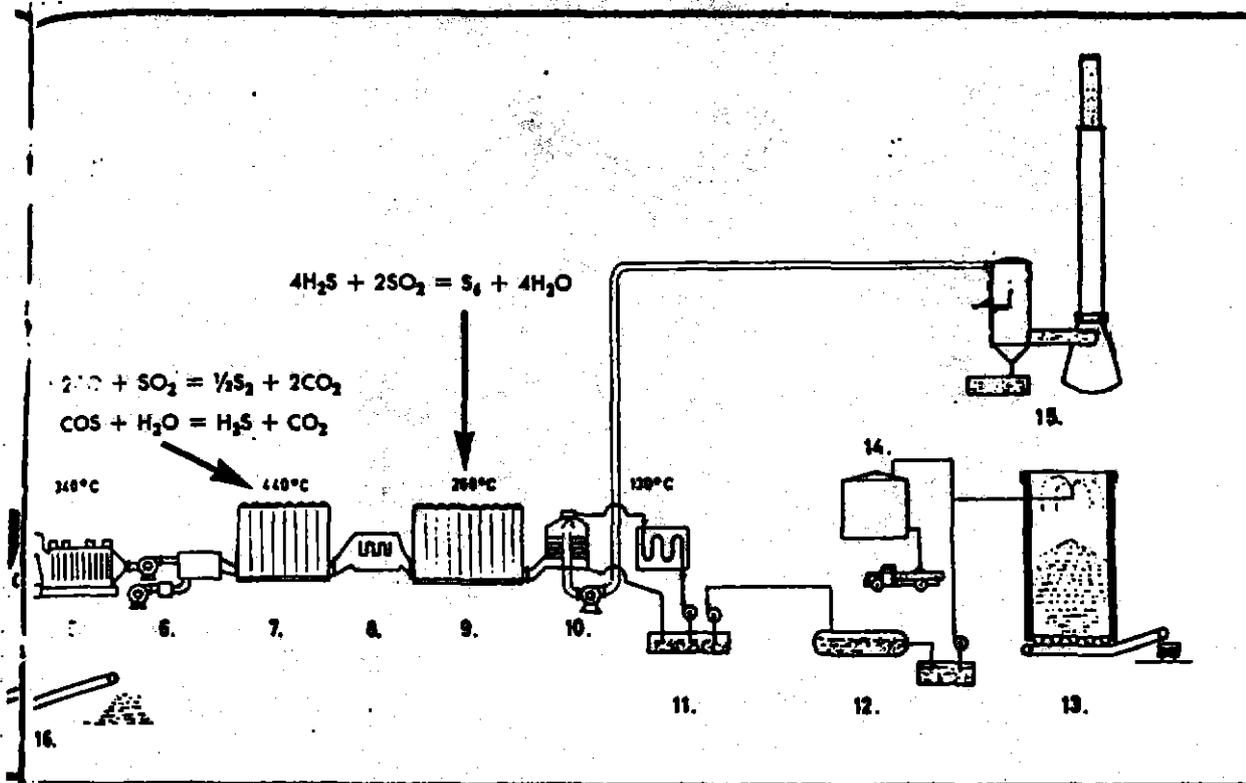
pyrite-to-sulphur conversion to 91 percent

(50.5 percent S) to produce, with other process steps:

* 1. 113,500 tons of sulphur in form of prills or liquid sulphur (99.99 percent S).

* 2. 107,000 tons per year of sulphur in SO_2 gas (8.2 percent sulphur dioxide in gas).

* 3. 325,000 tons of coarse iron oxide calcine (67.0 percent Fe and 0.3 percent S) which is sold as a high grade iron blast furnace feed.

* 4. Continuous energy in the form of process steam which generates 42 megawatts (nominal) of electricity.

While elemental sulphur is the key product of smelting, the co-products make the operation commercially feasible. For example, the electrical engineer would term the smelter a thermal electric power plant burning pyrite and Bunker C oil to produce electricity with byproducts sulphur, sulphuric acid, and iron oxide.

Kokkola's Market Advantages

It must be remembered that this smelter was built to make profit out of the massive deposit which was discovered by Outokumpu's geologists in late 1958 near Pyhäsalmi, central Finland, 90 miles east of Kokkola. Development soon indicated 17,000,000 tons assaying 0.85 percent copper, 2.8 zinc, 33.0 iron, and 37.0 percent sulphur.

Under the then prevailing copper and zinc prices it would be necessary to derive revenue from Fe and S (pyrite) content if the deposit was

to become ore. Direct sales to European pyrite burners proved to be financially impracticable.

Outokumpu's metallurgists were developing processes for pyrite with the goal of producing saleable sulphur and iron cinders. The process was developed and proved in 1961, in a 1.0 ton per hour pilot plant at Pori, Finland.

The commercial smelter was built near Kokkola, fairly close to the pyrite deposit, near several pulp mills that were importing elemental sulphur, and 75 miles south of the iron blast furnace of Rautaruukki Company at Raahe. This company buys the Kokkola iron cinders for furnace feed.

Electric power was immediately saleable at a good price, 1.24 cents per kilowatt hour. In fact, Kokkola power is used at an important saving in cost by Outokumpu at its own

2.
Catalyser I (at 400-450°C.)
$2CO + SO_2 = 1/2S_2 + 2CO_2$
$COS + H_2O = H_2S + CO_2$

3.
Catalyser II (at 240-260°C.)
$4H_2S + 2SO_2 = S_6 + 4H_2O$

4.
Sulphide matte roasting
$2FeS + 3 1/2O_2 = Fe_2O_3 + 2SO_2$

Flash Smelting Furnace

Using experience gained by development and operation of the Outokumpu copper flash smelting furnace, the Outokumpu metallurgists built the furnace reaction shaft 17.5 feet in diameter, 33 feet high, and lined the top with water cooled chrome magnesite refractory bricks. Incidentally, there have been no special problems with refractories despite a calculated combustion flame temperature of about 2,000°C. They are checked and replaced as needed during the annual smelter repair period.

The open bottomed reaction shaft is built on top of and at one end of a horizontal reverberatory shaped rectangular furnace in which the molten matte collects and is continuously tapped off.

Concentrate is dried to zero moisture in a rotary kiln with an inlet gas temperature of 900°C. (fuel oil is used). Temperature at discharge end is 120°C. Dust is recovered in dry cyclones and combined with the dried concentrate.

The dried pyrite is elevated to the top of the furnace and blown into the furnace through tuyeres installed in the roof. Three oil burners, operating with about 340°C. preheated combustion air, burn Bunker C oil in an amount equal to about 12.5 percent of the weight of the pyrite feed. Volume of air is carefully regulated to close to the stoichiometric amount. The combustion gases must be oxygen-free to prevent sulphur from burning to SO_2. Preheating of combustion air in an oil treated heat exchanger has been an important factor in raising capacity above the initial furnace operation. Vapors, hot combustion gases, and molten particles, drop through the shaft. The molten matte collects in the bottom of the horizontal furnace while the gases sweep over the molten pool to the exhaust flue. Only about five percent of feed is carried through furnace as dust by gas stream; 25 percent of this is removed in the waste heat boiler and the rest of it in the electrostatic precipitator.

The temperature of the furnace off-gases is about 1,250°C. while the

---

smelters, refineries and mines, including the Outokumpu copper mine in eastern Finland. See "Evolution of underground mining methods at Outokumpu" by Esko Pihko, WORLD MINING, April 1966, pages 40 to 43.

Roaster SO_2 gas could be sold to a nearby contact sulphuric acid plant making acid for a fertilizer factory.

Immediately nearby markets were thus available for all products. Unit sales values were high and shipping charges minimal.

The elemental sulphur is an especially marketable product. It is pure, uniformly sized, and easy to handle. By no means are sales possibilities for sulphur limited to nearby pulp mills. However, one is so close that its own tank truck transports liquid sulphur on a daily basis. Kokkola now produces 70 percent of Finland's requirements of elemental sulphur.

What about Furnace Feed?

The process is based on the fact that it is possible to distill off one of the two sulphur atoms of the pyrite molecule by heating it sufficiently. This means that other sulphides like pyrrhotite cannot be used for this purpose. To get a fairly good recovery of sulphur and iron, it is of course desirable to have the gangue components as low as possible, thus avoiding a great slag amount. At Kokkola, the slag is negligible and mostly joins the matte, but can, however, be tapped separately if required.

In direct pyrite roasting it is not

possible to get rid of the arsenic. In the Outokumpu Process the arsenic does not cause any troubles; it is almost completely volatilized in the smelting furnace and recovered during sulphur purification. Easily volatilizable metals also are lead and zinc. A part of them is collected in the flue dust from the furnace. The copper and the precious metals go into the matte, further to the iron sinter, from which it is possible, if needed, to recover them by a chlorination treatment of the sinter, for example.

Carbonaceous pyrite has also been tested on a pilot plant scale very successfully.
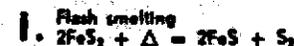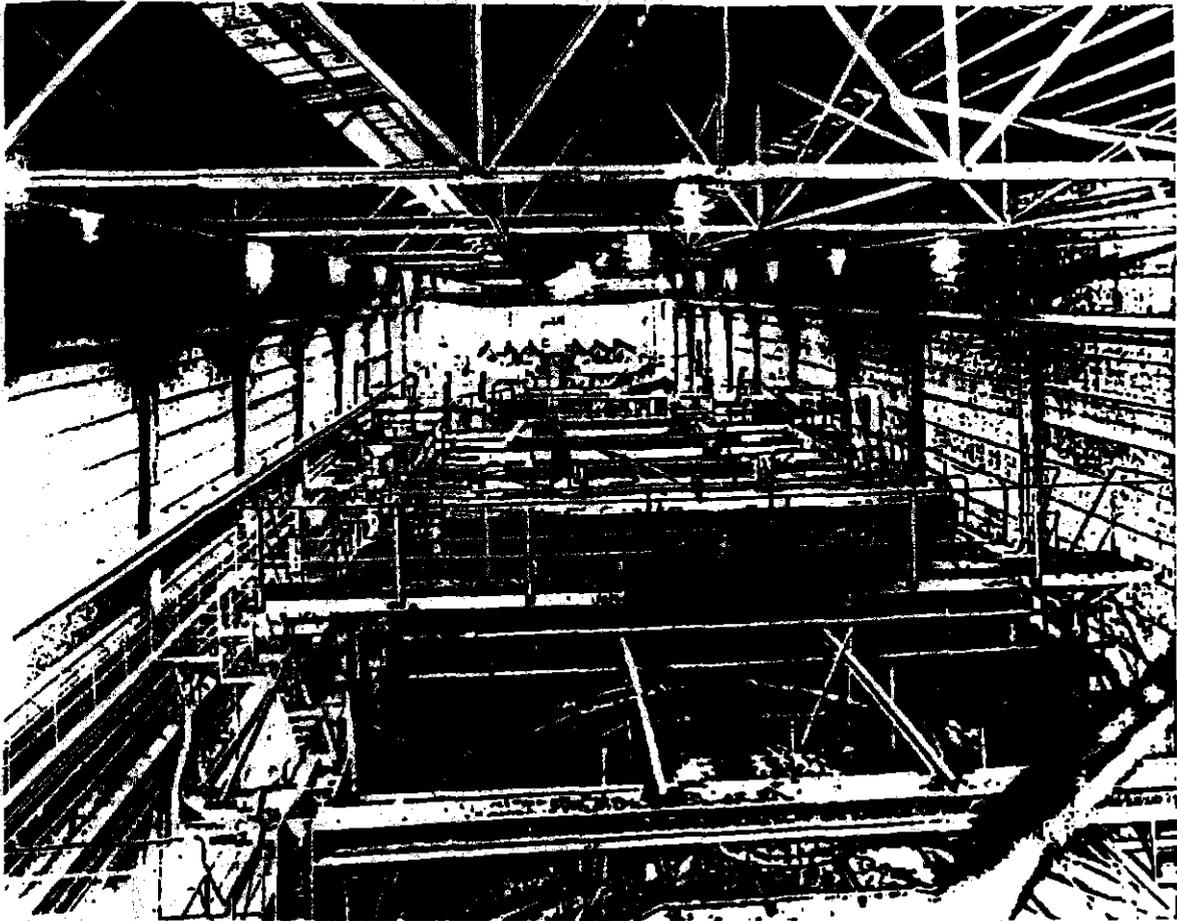
How the Process Works

Pyrite concentrate is dried and blown into the top of a flash smelting furnace to decompose pyrite to form elemental sulphur (vapor) and an iron sulphide matte (liquid).

Sulphur vapor and combustion gases are cooled, cleaned from dust, and treated in catalysers for heat recovery in the last sulphur condensating step.

The liquid matte is continuously tapped from the bottom of the furnace, granulated with water, and roasted to produce SO_2 gas and Fe_2O_3 calcine.

The key equations for the process are:

1.
Flash smelting
$2FeS_2 + \Delta = 2FeS + S_2$

20

INSIDE THE FURNACE BUILDING looking from furnace over the top of the sulphur gas treating equipment. In the foreground is the waste heat boiler and electrostatic precipitators. At the far end the first catalyser. No. 2 catalyser is outside building to right.

molten matte has about 50 to 100°C. lower temperature. The matte is continuously tapped from the furnace, granulated with jets of water, and stockpiled for further treatment in the roaster. Gangue in small amounts, mostly silica, in original furnace feed forms slag which is normally tapped together with the matte.

The hot gases, including sulphur vapor, pass through a radiation water wall boiler and a convection boiler where steam at 70 atmospheres pressure and a temperature of 285°C. is generated. This steam must be superheated in an oil burning heater to a temperature of 500°C. at 65 atmospheres pressure before it enters the turbogenerator in the power plant.

The high pressure steam boiler uses regular boiler tubes. By using a pressure of 70 atmospheres the tube wall temperature is kept over the dew point of sulphur vapor, but under the limit for sulphur corrosion on the tubes.

Gases leave the convection boiler at a temperature of 350°C. and pass through a flue into an electrostatic precipitator which removes all the dust carried over.

Second Catalyser Ups Recovery

Originally there was only one catalyser operating at a temperature of about 260°C. After some years of research and test work it was found out that the application of a second catalyser at a temperature slightly over 400°C. installed ahead of the original one would raise the sulphur recovery considerably. Therefore, a new oil fired reheater has recently been placed in the process circuit to raise temperature of the gas stream after the electrostatic precipitator to 400°C. After the reheater, the gases are led to the new catalyser and to a new gas cooler (a low pressure steam boiler) to cool the gases before entering the low temperature (260°C.) catalyser. The new equipment was placed in operation in early December

1966. It was built on the north side outside the main furnace building while the smelter operated with the original circuit. After it was completed the furnace was stopped and the new units were piped into the gas circuit.

Actual condensation of elemental liquid sulphur from the furnace gas starts in the low temperature boiler which acts as a cooler to remove heat from the gas stream. Condensation of sulphur occurs on the boiler tubes from which it drops to the bottom of the boiler.

The next step in the process is catalytic reaction to combine the SO_2 and H_2S in the gas into elemental sulphur. This takes place at about 260°C. and the temperature is high enough to keep the resultant sulphur in a gaseous phase. See reaction No. 3 above. Alumina soft concrete is used for catalysts. The crushed catalyst is placed in the catalyser shafts and the gases flow through it at atmospheric pressure.

SULPHUR purification takes place in these autoclaves. Arsenic is removed with milk of lime.

After catalytic action the sulphur separation is achieved by reducing gas temperatures in a cooling tower. Molten sulphur is pumped through a cooler where it is cooled to 120°C. with the generation of low pressure steam at a pressure of 0.6 atmosphere. The cooled sulphur is then sprayed into the top of the cooling tower and drops through a rising stream of hot gases. Sulphur in the gases condenses and falls to bottom of tower from where it flows to a collecting tank which feeds both the sulphur heat recovery and cooling tower circuit as well as the downstream autoclave.

After the furnace gases have been cooled by the molten sulphur they are further cooled by Baltic Sea water in a wash tower where gas temperature is reduced from 130°C. to 60°C. with the separation of a small additional amount of sulphur from the gases. The sulphur is separated from the sea water in a settling tank and remelted to the main sulphur collecting tank. The cooled gases are emitted to atmosphere through a 460-foot high stack.

Arsenic Removal from Sulphur

The small amount of arsenic, 0.04 percent, in the Pyhäsalmi pyrite concentrate is volatilized in the furnace and follows the sulphur through the precipitation steps. The molten sulphur from the collecting tank is pumped through an autoclave where it is washed with preheated milk of lime which strips the arsenic from the sulphur in the form of a soluble calcium-thio-arsenate.

The purified, arsenic free, molten sulphur is then pumped directly to steam heated storage tanks for shipment to local pulp mill or is pumped to the top of a prilling tower where it is discharged as droplets through special nozzles. These droplets solidify into prills before they reach the bottom of the tower. The prills are then stored in bulk, awaiting shipment.

Roasting Artificial Pyrrhotite

Remember that the flash furnace matte is FeS (63 percent Fe and 31.5 S) which was granulated and stockpiled outside the roaster building.

Granulated matte is reclaimed from stockpiles and conveyed to the top of roaster building where it continuously fills three feed bins above the roaster. Actually there are four identical 162 square foot each Lurgi fluidized bed roasters. Three are now in use, but the fourth will also be used when the new nearby cobalt recovery plant is in operation. These roasters have probably the highest capacity in the world, 3.0 long tons of feed per day per square foot of bed. They operate at a much higher temperature than is necessary for normal pyrite roasting. Close temperature control is the most important operational factor. At 1,000°C. there is not enough reaction velocity and the furnace will be extinguished (No.

4 equation): at 1,050°C. there is sintering of the granular feed. Temperature is controlled by regulating roaster feed.

There is practically no degradation of particle size during roasting. Calcine is less than 0.5 percent minus-200-mesh, but 100 percent is minus-10-millimeters in size. Sulphur content is 0.20 percent, iron is 67.0 percent. Coils in the fluid bed produce high temperature steam.

The sulphur dioxide-bearing roaster gas is cooled to 320°C. in a waste heat boiler and then the dust is removed by cyclones and electrostatic precipitation. The cool dust free SO_2 gas with a total sulphur content of 107,000 tons per year from roasting 340,000 tons of matte is piped to an adjacent sulphuric acid plant. The heat in the roaster calcine is recovered during calcine cooling in water cooled Redler conveyors. In addition to solving many operational problems, successful heat transfer control, abrasion damage, water proofing of Redler cooling jackets, and sinter handling have been solved by the roasting department.

The superheated steam which is recovered from the roasting process is fed directly to the turbine of the power plant. The roasting plant produces about 40 percent of the total steam power in the entire smelter.

Firm electric power is sold so that the power plant has two auxiliary oil fired boilers which are used when insufficient process steam is available. Nominal output is 42 megawatts of which about five are used in the Kokkola plants.

With the December 1966 increase in Gulf Coast sulphur prices by $2.50 per ton the Kokkola process is again being closely evaluated by many mining companies, both those that have a pyrite byproduct from base metal ores and those companies mining pyrite ore bodies for their sulphur content. Especially interested are the Japanese, Australian, Indian, Brazilian, Canadian, and United States companies which have shown keen interest in this process.
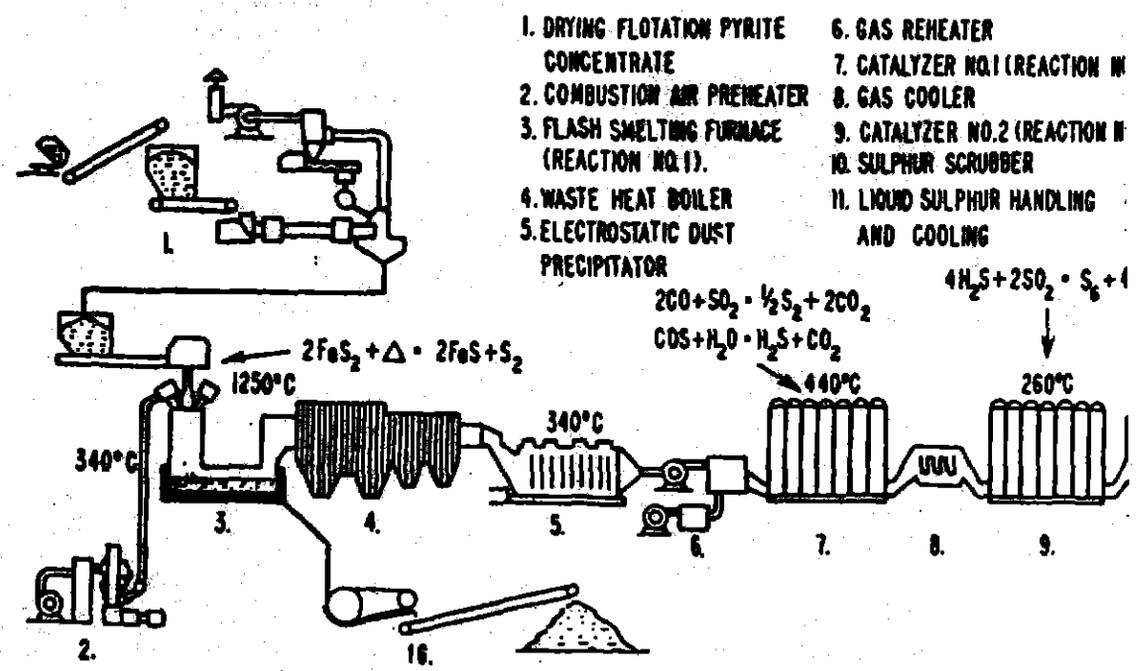
Outokumpu's sulphur staff includes: Jorma Honkasalo, the company's metallurgical director with headquarters in Helsinki; Kauko Kaasila, chief of the process engineering department in Helsinki; Heikki Tanner, Kokkola plant manager; Esko Nermes, sulphur plant superintendent; and Osmo Vartiainen, superintendent of the iron roasting plant.   END

22

Producing Country	1961	1962	1963	1964	1965	1966	1961	1962
World pyrite production in long tons						**Estimated**		
COMMONWEALTH COUNTRIES								
United Kingdom	164	26,988	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Rhodesia	58,442	90,456	65,055	51,129	n.a.	n.a.	22,200	19,200
Canada (c)	461,837	461,982	425,391	314,132	341,229	289,489	228,014	229,539
Cyprus (a)	824,771	809,263	957,940 (r)	673,786	977,829	947,444	398,000	388,000
Australia (b)	213,423	148,566	194,059	220,070	204,811	253,050	182,100	65,100
TOTAL	1,558,637	1,497,155	1,600,000	1,300,000	1,600,000	1,800,000	—	—
Irish Republic	49,321	37,229	—	—	—	—	n.a.	n.a.
OTHER COUNTRIES								
Bulgaria (b)	119,785	140,056	127,880	144,530	150,501	n.a.	50,000	59,000
Czechoslovakia (b)	363,000	395,000	342,000	355,000	369,000	346,000	141,000	153,000
Finland (b)	270,043	467,171 (r)	532,556 (r)	542,561	576,184	900,000	114,000	118,000
France (d)	200,930	239,153	248,325	188,319	132,239	86,685	117,769	124,592
Germany (Federal)	501,475	380,717	351,813	419,088	438,984	444,015	194,057	148,133
Germany (East)	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	40,000	41,000
Greece	184,917	141,891	110,886 (r)	113,141	n.a.	n.a.	86,000	66,000
Italy	1,555,229	1,558,970 (r)	1,390,224	1,373,812	1,379,396	1,283,046	716,000	721,000
Norway	721,808	797,077	710,052	708,060	687,675	666,794	319,000	355,000
Poland	198,000	219,000	213,000	230,000	n.a.	n.a.	76,000	82,000
Portugal	642,609	631,164	592,559	567,088	606,657	548,043	295,600	290,336
Roumania	259,000	300,000	328,000	403,000	n.a.	n.a.	103,000	120,000
Spain	2,096,873	2,095,432	1,995,226	2,355,421	2,386,100	2,388,635	1,001,000	997,000
Sweden	431,322	371,683	398,260	445,239	434,363	426,800	220,424	188,967
USSR (g)	2,756,000	2,953,000	3,149,000	3,149,000	3,250,000	3,250,000	1,457,000	1,565,000
Yugoslavia (e)	358,000	407,000	350,000	421,000	401,000	372,000	143,000	163,000
Algeria	46,085	42,000	37,154	55,304	55,078	40,729	22,000	19,000
Morocco	13,855	20,417	22,777	20,885	18,029	14,661	5,000	7,000
South Africa	439,751	434,090	411,943	425,647	421,532	473,587	176,000	174,000
Cuba	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
U.S. (f)	987,309	915,890	824,809	847,493	874,957	872,414	398,519	379,046
China (g)	984,000	1,083,000	1,181,000	1,279,000	1,480,000	1,480,000	443,000	492,000
Formosa	46,717	44,772	46,021	45,582	38,640	41,342	17,500	16,800
Japan (h)	2,476,924	2,539,396	2,514,762	2,728,846	2,823,000	n.a.	1,132,112	1,170,121
Korea (South)	773	—	37 (r)	90	168	21,234	300	—
Korea (North) (g)	295,000	344,000	384,000	415,000	445,000	445,000	118,000	138,000
Philippine Republic	50,776	55,400	57,138	43,336	108,830	113,000	24,000	27,000
Turkey	96,063	105,389	95,548	111,582	138,972	170,788	45,000	51,000
TOTAL*	16,300,000	16,900,000	16,600,000	17,600,000	18,200,000	18,000,000	—	—
WORLD'S TOTAL*	17,900,000	18,400,000	18,200,000	18,900,000	19,800,000	20,000,000	8,200,000	8,400,000 (r)

Outokumpu Oy's flash smelting technique is based on distilling off one

1. DRYING FLOTATION PYRITE CONCENTRATE
2. COMBUSTION AIR PREHEATER
3. FLASH SMELTING FURNACE (REACTION NO. 1).
4. WASTE HEAT BOILER
5. ELECTROSTATIC DUST PRECIPITATOR
6. GAS REHEATER
7. CATALYZER NO.1 (REACTION N
8. GAS COOLER
9. CATALYZER NO.2 (REACTION N
10. SULPHUR SCRUBBER
11. LIQUID SULPHUR HANDLING AND COOLING

$$2CO + SO_2 \cdot \tfrac{1}{2}S_2 + 2CO_2$$
$$COS + H_2O \cdot H_2S + CO_2$$
$$4H_2S + 2SO_2 \cdot S_6 + $$

$2FeS_2 + \triangle \cdot 2FeS + S_2$

1250°C

340°C

340°C

440°C

260°C

sulphur content of the pyrite in long tons

1963	1964	1965	1966
n.a.	n.a.	n.a.	n.a.
24,709	30,800	n.a.	n.a.
210,197	154,627	170,000	149,000
442,300	324,000	474,000	456,000
94,600	94,600	88,900	111,000
—	—	—	—
54,000	61,000	63,000	n.a.
133,000	140,000	142,000	133,000
224,000	260,000	280,000	260,000
106,785	78,434	55,887	38,676
145,474	178,843	183,392	191,651
43,000	41,000	43,000	53,000
52,000	53,000	n.a.	n.a.
640,000 (r)	640,000	640,000	590,000
123,000	317,000	313,000	297,000
85,000	87,000	n.a.	n.a.
272,577	274,991	279,062	252,580
131,000	160,000	n.a.	n.a.
950,000	1,117,000	1,131,000	1,131,000
202,161 (r)	227,524	221,475	220,800
1,673,000	1,673,000	1,720,000	1,720,000
140,000	168,000	160,000	149,000
17,000	28,000	26,000	19,000
7,000	6,000	5,000	4,000
152,000	164,000	158,000	179,000
n.a.	n.a.	n.a.	n.a.
343,568	353,831	353,645	355,512
531,000	575,000	665,000	668,000
17,300	17,100	14,900	18,560
1,163,690 (r)	1,261,439	1,350,000	n.a.
17	27	n.a.	n.a.
157,000	167,000	177,000	177,000
27,000	20,000	48,000	56,000
46,000	51,000	60,000	78,000
8,400,000	8,700,000	9,200,000	9,000,000

two S atoms existing in pyrite

12. AUTOCLAVES FOR ARSENIC REMOVAL
13. SULPHUR PRILLING AND STORAGE TOWER
14. LIQUID SULPHUR TANK
15. GAS SCRUBBER
16. MATTE GRANULATOR

Outokumpu Oy's (OO) new process for obtaining elemental sulphur from pyrite is also of interest. The Lummus Co. has the rights to license these inventions on a worldwide basis, with the exception of Scandinavia.[24]

OO's Kokkola smelter in Finland has used this process since 1962, and has operated in excess of its furnace design capacity of 360,000 metric tpy.

The Kokkola process is a flash smelting technique whereby OO smelts 480,000 tpy pyrite flotation concentrate (bearing 50.5% S). Based on this feed, the plant produces about 114,000 tpy of sulphur in prill or liquid form analyzing 99.94% S, 107,000 tpy of sulphur in SO_2 gas (8.2% sulphur dioxide in gas), 325,000 tpy coarse iron calcine (67.0% Fe plus 0.3% S) which is sold as high-grade blast furnace feed, and 42 megawatts of continuous electrical energy derived from process steam.

The byproduct S is sold to several nearby paper pulp mills, the Fe calcine to the Rautaruukki Co. at Rache, 75 miles to the north, the electric power is sold at 1.24¢ per kilowatt-hour to neighboring industry, and the roaster SO_2 gas is sold to a contact sulphuric acid plant.

This process is based on the principle that it is possible to distill off one of the two S atoms of pyrite molecules if heated high enough. Other sulphides including pyrrhotite cannot be utilized in this process. Another advantage of Outokumpu's process—arsenic does not create a problem for it is almost completely volatilized during smelting and recovered during sulphur purification. Carbonaceous pyrite has also been treated successfully.

The pyrite concentrate is first dried, then blown in the top of a flash smelter to decompose the feed to sulphur vapor and iron sulphide, and liquid matte.

The vapor phase is cleaned and then treated in a catalyzer. Liquid matte is tapped continuously, granulated with a water quench and roasted. When matte is roasted, SO_2 gas and Fe_2O_3 calcine is produced. In the furnace, about 12.5% of the weight of pyrite feed is added as Bunker C oil with close oxygen control.

Hot gases issuing from the furnace at 1,250° C (including sulphur) pass through a radiation boiler followed by a convection boiler where steam at 70 atmospheres of pressure and 285° C temperature is generated. This high pressure prevents the tube wall temperatures from reaching the dewpoint of sulphur vapor, but above the limit for sulphur corrosion. Gases leave the convection boiler at 350° C and then pass through an electrostatic precipitator.

Current practice at Kokkola uses two catalyzers following the electrostatic precipator. The first operates at 400° C, the second at 260° C.

The next step reacts SO_2 and H_2S at about 260° C with an alumina soft concrete catalyst to form elemental S.

After catalytic processing, sulphur is collected by reducing gas temperatures so sulphur will liquefy. This liquid sulphur is sprayed into a cooling tower through a rising stream of hot gases. Sulphur condenses and is collected in tanks which feed the sulphur heat recovery and cooling tower circuit as well as the following autoclave.

Furnace gases are cooled again by Baltic Sea water in a wash tower from 130° to 60° C, and a smaller amount of S is recovered. The S is then separated from sea water.
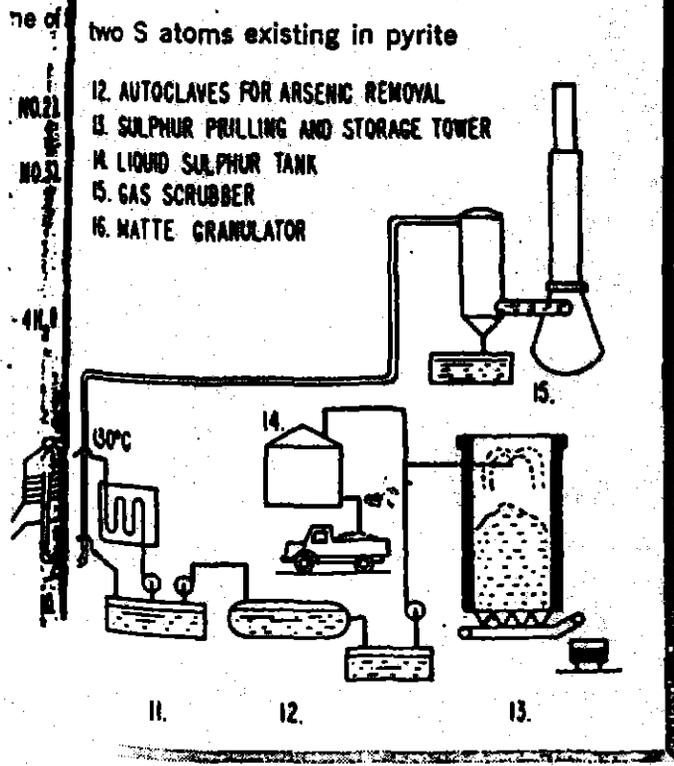
HYDROMETALLURGICAL TREATMENT OF BLACKBIRD COBALT CONCENTRATE

G.B. Harris, S. Monette and R.W. Stanley

Noranda Research Centre
240 Hymus Boulevard
Pointe Claire, Quebec
Canada, H9R 1G5

A hydrometallurgical process has been developed for the production of cathode cobalt from Blackbird cobaltite-pyrite concentrate, based upon the selective dissolution of cobaltite. Laboratory and miniplant studies have shown that overall cobalt recoveries of 93 to 94% can be obtained by: leaching concentrate with 30 g/L Na_2SO_4 solution at 150°C, 1200 kPa oxygen and 30% solids; treatment of the leach solution for the removal of impurities (Fe, As, Cu, Zn, Ni, etc.) deleterious to cobalt electrowinning; precipitation of cobaltous hydroxide from the purified solution with lime; dissolution of cobaltous hydroxide in spent electrolyte; and cobalt electrowinning from sulphate medium.

The reasons for adopting the pressure leach and the preferred operating parameters are discussed. Data are presented on the effect of temperature, oxygen pressure and sodium sulphate concentration. The requirements for solution purification are outlined and the results of an integrated batch miniplant operation are given.

BULK CONCENTRATE

WASH LIQUOR

OXYGEN → PRESSURE LEACH ← Na_2SO_4 WASH

LIME

JAROSITE PRECIPITATION → L/S SEPARATION → RESIDUE

WASH

HYDROGEN SULPHIDE → Cu/As PRECIPITATION → L/S SEPARATION → RESIDUE (Cu/As SULPHIDE)

LIMESTONE

OXYGEN → Fe OXIDATION AND PRECIPITATION → L/S SEPARATION → RESIDUE

ELUANT H_2SO_4 → ZINC ION-EXCHANGE → ELUATE TO PRECIPITATION WITH LIME

PRIMARY ELUATE

ELUANT H_2SO_4 → NICKEL ION-EXCHANGE → SECONDARY ELUATE TO PRECIPITATION WITH SODA ASH

LIME

COBALT PRECIPITATION → L/S SEPARATION → FILTRATE

H_2SO_4

COBALT DISSOLUTION → L/S SEPARATION → RESIDUE (GYPSUM)

SPENT

CARBON COLUMN

COBALT ELECTROWINNING → EVAPORATION

Refining Flowsheet
Figure 24

Acknowledgement

References

1. Whyte, R.M., et al; "Development of a process for the recovery of electrolytic copper and cobalt from Rokana converter slag", Advances in Extractive Metallurgy 77, IMM, London, 1977, p. 57.

2. Brummitt, R.N., "Cobalt production in Central Africa", paper presented at Cobalt 80, CIM 10th Annual Hydrometallurgical Meeting, Edmonton, Alberta, October 1980.

3. Aird, J., Celmer, R.S. and May, A.V., New cobalt production from R.C.M.'s Chambishi roast-leach-electrowin process, Mining Magazine 140 (10), October 1980, p. 320.

4. Mackiw, V.N. and Benz, T.W., "Application of pressure hydrometallurgy to the production of metallic cobalt", p. 503, in Extractive Metallurgy of Copper, Nickel and Cobalt, Paul Queneau (Ed.), Interscience, New York, 1961.

5. Thornhill, P.G. Wigstol, E. and Van Weert, G., "The Falconbridge matte leach process", Journal of Metals 23 (7), July 1971, p. 13.

6. Elchardus, E.M. and Reynaud, J.F., "Le Cobalt en France", Chimie et Industrie, 86 (5), 1961, p. 531.

7. Chilton, C.H., "Leach licks arsenic bugaboo in metal ore", Chem. Eng., 65 (1), 1958, p. 80.

8. Mitchell, J.S., "Cobalt pressure leaching and reduction at Garfield", Journal of Metals 9 (3), March 1957, p. 343.

9. Warren, I.H., "The generation of sulphuric acid from pyrite by pressure leaching", Australian J. Applied Science, 7, 1956, p. 346.

10. Strong, B. and Henry, R.P., "The purification of cobalt advance electrolyte using ion exchange", Hydrometallurgy, 1 (4), 1976, p. 311.

11. Rosato, L., Harris, G.B. and Stanley, R.W. "Separation of nickel from cobalt in sulphate medium by ion exchange", presented at 111th AIME Annual Meeting, Dallas, Texas, 14-18 February 1982.

Report of Investigations 9190

Recovery of Cobalt From Spent Copper Leach Solutions—Improved Elution and Impurity Removal Techniques, With Revised Process Economics

By P. G. Bennett and T. H. Jeffers

045

TABLE 4. - Estimated annual operating costs

	Annual cost	Costs per pound electrolytic cobalt
Direct costs:		
Raw materials:		
Sulfuric acid at $40/st	$1,453,500	$1.518
Ammonia at $0.14/lb	91,100	.095
Sodium carbonate, 58 pct at $83/st	3,122,500	3.261
Dextrose at $0.26/lb	457,700	.478
Ion-exchange resin at $600/ft³	944,500	.986
DEHPA solvent at $3.60/gal	21,500	.022
Cyanex 272 solvent at $15/gal	89,600	.094
Chemicals for steamplant H_2O treatment	5,900	.006
Total	6,186,300	6.460
Utilities:		
Electric power at $0.047/kW·h	535,500	.559
Process water at $0.25/Mgal	59,500	.062
Raw water at $0.05/Mgal	1,000	.001
Natural gas at $6.00/MMBtu	1,615,600	1.687
Total	2,211,600	2.309
Direct labor:		
Labor at $9/h	636,500	.665
Supervision, 15 pct of labor	95,500	.100
Total	732,000	.765
Plant maintenance:		
Labor	297,000	.310
Supervision, 20 pct of maintenance labor	59,400	.062
Materials	297,000	.310
Total	653,400	.682
Payroll overhead, 35 pct of above payroll	380,900	.398
Operating supplies, 20 pct of plant maintenance	130,700	.136
Total direct cost	10,294,900	10.750
Indirect cost, 40 pct of direct labor and maintenance	554,200	.579
Fixed cost:		
Taxes, 1 pct of total plant cost	186,100	.194
Insurance, 1 pct of total plant cost	186,100	.194
Depreciation, 10-yr life	1,960,800	2.047
Total operating cost	13,182,100	13.764
Credit:		
Copper-nickel residue at $0.79/lb	3,127,100	3.265
Zinc carbonate at $0.24/lb	2,313,800	2.416
Nickel carbonate at $3.10/lb	2,808,700	2.933
Cobalt-Manganese carbonate at $2.70/lb	52,400	.055
Total	8,302,000	8.669
Net operating cost	4,880,100	5.095

SUMMARY AND CONCLUSIONS

A Pachuca air-lift reactor reduced the overall resin inventory of a cobalt recovery process by 27 pct and the elution resin inventory by 72 pct when compared with a fixed-bed elution column. Total elution time was reduced from about 180 min using the fixed-bed column to approximately 50 min utilizing the Pachuca reactor. Six 5-min contacts employing one 10-g/L H_2SO_4 scrub, two 40-g/L H_2SO_4 eluant contacts with a 250-g/L H_2SO_4 eluant as the second contact of the fourth elution, one water wash, two 4N NH_4OH eluant contacts with a 20-pct bleed stream, and one 2-min water wash effectively eluted the loaded resin. Approximately 18 min was required for eluant transfer, eluant drainage, and resin drainage at the end of a completed elution.

Processing fresh cementation plant effluent resulted in product eluates containing approximately 1 g/L Fe; whereas, those obtained from aged feed solution contained 6 g/L Fe. Lower iron concentrations in the eluates reduced reagent consumption for impurity removal. Total sodium carbonate consumption was reduced by 40 pct and dextrose consumption by 70 pct. Resin inventory was reduced by 20 pct using the eight-compartment MCIX column.

Utilizing the eight-compartment MCIX loading column along with the Pachuca reactor for resin elution resulted in a decrease in the total resin inventory by 47 pct; thus, a reduction in the capital cost. The processing of fresh feed solution lowered iron concentrations in the eluates, thus reducing the reagent cost for iron removal. Implementing the Pachuca elution system and processing fresh feed solution resulted in a net operating cost of $5.10/lb of cobalt. This value included byproduct credits.

REFERENCES

1. Jeffers, T. H., and M. R. Harvey. Cobalt Recovery From Copper Leach Solutions. BuMines RI 8927, 1985, 12 pp.

2. Jeffers, T. H., K. S. Gritton, and P. G. Bennett. Recovery of Cobalt From Spent Copper Leach Solution Using Continuous Ion Exchange. Paper in Recycle and Secondary Recovery of Metals, ed. by P. R. Taylor, H. V. Sohn, and N. Jarrett (Proc. Int. Symp. on Recycle and Secondary Recovery of Metals, TMS-AIME, Ft. Lauderdale, FL, Dec. 1-4, 1985). Metall. Soc. AIIME, 1985, pp. 609-621.

3. Jeffers, T. H., K. S. Gritton, P. G. Bennett, and D. C. Seidel. Recovery of Cobalt From Spent Copper Leach Solution Using Continuous Ion Exchange. BuMines RI 9084, 1987, 17 pp.

4. Kirk, W. Cobalt. Sec. in BuMines Mineral Commodity Summaries, 1986, pp. 38-39.

5. Hallet, C. J., A. J. Monhemium, and D. G. C. Ropertson. Paper in Extractive Metallurgy '81. Inst. Min. and Metall., London, 1981, pp. 308-320.

6. Geldard, D., and L. D. Mann. The Introduction and Application of Carbon-in-Pulp in a Heap Leach Project. Australasian. Inst. Min. and Metall., v. 7, 1982, pp. 13-15.

7. Jose, J. M., J. L. Merino, and P. Gasos. The Use of Pachuca Tanks in Hydrometallurgy. Rev. Metal. (Madrid), v. 3(1), 1967, pp. 48-57.

8. Clocte, F. L. D. The Relix Process for the Carbon-in-Pulp Recovery of Uranium. J. S. Afr. Inst. Min. and Metall., v. 3, 1981, pp. 66-73.

9. Grinstead, R. R. Selective Absorption of Copper, Nickel, Cobalt, and Other Transition Metal Ions From Sulfuric Acid Solutions With the Chelating Ion Exchange Resin XFS 4195. Hydrometallurgy (Amsterdam), v. 12, 1984, pp. 387-400.

10. Weaver, J. B., and H. C. Bauman. Cost and Profitability Estimation. Sec. 25 in Chemical Engineers' Handbook, ed. by R. H. Perry and C. H. Chilton. McGraw-Hill, 5th ed., 1973, p. 47.

TABLE 11

LAKE ERIE BASE PRICES OF IRON ORE[1] AND PELLETS AND COMPOSITE PRICE OF PIG IRON

Year	Old Range Nonbessemer	Mesabi Regular[2]	Mesabi Nonbessemer	High Phosphorus	Pellets[3]	Pig Iron[3]
1910 ...	$ 4.20	$ 4.75	$ 4.00	....	—	$ 17.25
1915 ...	3.00	3.45	2.80	....	—	12.75
1920 ...	6.70	7.20	6.55	$ 6.35	—	40.00
1925 ...	4.40	4.40	4.25	4.15	—	21.00
1930 ...	4.65	4.65	4.50	4.40	—	18.50
1935 ...	4.65	4.65	4.50	4.40	—	18.50
1940 ...	4.60	4.60	4.45	4.35	—	23.00
1945 ...	4.80	4.70	4.55	4.55	—	25.00
1950 ...	7.95	7.85	7.85	7.70	—	46.50
1955 ...	10.25	10.25	10.10	10.00	—	56.50
1960 ...	11.70	11.60	11.45	11.45	—	66.50
1965 ...	10.80	10.70	10.55	10.55	25.2	63.50
1970 ...	11.05	10.95	10.80	10.80	26.6	66.00
1975 ...	18.75	18.65	18.50	18.50	47.2	187.67
1980 ...	28.50	28.50	28.50	28.50	73.66	203.00
1981 ...	32.78	32.53	32.25-32.53[5]	32.25-32.53	80.50	203.00
1982 ...	32.78	32.53	32.25-32.53	32.25-32.53	80.50-86.9	213.00
1983 ...	32.78	32.53	30.03-31.53[7]	30.03-31.53	80.50-86.9	213.00
1984 ...	[6]	32.53	30.03-31.53	30.03-31.53	80.50-86.9	213.00
1985 ...	....	32.53	30.03-31.53	30.03-31.53	58.0-86.9[8]	213.00
1986 ...	....	32.53	30.03-31.53	30.03-31.53	58.0-86.9	213.00
1987 ...	....	32.53	30.03-31.53	30.03-31.53	37.344-86.9[8]	213.00

[1]Based on following analysis: 1910-24: Bessemer 55% Fe (Nat.): Nonbessemer 51.50% Fe (Nat.). 1925-date: Bessemer 51.50% Fe (Nat.) and 0.045 Phos (Dry); Nonbessemer 51.50% Fe (Nat.).

[2]Prior to 1980, prices are for Mesabi Bessemer.

[3]Cents per unit of iron natural.

[4]Composite price based on average prices for basic and foundry pig iron at representative producing points in the U.S.

Individual sellers quote structure premiums and penalties, as well as phosphorus premiums. A typical coarse ore premium is 80 cents per ton and a typical fine ore penalty is 45 cents per ton.

[5]$32.25—Cleveland-Cliffs as of 4/27/81; $32.53-U. S. Steel as of 6/30/81.

[6]37.344¢—U.S. Steel, 5/20/87, FOB Mount Iron, Minn. **46.84¢—Inland Steel Mining, 1/1/87. 58.0¢—Mineral Services, 8/13/85. 59.4¢—Pickands Mather, 8/1/85, delivered hold of vessel, upper Lake ports. ***72.45¢—Cleveland-Cliffs, 12/30/86; Oglebay Norton, 3/1/87. 86.9¢—Hanna Mining, 2/26/82.

[7]Effective 8/1/83.

[8]No longer being produced. Listing discontinued by Iron Age, effective February 6, 1984.

Source: Iron Age, September 1988, p. 74.

TABLE 12

ORE PRICES FOR VARYING IRON CONTENT CALCULATIONS OF LAKE ERIE SELLING VALUES
(According to formula adopted in 1923)

Standard Lake Erie selling values for iron ore, as quoted in trade journals and ore sales contracts, are per gross ton of 2,240 pounds, delivered at rail of vessel at Lower Lake Ports and are based on the following classification and guaranteed base analyses:

Old Range Bessemer	51.50% Iron Natural,	.045% Phosphorus Dry
Old Range Nonbessemer	51.50% Iron Natural	
Mesabi Bessemer	51.50% Iron Natural,	.045% Phosphorus Dry
Mesabi Nonbessemer	51.50% Iron Natural	
High Phosphorus	51.50% Iron Natural,	+ .180% Phosphorus

Price Adjustments for Iron Content Above or Below the Guarantee. All Grades:

Selling values of ores of different iron content than the base ores are determined as follows: The base price is divided by 51.50, the number of units in the base ore. The resulting quotient is the base unit value, used to determine additions to or subtractions from the base price, for iron contents above or below the basis analysis, as follows:

When less than 51.50 and not less than 50.00% Iron: from the base price deduct, for each unit or fraction of a unit of iron less than 51.50% iron, at the rate of the base unit value.

When less than 50.00 and not less than 49.00% Iron: from the price computed for 50.00% iron deduct, for each unit or fraction of a unit of iron less than 50.00% iron, at the rate of one and one-half times the base unit value.

When less than 49.00% Iron: from the price computed for 49.00% iron deduct, for each unit or fraction of a unit of iron less than 49.00% iron, at the rate of two times the base unit value.

When over 51.50% Iron: to the base price add, for each unit or fraction of a unit of iron more than 51.50% iron, at the rate of the base unit value.

Price Adjustment for Phosphorus:

All ores containing .045% phosphorus, or less, are classed as Bessemer. Phosphorus content lower than .045% commands a premium, determined in accordance with the standard table of phosphorus values. All ores containing more than .045% phosphorus are classed as Nonbessemer. Ores containing more than .180% phosphorus are classed as High Phosphorus.

Penalties:

In addition to the standard deductions applied for iron contents of less than 50%, which are computed as above, arbitrary penalties are also exacted for high silica and for fine structure.

Premiums for Lump Structure and High Manganese Content:

Hard ores of high iron, low silica contents are often sold as lump grade, generally being priced as Old Range Nonbessemer plus premiums for lump structure.

Ores containing in excess of 5% natural manganese are recognized as standard manganiferous iron-ores and are generally priced as Old Range Non-bessemer on the combined natural iron and manganese content, plus a premium for the natural manganese in excess of 5%. Ores containing between 2% and 5% of natural manganese are also sometimes marketed as manganiferous at prices which recognize some small value for the manganese content.

Premiums for lump structure and high manganese content vary and are determined by negotiation between buyer and seller.

IRON ORE MANUAL

1988-89

Published By

The TEX Report Co.,Ltd.

★ FOB Price Changes of Major Brands in European Market

(Per Fe 1%/DMT FOB)

Ore Bland	Jan-Dec Nautical Miles:Sailing days	1982	1983	1984	1985	1986	1987	1988 Price	1988 88/87(%)
●Fines									
LKAB(1) (Sweden)	1.150	¢36.20	¢31.60	¢29.20	¢29.75	¢28.60	¢25.95	¢26.70	△ 2.89
LKAB(2) (Sweden)	1.150	-	-	-	-	-	¢25.25	¢26.00	△ 2.97
SNIM(1) (Mauritania)	2.216	¢34.00	¢30.75	¢27.75	¢28.95	¢28.60	¢26.00	¢25.10	▲ 3.46
SNIM(2) (Mauritania)	2.216	-	-	-	-	¢28.00	¢24.90	¢24.40	▲ 2.01
Granada (Spain)		-	-	-	-	-	¢19.40	¢18.90	▲ 2.58
Mt.Wright (Canada)	3.101	¢33.00	¢29.30	¢26.80	¢26.80	¢26.50	¢24.03	¢23.68	▲ 1.46
Carol (Canada)	3.101	¢33.00	¢29.30	¢26.80	¢26.80	¢26.50	¢24.03	¢23.68	▲ 1.46
Lamco (Liberia)	3.710	¢30.00	¢27.00	¢24.00	¢25.10	¢24.70	¢22.75	¢21.00	▲ 7.69
CVRD (Brazil)	5.044	¢32.50	¢29.00	¢26.15	¢26.56	¢26.26	¢24.50	¢23.50	▲ 4.08
Carajas (Brazil)		-	-	-	-	¢27.50	¢25.30	¢24.30	▲ 3.95
Iscor (S.Africa)	6.200	¢31.40	¢27.90	¢20.60	¢23.50	¢22.70			
Robe River (Australia)	11.400	¢18.00	¢16.92	¢14.00	¢14.75	¢14.75	¢14.10	¢12.75	▲ 9.57
Mt.Newman (Australia)	11.400	¢40.40	¢34.90	¢32.90	¢34.30	¢32.40	¢29.35	¢31.35	△ 6.81
Hamersley (Australia)	11.400	¢40.40	¢34.90	¢32.90	¢34.30	¢32.40	¢29.35	¢31.35	△ 6.81
P.Orinoco (Venezuela)		-	-	¢32.26	¢34.00	¢32.17	¢29.20	¢30.90	△ 5.82
●P·F									
MBR (Brazil)	-	¢26.00	¢23.20	¢20.92	¢21.25	¢21.25	¢19.60	¢18.80	▲ 4.08
Samarco (Brazil)		¢26.00	¢23.20	¢20.92	¢21.25	¢21.25	¢19.60	¢18.80	▲ 4.08
●Lump									
SNIM (Mauritania)	2.216	-	-	-	-	-	¢26.40	¢25.60	▲ 3.03
Iscor (S.Africa)	6.200	¢35.90	¢31.30	¢24.00	¢29.00	¢26.70	-		
Mt.Newman (Australia)	11.400	¢44.75	¢38.15	¢36.15	¢38.48	¢36.20	¢33.15	¢36.00	△ 8.60
Hamersley (Australia)	11.400	¢44.75	¢38.15	¢36.15	¢38.48	¢36.20	¢33.15	¢36.00	△ 8.60
●Pellets									
LKAB (Sweden)	1.150	¢50.20	¢41.00	¢38.60	¢38.60	¢38.15	¢41.15	¢46.35	△12.60
Sydvaranger (Norway)	1.150	¢49.00	¢40.50	¢37.95	¢38.50	¢38.00	¢38.90	¢42.75	△ 9.90
CVRD (Brazil)	5.044	¢47.50	¢39.00	¢36.00	¢36.00	¢35.60	¢36.70	¢40.35	△ 9.95
Samarco (Brazil)		¢46.00	¢37.30	¢34.00	¢34.00	¢34.00	¢35.55	¢39.08	△ 9.93
Carol (Canada)	3.101	¢43.64	¢38.75	-	¢36.50	¢36.50	¢37.15	¢39.95	△ 7.54
Mt.Wright (Canada)		-	-	-	¢36.50	¢36.50	¢37.15	¢39.95	△ 7.54

Notes : 1) ¢ = Price per Fe 1% CIF Rotterdam.
  2) Swedish fine ore price(1) is of MAF and (2) is of KBF.
  3) Robe River fine ore ¢ Price per Fe 1%/DLT.
  4) SNIM fine ore price(1) is of TZF(Tazadit fine) and (2) is of Guelb fine.
  5) ▲ = Minus, △ = Plus.

COPPER PRICES (In cents per pound) 1/

YEAR	U.S. PRODUCER CURRENT$	1987$	YEAR	U.S. PRODUCER PRICES CURRENT $	1987 $	LME HIGH GRADE CURRENT $	YEAR	U.S. PRODUCER PRICES CURRENT $	1987 $	LME HIGH GRADE CURRENT $
1850	22.00	N.A.	1900	16.19	231.02	16.25	1949	19.50	97.50	22.25
1851	16.00	N.A.	1901	16.19	229.75	14.79	1950	21.58	106.31	22.37
1852	22.00	N.A.	1902	11.00	156.95	11.60	1951	24.50	115.02	---
1853	22.00	N.A.	1903	13.20	188.46	12.84	1952	24.50	112.90	---
1854	22.00	N.A.	1904	12.00	102.76	13.03	1953	29.05	132.05	32.25
1855	27.00	N.A.	1905	15.60	222.75	15.36	1954	29.84	134.26	31.20
1856	27.00	N.A.	1906	19.30	275.60	19.30	1955	37.51	162.30	43.03
1857	25.00	N.A.	1907	20.00	285.60	19.22	1956	42.00	175.73	41.03
1858	23.00	N.A.	1908	13.20	188.64	13.25	1957	30.17	122.15	27.36
1859	22.00	N.A.	1909	13.11	187.29	12.99	1958	26.31	104.40	24.79
1860	23.00	N.A.	1910	12.00	181.41	12.02	1959	30.99	120.12	29.80
1861	22.00	N.A.	1911	12.55	176.70	12.38	1960	32.34	122.97	30.81
1862	22.00	N.A.	1912	16.40	222.70	16.13	1961	30.32	114.42	28.73
1863	34.00	N.A.	1913	15.52	212.60	15.08	1962	31.00	114.30	29.33
1864	47.00	N.A.	1914	13.31	177.47	13.32	1963	31.00	112.73	29.25
1865	39.20	N.A.	1915	17.47	223.97	15.67	1964	32.35	115.54	30.99
1866	34.20	N.A.	1916	28.46	323.41	25.09	1965	35.36	123.21	58.52
1867	25.40	N.A.	1917	29.19	272.00	27.04	1966	36.00	121.21	69.04
1868	23.00	N.A.	1918	24.60	214.61	24.90	1967	38.10	124.92	51.19
1869	24.20	N.A.	1919	18.19	133.75	18.26	1968	41.17	128.66	56.13
1870	21.20	381.21	1920	17.50	160.70	16.22	1969	47.43	140.33	66.24
1871	24.10	345.04	1921	12.65	95.03	12.12	1970	58.07	162.66	62.98
1872	35.60	510.63	1922	13.55	111.07	12.49	1971	52.00	138.17	48.49
1873	28.00	401.66	1923	14.75	116.14	13.60	1972	51.44	131.90	48.53
1874	22.00	315.01	1924	13.20	106.24	12.66	1973	59.40	141.31	80.86
1875	22.70	325.48	1925	14.30	113.60	13.50	1974	77.27	168.35	93.13
1876	21.00	300.91	1926	14.05	111.51	12.79	1975	64.16	127.30	56.00
1877	19.00	272.04	1927	13.05	106.10		1976	69.63	129.90	66.30
1878	16.60	237.61	1928	14.61	119.44	14.67	1977	66.77	116.73	59.44
1879	18.00	265.74	1929	18.35	147.90	16.03	1978	66.51	108.50	61.86
1880	21.40	305.73	1930	13.23	109.34	12.06	1979	93.33	139.72	90.07
1881	18.20	260.29	1931	8.37	78.09	7.90	1980	102.42	140.64	99.25
1882	19.10	273.20	1932	5.70	59.00	7.05	1981	85.12	106.53	79.46
1883	16.50	236.11	1933	7.20	76.63	6.26	1982	74.31	87.42	67.17
1884	13.00	185.00	1934	8.66	83.27	6.93	1983	79.26	89.76	72.13
1885	10.80	154.30	1935	8.80	83.77	7.09	1984	68.16	74.49	62.54
1886	11.10	150.46	1936	9.71	91.60	8.60	1985	66.97	71.09	64.66
1887	13.60	196.92	1937	13.30	120.63	12.23	1986	66.85	68.24	62.53
1888	16.60	239.02	1938	10.22	92.91	9.04	1987	82.50	82.50	80.88
1889	13.50	192.75	1939	11.20	103.70	8.00	1988	120.51	116.55	117.90
1890	15.60	222.63	1940	11.53	104.82	11.35				
1891	12.80	182.69	1941	12.00	102.58	11.35				
1892	11.60	185.55	1942	12.00	98.00	11.35				
1893	10.80	154.03	1943	12.00	93.75	11.35				
1894	9.50	135.52	1944	12.00	92.31	11.35				
1895	10.70	152.46	1945	12.00	90.23	11.35				
1896	10.80	153.92	1946	14.04	85.09	14.12				
1897	11.29	170.93	1947	21.27	113.14	23.86				
1898	12.03	170.93	1948	22.32	111.04	24.50				
1899	16.70	237.98								

1/ Wirebar prices to 1977. Cathode, 1978-1988.
Sources: U.S.B.M. Yearbooks, Quin's Metal Hand-
book, Engineering & Mining Journal, American
Metal Market, Metals Week.

| | Official Price | | Market Price | | | | | |
| | | | Average | | High | | Low | |
	Dollars per Kilogram	Dollars per Troy ounce	Dollars per Kilogram	Dollars per Troy ounce	Dollars per Kilogram	Dollars per Troy ounce	Dollars per Kilogram	Dollars per Troy ounce
April 2, 1792- June 27, 1834----	623.40	19.39+						
June 28, 1834- Jan. 17, 1837---	665.20	20.69-						
Jan. 18, 1837- Jan. 30, 1934----	664.56	20.67+						
Jan. 31, 1934- March 15, 1968---	1,125.28	35.00						
1968----------------	1,125.28	35.00	1,287.96	1/40.06	1,376.37	1/42.81	1,207.26	1/37.55
1969----------------	1,125.28	35.00	1,334.58	41.51	1,422.67	44.25	1,139.74	35.45
1970----------------	1,125.28	35.00	1,170.61	36.41	1,268.35	39.45	1,131.71	35.20
1971----------------	1,125.28	35.00	1,326.22	41.25	1,427.49	44.40	1,216.90	37.85
1972----------------	1,221.73	38.00	1,884.03	58.60	2,265.02	70.45	1,429.10	44.45
1973----------------	1,221.73	38.00	3,144.66	97.81	4,065.46	126.45	2,068.90	64.35
1974----------------	1,357.40	42.22	5,135.76	159.74	6,261.35	194.75	3,761.63	117.00
1975----------------	1,357.40	42.22	5,192.02	161.49	5,972.00	185.75	4,155.48	129.25
1976----------------	1,357.40	42.22	4,029.13	125.32	4,528.43	140.85	3,313.13	103.05
1977----------------	1,357.40	42.22	4,768.27	148.31	5,415.79	168.45	4,187.63	130.25
1978----------------	1,357.40	42.22	6,222.77	193.55	7,814.23	243.05	5,343.45	166.20
1979----------------	1,357.40	42.22	9,886.35	307.50	16,621.92	517.00	6,981.53	217.15
1980----------------	1,357.40	42.22	19,694.25	612.56	27,328.11	850.00	15,519.15	482.70
1981----------------	1,357.40	42.22	14,777.76	459.64	19,266.32	599.25	12,578.97	391.25
1982----------------	1,357.40	42.22	12,085.78	375.91	15,464.50	481.00	9,540.73	296.75
1983----------------	1,357.40	42.22	13,631.91	424.00	16,372.76	509.25	12,045.27	374.65
1984----------------	1,357.40	42.22	11,595.48	360.66	13,048.37	405.85	9,899.21	307.90
1985----------------	1,357.40	42.22	10,213.00	317.66	10,973.04	341.30	9,151.70	284.65
1986----------------	1,357.40	42.22	11,839.18	368.24	14,098.09	438.50	10,503.64	326.70
1987----------------	1,357.40	42.22	14,401.92	447.95	16,115.55	501.25	12,587.33	391.51
1988----------------	1,357.40	42.22	14,091.98	438.31	15,605.00	485.37	12,751.62	396.62

1/Based on last ten months of the year.

Source: U.S. Treasury until March 15, 1968 and Engelhard Industries, thereafter.

Table 63. Crude Oil Domestic First Purchase Price,[1] 1949-1988
(Dollars per Barrel)

Year	Alaska North Slope (current)	Other U.S. (current)	U.S. Average (current)	U.S. Average (constant)[2]
1949	—	2.54	2.54	10.81
1950	—	2.51	2.51	10.50
1951	—	2.53	2.53	10.08
1952	—	2.53	2.53	9.92
1953	—	2.68	2.68	10.35
1954	—	2.78	2.78	10.57
1955	—	2.77	2.77	10.18
1956	—	2.79	2.79	9.93
1957	—	3.09	3.09	10.62
1958	—	3.01	3.01	10.13
1959	—	2.90	2.90	9.54
1960	—	2.88	2.88	9.32
1961	—	2.89	2.89	9.26
1962	—	2.90	2.90	9.09
1963	—	2.89	2.89	8.92
1964	—	2.88	2.88	8.75
1965	—	2.86	2.86	8.46
1966	—	2.88	2.88	8.23
1967	—	2.92	2.92	8.13
1968	—	2.94	2.94	7.80
1969	—	3.09	3.09	7.76
1970	—	3.18	3.18	7.57
1971	—	3.39	3.39	7.64
1972	—	3.39	3.39	7.29
1973	—	3.89	3.89	7.86
1974	—	6.87	6.87	12.72
1975	—	7.67	7.67	12.93
1976	—	8.19	8.19	12.98
1977	[3] 6.32	[3] 8.63	8.57	12.73
1978	5.21	9.56	9.00	12.47
1979	10.57	13.01	12.64	16.08
1980	16.87	22.65	21.59	25.19
1981	23.23	33.71	31.77	33.80
1982	19.92	30.43	28.52	28.52
1983	17.69	28.00	26.19	25.21
1984	17.91	27.59	25.88	24.03
1985	16.98	25.74	24.09	21.72
1986	6.45	14.13	12.51	10.98
1987	10.83	16.83	15.40	13.08
1988[4]	8.43	13.97	12.57	10.33

[1] See Appendix E, Note 9.
[2] In 1982 dollars, calculated using implicit GNP price deflators. See Appendix C.
[3] Average for July through December only.
[4] Preliminary.
— = Not applicable.

Sources: •1949 through 1973—Bureau of Mines, *Minerals Yearbook*, "Crude Petroleum and Petroleum Products" chapter •1974 through January 1976—Federal Energy Administration, FEA Form 90, "Crude Petroleum Production Monthly Report." • February 1976 through September 1979—Federal Energy Administration, Form FEA P-124, "Domestic Crude Oil Purchaser's Monthly Report." • October 1979 through 1982 – Economic Regulatory Administration, Form IR2, "Domestic Crude Oil First Purchase Report." • 1983 and forward—Energy Information Administration, Form IR2, "Domestic Crude Oil First Purchase Report."

Annual Energy Review 1988

DEFINITIONS

Free on Board (FOB) Cost: The f.o.b. cost is the price actually charged at the producing country's port of loading. The reported price includes deductions for any rebates and discounts or additions of premiums where applicable and should be the actual price paid with no adjustment for credit terms.

Landed Cost: Landed cost represents the dollar per barrel price of crude oil at the port of discharge. Includes charges associated with the purchase, transporting and insuring of a cargo from the purchase point to the port of discharge. Does not include charges incurred at the discharge point (e.g., import tariffs or fees, wharfage charges, and demurrage).

Refiner Acquisition Cost: The average price paid by refiners for crude oil booked into their refineries in accordance with accounting procedures generally accepted and consistently and historically applied by the refiners concerned. Domestic crude oil is that oil produced in the United States or from the outer continental shelf as defined in 43 USC Section 1131. Imported crude oil is any crude oil which is not domestic oil. The composite is the weighted average price of domestic and imported crude oil. Prices do not include the price of crude oil for the Strategic Petroleum Reserve.

World Crude Oil Price: The estimated contract price of crude oil between two countries based on government-selling prices, netback values, or spot market quotations.

Specifications: Vibrocorer Model No.4507.

Coring Head.

Diameter	450	mm.
Height overall	500	mm.
Weight	100	kg.
Construction	Mild steel and aluminium	
Power requirement ·	415V 3 phase 6 amps.	

Core Barrels. (Expendible)

Aluminum Irrigation Pipe   7m x 76mm OD, 1.22mm wall thickness.

(Alcan catalogue No. EDP 3048L)

Bits.   Steel, with an appropriate core retainer.
                             Bits are plated to prevent corrosion.

Tower.

Aluminum construction.

Overall dimensions:

Diameter	1 m.	
Height	8 m.	
Weight	100 kg. (without lead)	
Lead ballast	90 kg.	

Vibrators.

Frequency	1440 per minute.
Centifugal Force	8500 newtons.
Power requirements	1.2KW 3 phase
Working Moment	75 Kg cm.

Alternator.

Powerlite 5 KVA Diesel Generator..

Shipping weight for the complete vibrocorer system approximately
300 Kg.   Core barrels are approx 6 Kg each for a 6 M length; the
maximum lenght for air freight.