The Leachox™ Refractory Gold Process – The Testing, Design, Installation and Commissioning of a Large Scale Plant at the VASGOLD Gold Mine, Kazakhstan

Stephen Flatman [1], Michael Battersby [1], Rainer Imhof [1], Richard Mark Battersby [1] and Sabit Ibrayev

1. Maelgwyn Mineral Services Ltd, Ty Maelgwyn House, 1A Gower Street, Cathays, Cardiff, CF24 4PA, UK. email: mms-contact@maelgwyn.com
2. JSC Vasilkovsky GOK, Gorky Street, 29b, Kokshetau, Severo-Kazakhstan Oblast, 47500, Kazakhstan

ABSTRACT

The Leachox refractory gold process has to date been applied to the selective treatment of several refractory gold ores but on a relatively small scale. In this paper the successful implementation of the process, from initial testing through to commissioning at one of the world's largest refractory gold mines is described. The Leachox plant has been designed to treat 50 tons per hour of a sulphide concentrate. This is over 50% higher than the largest bio-oxidation plant currently operating worldwide and represents a major breakthrough in the treatment of traditionally difficult to process refractory gold ores. The process offers a low capital and operating cost route, but with similar high gold recoveries to bio-oxidation.

INTRODUCTION

The Problem of Refractory Ores

Refractory gold ores are generally defined as those ores that do not give economical recoveries in conventional cyanide circuits where the ore has been ground to around 53 to 75 microns. There are several reasons as to why an ore may be refractory. The two most common are:-

1) The gold is locked in refractory sulphide minerals such as pyrite, arsenopyrite or pyrrhotite and occurs in both the chemically bonded state and as micro or nano-size grains of metallic gold

2) The gold is locked in silica

Other reasons for an ore not responding to conventional cyanidation include:-

1)"Preg robbing" due to the presence of carbonaceous material

2) Other minerals reacting with cyanide

3) The presence of lead, copper or antimony minerals

4) Interference and passivation from the decomposition products of pyrite and other minerals

Knowledge of the mineralogy of an ore body whilst always an essential starting point for drafting any process flowsheet, then becomes even more vital for refractory ores. Once the process mineralogy has been established there are some commercially successful processes developed over the years to treat such refractory gold ores. Processes that have been used include:-

Roasting

Pressure Oxidation

Bacterial Oxidation

Other ways to increase recoveries are:-

Ultra-fine grinding

Pre-aeration

Pre-oxidation

Chemical oxidation

Electrochemical Oxidation

High Pressure cyanidation

Processes such as roasting, pressure oxidation, and bacterial oxidation are all aimed at breaking down the sulphide matrix to liberate gold. Ultra-fine grinding performs the same function particularly where gold is locked in silicates or other minerals. Many of these processes are well developed and can yield very high gold recoveries but unfortunately all tend to have inherent issues. Roasting for instance is an environmentally unfriendly process and presents permitting issues in many countries; pressure oxidation requires a fairly high degree of operator skill and control not to mention often exotic materials of construction. Bacteria used in bacterial oxidation whether heap of tank leaching are susceptible to changes in environmental conditions and require careful control.

Whilst it is beyond the scope of this paper to go into the relative merits and demerits of each process the biggest issue that they all have in common to some degree is the very high associated capital and operating costs.

Given the combination of complex/difficult mineralogy process flowsheets combined with high capex and opex costs mining companies have historically where possible then steered well clear of refractory ore bodies . When these high capital and operating costs are input into financial models, the project financials i.e. NPV and IRR are often unattractive even before discounting risk. The project then remains as a paper exercise only awaiting either a sustained increase in the commodity price (without escalation of other costs) or a step improvement in technology. Accepting the above rather gloomy scenario and faced with fewer and fewer free milling, free leaching ore bodies available, mining companies are increasingly investigating the treatment of these refractory ore bodies buoyed on by the higher precious metals prices of recent years.

LEACHOX BACKGROUND

The mining company options have now expanded considerably with the introduction of the Leachox process developed by Maelgwyn Mineral Services (MMS) which is a fully fledged low cost refractory gold treatment process. Whilst possibly not giving quite as high recoveries as pressure oxidation this is more than compensated for by the significantly lower operating and capital cost. The process is currently being adopted by mining companies in order to turn their projects from "not economic" to economic with several major projects incorporating the process advancing and one at Vasilkovskoye Gold mine in Kazakhstan currently being commissioned. The test work leading up to the selection of Leachox for this gold mine is discussed later in the paper. Firstly however it is necessary to understand the Leachox process and the reasons for its selection.

The Leachox Process is essentially the integration of "in house knowledge" plus a range of established and new unit process steps , such as Imhoflot flotation, ultra-fine grinding of the flotation concentrate and then partial oxidation of sulphides followed by leaching in a leach

column to replace conventional open to atmosphere leach tanks. More recently a low cost cyanide destruction process has been added to the flowsheet. Central to all of these unit process steps is a high shear reactor the Aachen reactor.

Leachox has been used commercially in some form at the Agnes mine near Barberton and TGME's operation near Pilgrim's rest both in South Africa. A number of mining companies have recently committed to incorporating Leachox into their overall process flowsheets including RandGold Resources Tongon mine in Cote d'Ivoire, West Africa.

LEACHOX PROCESSES

Imhoflot Flotation

For refractory sulphide ore bodies Imhoflot pneumatic flotation would normally be the first step of the Leachox process to produce a high grade, low mass concentrate from a combination of V-cells and G-cells.

The term pneumatic flotation is generally associated with flotation where the aeration of the pulp takes place outside of the flotation cell. This is the main differentiating factor between pneumatic flotation and conventional tank flotation. The energy required by conventional cells to keep particles in suspension and generate bubbles is now focused solely on the production of very fine bubbles in the Imhoflot system, and the suspension of particles is catered for in the surplus energy of the system.

The external aeration is usually achieved either by utilising a simple venturi system in a pipe with down comers or by using specialised fine bubble generation technology. This fine bubble generation technology is a core feature of the Imhoflot system and contributes to a number of advantages over conventional tank cells including:-

- Enhanced Kinetics
- Significantly reduced residence time, leading to reduced capital and operating costs due to a much smaller footprint
- Lower power consumption
- No moving parts

Figure 1.0 Imhoflot G-cell –Cleaning a gold pyrite/arsenopyrite ore

Ultra Fine Grinding

Often when gold is locked in sulphides and silicates it is necessary to reduce the particle size to a size where gold is liberated or partially liberated. For refractory ores this generally translates into grinding below 10 microns and often to as low as 3-4 microns. Historically this was cost prohibitive as the only mills that were available to do this were tumbling mills which become highly inefficient at these low sizes particularly below 20 microns. The last 10-15 years however, have seen the development of a number of grinding mills specifically designed for ultra-fine grinding in the minerals industry. This has lead to a commensurate reduction in cost to grind fine and has been the catalyst for the development of many refractory ore treatment processes.

One of the drawbacks of grinding finer is that in addition to liberating the desired mineral it also increases the surface area of other host minerals .This can result in order of magnitude increases in reagent consumption particularly cyanide for the subsequent leaching process unless cognisance of this is taken in the final process design.

In addition to the commercialisation of ultra fine grinding mills similar advances have taken place in terms of developing specific ceramic grinding media for theses mills. This is beneficial due to less iron contamination which would otherwise further consume lixiviants.

Ultra fine grinding mills can be basically split into three main types:-

- Vertical stirred ball mill- (high speed and low speed)
- High intensity horizontal disc mill
- High intensity vertical mill

Aachen Reactors and MMS Leach Column

In theory grinding ores to gold particle liberation size should then render them amenable to cyanidation and thus no longer refractory. However for most ore bodies grinding to below 20 microns and particularly below 10 microns as previously mentioned grinding , whilst liberating more gold, also liberates many other mineral species which effect the oxidative and cyanide downstream process. This can dramatically increase reagent consumptions in open tank oxidative leach processes often by an order of magnitude and thus the costs effectively rendering lower grade ores uneconomic.

Fortunately this fatal flaw of many refractory processes is solved in the leachox process through the use of Aachen reactors. The Aachen reactors are basically designed to facilitate mass transfer by increasing the dispersion of oxygen gas into slurry. This is done by accelerating the slurry stream at the gas addition point by the use of a ceramic inserts which increase shear rates in the subsequent flow mixing zone. The resultant output is the generation of extremely fine gas bubbles which then provide for a high rate of oxidation of sulphides.
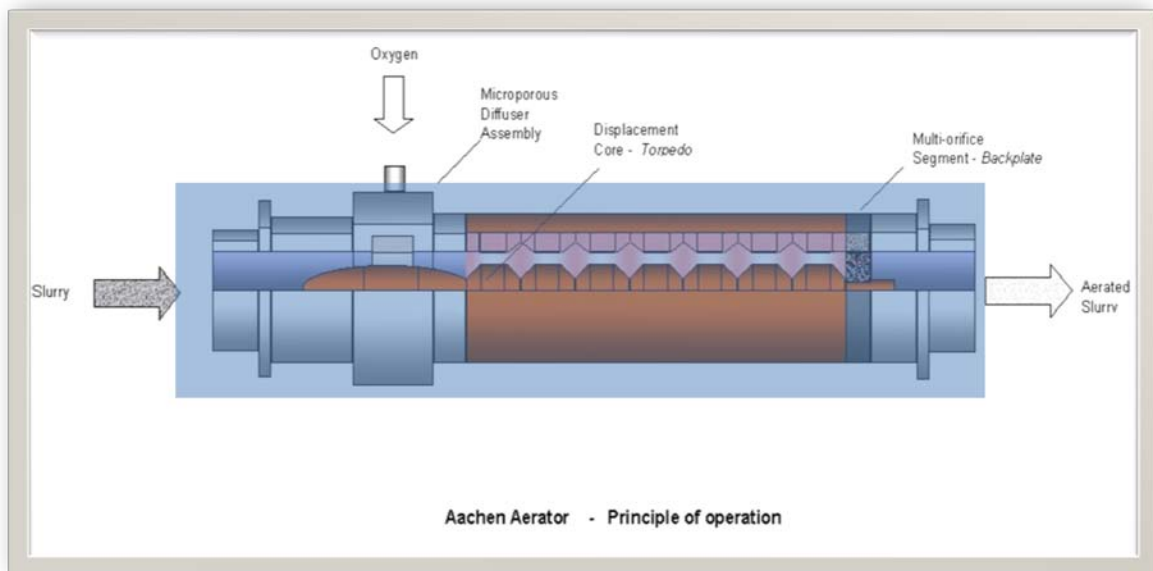

Figure 2.0 Aachen Reactor –Principle of Operation

Whilst the Aachen reactors are highly efficient mass transfer devices and can be used to assist open tank leach processes a natural progression has been to further accelerate leach kinetics through the use of MMS leach columns. A typical installation is depicted in Figure 3.0 below. The use of these leach columns can result in significant reductions in residence time on any ore compared to conventional open tank leaching but particularly for refractory gold ores which can be very slow leaching. This leads to a significant reduction in both capital and operating cost which then feeds back favourably into the overall project economics.
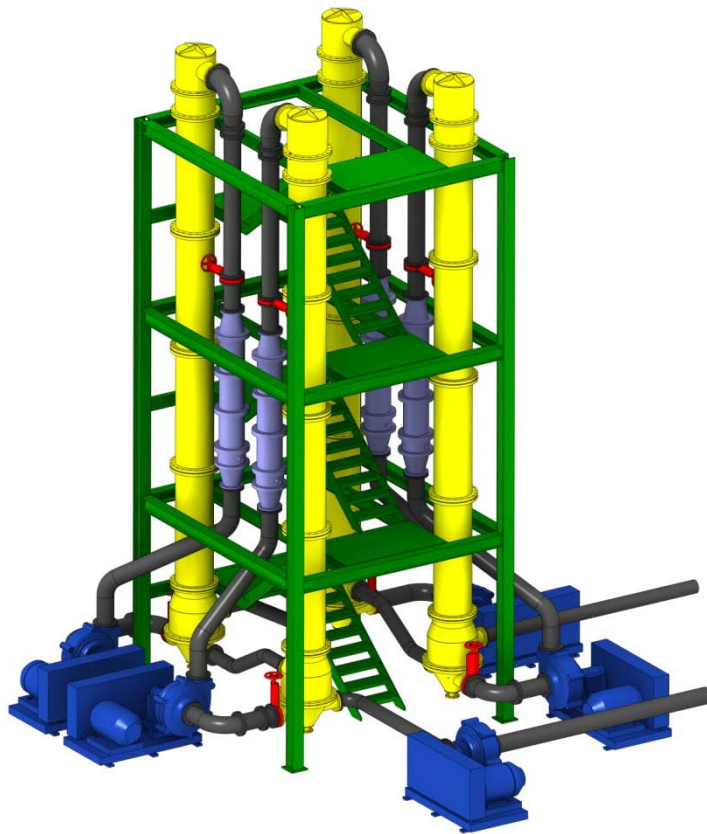
Figure 3.0 Leach columns and Aachen reactors

Cyanide Destruction CN-D Process

With the introduction of the International Cyanide Code in the mining industry mining companies are now forced to make provision for a cyanide destruction plant within their financial model. Whilst the cost of this is obviously a concern for even a free leaching relatively low cyanide consumption oxide ore it is of particular concern for many refractory processes due to the much higher cyanide additions required. Whilst careful control of cyanide addition linked to feed forward and feedback control systems can help to moderate residual cyanide tenors for many operations this is insufficient and a further reduction in tenor is required. There are several commercial processes available to do this and they can be broadly split into two different categories namely destruction or recovery based processes. It is beyond the scope of this paper to go into the relative merits and demerits of the individual processes but unfortunately they all have inherent disadvantages mainly associated with high reagent additions or high capex costs which can have a significantly adverse effect upon project financials.

In order to address these concerns MMS has recently developed a low cost cyanide destruction process, the CN-D Process. This process also incorporates the use of Aachen Reactors in another mass transfer application. It has the ability to destroy cyanide to well below the levels required

by the Code at a fraction of the cost of existing processes. In the CN-D Process free and WAD cyanide are oxidised by the Aachen Reactors using oxygen and activated carbon at specific Eh and pH values in the presence of ore specific catalysts. The optional use of a catalyst can further reduce SAD and total cyanide as well as cyanates if required. Apart from its inherent simplicity the process, with its utilisation of activated carbon, has the added advantage of acting as a "policeman" or safety net to further reduce any residual soluble loss of gold. MMS has calculated the operating cost for this cyanide destruction process to be very competitive at approximately US$ 0.20/ton treated based on a 450tph operation. This cost includes the license cost and equipment supply cost but excludes the revenue from any incremental gold recovered which, depending upon the upfront adsorption efficiency, has the potential to not only reduce this cost further to zero but to provide a credit. A typical layout of the process is illustrated below in Figure 4.0 below:-
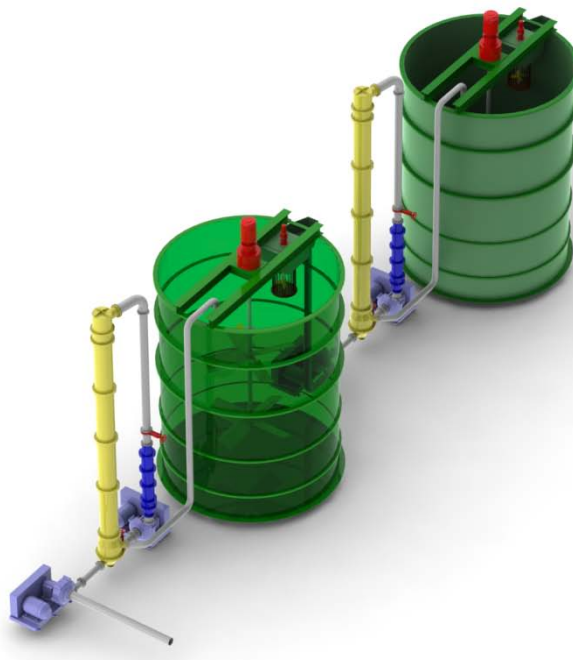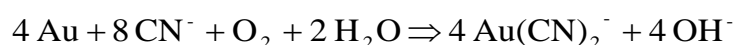
Figure 4.0 Graphic Representation of CN-D Process

Chemistry of Cyanidation

Whilst cyanide is one of the major drivers if not the major driver of gold dissolution it cannot be viewed in isolation particularly in respect of the relationship between cyanide and oxygen derived from the well known Elsner's equation.

$$4\,\mathrm{Au} + 8\,\mathrm{CN}^- + \mathrm{O}_2 + 2\,\mathrm{H}_2\mathrm{O} \Rightarrow 4\,\mathrm{Au(CN)}_2^- + 4\,\mathrm{OH}^-$$

From the above it is apparent that both cyanide and oxygen are required in an aqueous solution in order to leach gold. For the leaching of pure gold, the stoichiometry of Elsner's reaction gives the required molar ratio of free cyanide to oxygen as 8: 1 (or 6½: 1 when expressed as a mass

ratio). As cyanide is the more expensive reagent, it is desirable from a financial point of view to ensure that cyanide is the rate-limiting reagent. It is important to note that the rate of dissolution of gold in alkaline cyanide solutions is controlled by the rate of dissolution of oxygen from the bulk solution to the metal surface. This explains why the Aachen reactors are so successful in boosting leach kinetics and reducing reagent consumption as the reactors are specifically designed to maximise the phase interface surface area.

Pilot Plant Test work

MMS has a number of pilot plant facilities based in the United Kingdom, South Africa, Russia and Australia for pilot scale testing of refractory flotation concentrates. These facilities include a selection of both horizontal and vertical high intensity mills.

The United Kingdom installation is based at Cardiff University in Wales and consists of a 10 litre vertical ultra fine grinding mill to grind samples from a feed size of approximately 100 microns down to as low as 2 microns. The exact grind size will obviously be dictated by the mineralogy.

Figure 5.0 Ultra Fine Grinding Milling Pilot Plant

Figure 6.0 Leachox Pilot Plant

Milled feed is introduced into a 65 litre capacity storage tank from where it is pumped into a down flow leach column in tandem with a laboratory Aachen shear reactor. Cyanide and caustic addition were originally designed to be added directly to the column but practical experience has indicated that addition to the feed tank was easier.

A variable speed recirculation pump recycles the column through the Aachen reactor where oxygen is injected at approximately 8 bar pressure. The column pressure is approximately 3 bar. The leach column is fully automated with on line measurement of pressure, temperature, pH, redox potential, dissolved oxygen level, and pulp flowrates linked to a PC. A flow sheet of the pilot plant is given below in Figure7.0
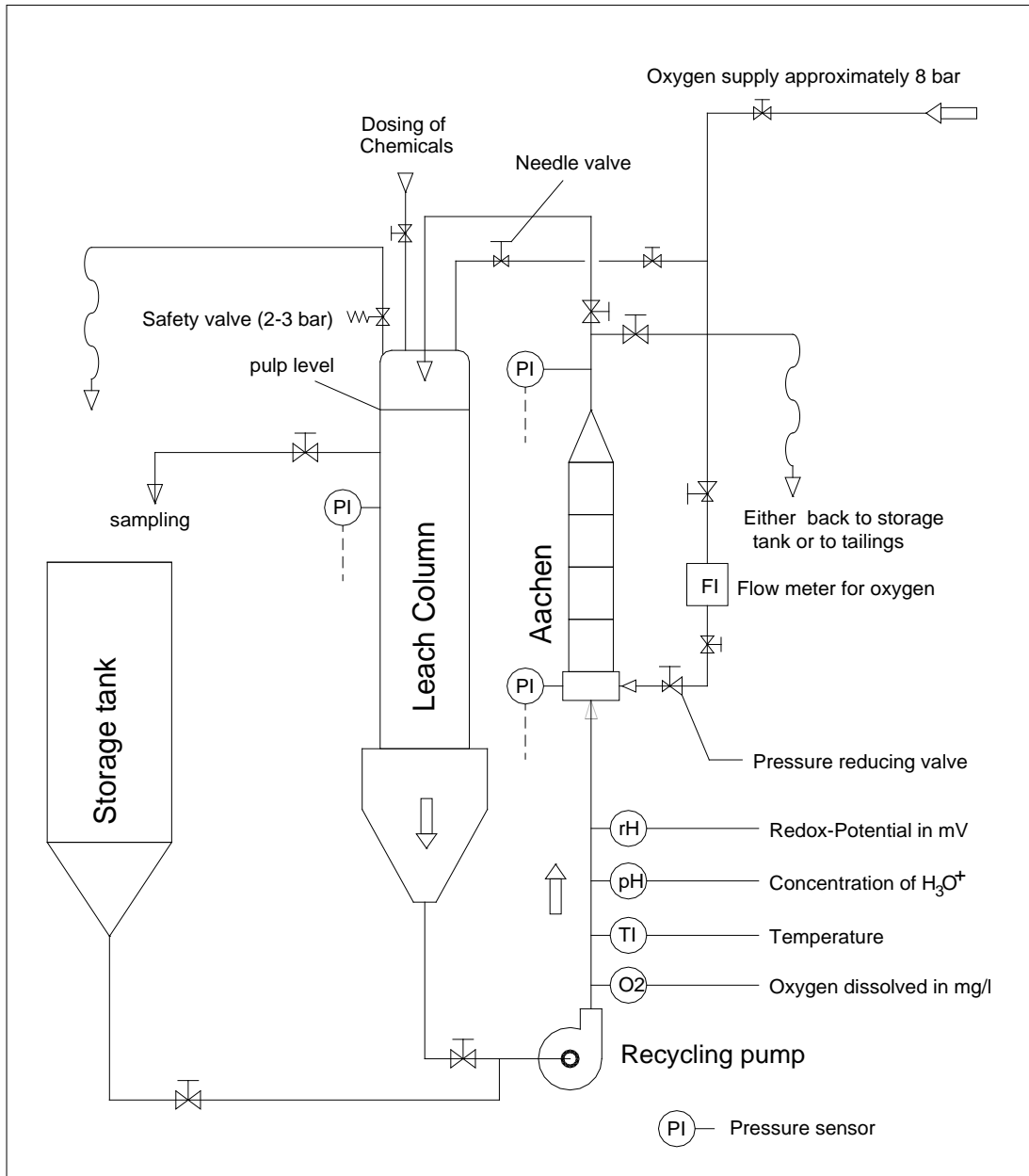
Figure 7.0 Flowsheet of leach column Pilot Plant

Vasilkovskoye Gold Mine Leachox

Leachox has recently been integrated into the flow sheet of Kazakhstan based Vasilkovskoye gold mine with the plant currently being commissioned. The laboratory and pilot plant test work leading up to this installation are now described in detail including the initial commissioning progress.

Testwork

Laboratory and pilot plant scale test work was performed by MMS on a concentrate sample supplied by Vasilkovskoye to model the Leachox process.
The head grade of the bulk sample assayed 48.4g/t Au and 15.5% S.

In order to gain an initial appreciation for the amenability to leaching diagnostic leaching testwork was carried out using sequentially more aggressive conditions and indicated the following gold/mineral associations:-

Table 1.0 Diagnostic Leach Results

Gold association	Gold	
	g/t	%
Available to direct cyanidation	41.8	81.76
Preg-robbed-CIL	1.84	3.60
Preg –robbed-CIL with finer milling	4.57	8.93
HCl Digestible Minerals	1.02	2.00
HNO_3 digestible minerals	1.50	2.93
Carbonaceous matter	0.24	0.47
Quartz (balance)	0.15	0.31
Total	51.12	100.00

The sample gold head grade assayed 51.12g/t. Of the total contained gold, 81.76% was amenable to direct cyanidation indicating the sample supplied to be moderately refractory. Under direct cyanidation conditions, 3.60% of the gold was preg-robbed by components occurring within the ore. Approximately 85.36% of the contained gold should therefore be recoverable via carbon-in-leach processing in the "as received" sample. A further 8.93% (4.57g/t Au) gold can be recovered if the sample is milled finer before the carbon-in-leach. Of the remaining gold, some 0.31% (0.15g/t Au) was found to be occluded in gangue components. Some 0.47% (0.24g/t Au) of the gold was associated with carbonaceous constituents. The HCl digestion revealed that 2.00% (1.02g/t Au) of the contained gold was associated with HCl digestible minerals. The HNO_3 digestion phase showed that 2.93% (1.50g/t Au) of gold was associated with HNO_3 digestible minerals.

X-ray diffraction analysis was also done on the sample indicating that the primary minerals were arsenopyrite and quartz with minor amounts of pyrite, biotite chlorite, K-feldspar, albite dolomite and scorodite.

QEMSCAN was also applied to the sample and Trace Mineral Analysis (TMA). The estimated mineralogical composition of the sample is summarized in Table 2.0 below:-

Table 2.0 Estimated Mineral Composition of Sample

Mineral	Approx. Formula	Mass%
Arsenopyrite	FeAsS	68.88
Pyrite/pyrrhotite	FeS_2	3.75
Galena	PbS	0.34
Sphalerite	ZnS	0.15
Chalcopyrite	CuFeS	0.17
Other Sulphides		0.03
Total Sulphides		72.33
Quartz	SiO_2	9.27
Plagioclase		3.28
K-Feldspar		6.08
Biotite		2.76
Chlorite		2.26
Amphibole		0.06
Total Silicates		23.72
Fe-oxide/hydroxide	FeO(OH)	2.11
Scorodite		0.02
Carbonate		0.31
Ilmenite		0.18
Apatite		0.18
Zircon		0.06
All Others		0.10
Total		100.00

The size distribution of the head sample is presented in Table 3.0 below.

Table 3.0: Head Sample feed Size Distribution

Screen size microns	Mass g	% Retained	% Cumulative Retained	% Passing	% Cumulative Passing
212	8.5	1.7	1.7	1.0	98.3
150	4.9	1	2.7	1.9	97.3
106	9.3	1.9	4.5	4.9	95.5
75	24.4	4.9	9.4	26.6	90.6
53	133.1	26.6	36.0	8.2	64.0
45	41.1	8.2	44.2	14.1	55.8
25	70.6	14.1	58.4	41.6	41.6
Total on screens	291.8				
Total initial mass	500.0				

The screen analysis indicated that the "as received" sample had a grind of 90.6% passing 75 microns. 55.8% of the material passed through 45 microns whilst 41.6% passed through 25 microns.

Kinetic Scouting Tests

Three CIL bottle roll leaches were performed on the as received sample for 8, 24 and 48 hour leach times to provide an indication of leach kinetics. The roll bottle leach conditions were:-

- 10kg/t cyanide addition (as100% NaCN)
- pH 10.5
- 10g/l carbon addition
- 8, 24, and 48hr leach times.

Table 4.0 presents the results for the base case bottle rolls:-

Table 4.0: Base Case Kinetic CIL Bottle Rolls

	Au g/t	Au Rec %	%S	Lime kg/t
Head	48.4		15.5	
CIL Leach Residue 8hrs	8.65	82.13	14.7	12
CIL Leach Residue 24hrs	8.37	82.71	15.1	12
CIL Leach Residue 48hrs	7.62	84.26	15.1	12

The base case kinetic CIL bottle rolls indicated a fast initial leach (82.13% gold recovery in 8 hours) with a subsequent very slow leach up to 48 hours to yield a final gold recovery of 84.26%. Lime consumption was 12kg/t, which is typical for flotation concentrate material.

In order to expedite subsequent leach tests, a 24hour leach time was used for the rest of the testwork programme due to the large number of tests involved.

Graphite Flotation Work

A sub-sample of the "as received" material was also subjected to a standard laboratory bench float test with diesel to float any carbon that may be present in the sample. The following reagent suite was used:

- 1kg/t diesel
- 100g/t Sasfroth

No mass was pulled off in the graphite float test indicating that if there was any graphite present it occurred in a very low concentration.

Leachox Modelling Work

In order to assess the applicability of the Leachox process more detailed simulation work was conducted which involved the following:-

Separate aliquots of sample were milled to target d_{90}'s of 45, 25, 15, 10 and 5 microns respectively.

- At each grind, samples were split into two representative portions and subjected to:
    - 24 hour bottle roll leaches with and without carbon
    - Aachen simulation followed by 24hour bottle roll leaches with and without carbon
- In addition, the sample at a target grind of 5microns was further subjected to:
    - A 48 hour leach where carbon was added after 24hours to assess whether any possible preg-robbed gold could be retrieved by contacting with carbon.
    - A 24hour conditioning with diesel followed by a CIL bottle roll leach.

The conditions for the bottle roll leach tests were:-

- 20kg/t cyanide addition
- pH 10.5
- 10g/l carbon addition on bottle rolls with carbon
- 10kg/t diesel on bottle rolls with diesel
- Leach times as per above.

Testwork results are summarised in Table 5.0 below with detailed results provided in Appendix 1.0.

Table 5.0: Leachox Modelling Testwork Results Summary

	Grind d$_{90}$ microns	Leach Time hrs	Au g/t	Au Rec %	%S	Oxygen kg/t	Lime kg/t	Temperature °C during Aachen Simulation
Head			48.40		15.5			
Leach Residue, Aachen simulation, with carbon	75	24	7.70	84.09	14.3	20	12	36.8
Leach Residue with carbon	53	24	7.41	84.69	13	20	12	
Leach Residue, Aachen simulation, with carbon	53	24	5.52	88.60	15.5	20	12	36.8
Leach Residue, Aachen simulation, with carbon	43	24	6.03	87.54	14.4	20	15	38.1
Leach Residue, Aachen simulation, with carbon	9	24	2.02	95.83	14	20	18	38.3
Leach Residue, Aachen simulation, with carbon	5	24	1.84	96.20	13.3	20	22	40.2
Leach Residue, Aachen simulation, with carbon	4	24	1.86	96.16	12.4	20	33	42.7
Leach Residue, Aachen simulation, carbon after 24hrs	4	48	1.88	96.12	13	20	33	42.7
Leach Residue, Aachen simulation, diesel conditioning after 24hrs, carbon after 24hrs	4	24	1.56	96.78	13.1	20	33	42.7

A steady increase in Au recovery was observed as the sample was ground from a d$_{90}$ of 75 microns to a d$_{90}$ of 4 microns with the corresponding gold recovery increasing from 84% to 97%. At an anticipated concentrate treatment rate of 50tpd and 90% availability this is equivalent to an additional 2531kg of gold per annum. Assuming a gold price of US$1200/oz this equates to additional US$97million revenue per annum.

The samples showed mildly preg-robbing characteristics which as expected was exacerbated with grinding. However, with carbon addition, most of the preg-robbed gold was preferentially adsorbed to the more active carbon. Diesel conditioning to blind the preg-robber proved to be beneficial. The Aachen simulation work also appeared to reduce the severity of the preg-robbing characteristic.

The percentage sulphide sulphur values steadily dropped from 15.5% to around 13% with increasing grind and Aachen simulation. This corresponded with increasing temperatures during the Aachen simulation – from 36^{0}C to 42.7^{0}C. Lime consumption also increased with increasing grind – from12kg/t to 33kg/t. Oxygen consumption was calculated at 20kg/t.

Process description

The Vasilkovskoye gold mine circuit is highly innovative processing a total of 5mtpa of Run of Mine (ROM) ore to produce an arsenopyrite flotation concentrate which is treated at 50tph through a Leachox process consisting of ultra-fine grinding followed by pre oxygenation leach tanks in tandem with Aachen reactors. Leaching of the gold then takes place in a leach column in tandem with a single Aachen reactor followed by recovery of the dissolved gold through a resin in pulp circuit. A simplified block flowsheet is given in Figure 8.0 below:-
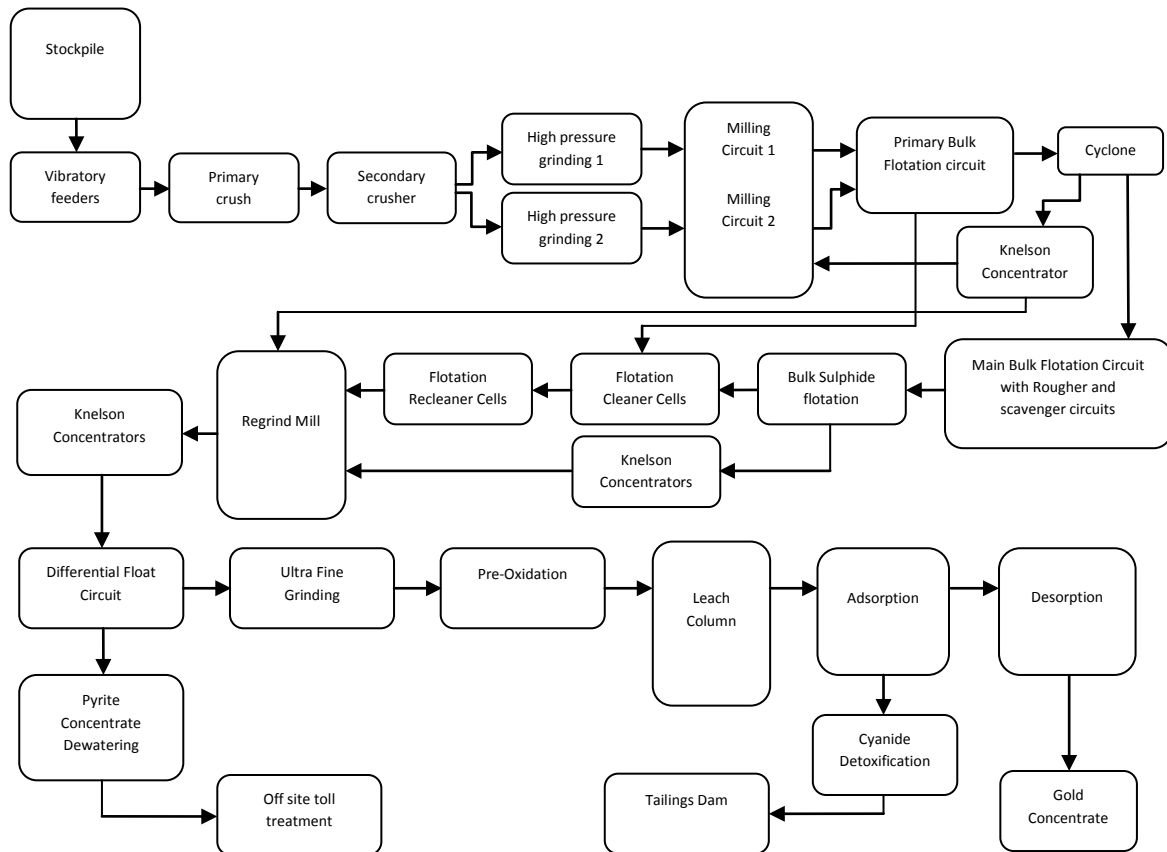

Figure 8.0 Simplified Block Flowsheet of Vasilkovskoye Gold Mine Concentrate Circuit

Reviewing the circuit in more detail ROM ore assaying approximately 6g/t Au is trucked to a ROM stockpile where -350mm material is fed via vibratory feeders and conveyed to primary and secondary crushing reducing the feed to 95% -35mm. Tertiary crushing consists of two KHD Humbolt Wedag high pressure grinding rolls (HPGR's) RP16 – 170/180 operating in parallel which crush the ore to 80% minus 5mm. Each HPGR has a feed range of 1000-1400 tph.

The primary milling circuit consists of two 6.7m x 11.3m EGL 4900KW Outotec ball mills operating in parallel each being fed from the corresponding roller press. Feed grading 80% minus 5mm enters the mills with the mill discharge being pumped to the primary cyclone bank

(6 cyclones). The 60% minus 74 micron cyclone product is then pumped to the primary bulk flotation cells, with the underflow being returned back to the mills. Primary bulk flotation tails are again cycloned (16 cyclones per unit) with the corresponding overflow of 90% minus 74 microns pumped to the main bulk flotation cells .The underflow is first screened at 1.5 mm (oversize returned to mill) then treated in a Knelson KCXD 70 separator, the concentrate being pumped to the regrind mill, with the tailings returned to the mill.

Following the primary float a bulk sulphide flotation circuit is then installed. This consists of 2 banks of Outotec 130m^3 tank cells, the first three cells of each bank are fed with d60 minus 74 micron primary cyclone (6 cyclones per side) overflow. The concentrate is sent for cleaning, whilst the tails are fed to the secondary cyclones (16 cyclones per side) set at a d90 of 74 microns. The secondary cyclone underflow is screened at 1.5mm and fed to a Knelson Kcxd 70 separator, the tailings of which report back to the ball mills ,the concentrate being returned for regrind. The remainder of the bulk circuit is split into secondary roughers and scavengers. The scavenger concentrate is recycled to secondary rougher feed, the secondary cons reporting for cleaning and recleaning. The cleaner circuit is 2 parallel banks of 4 x 10m3 Outotec Tank cells. The recleaner circuit consists of 2 parallel banks of 3 x 5m3 Outotec Tank cells. Cleaner tails report to primary roughing whilst recleaner tails are recycled to cleaner feed with recleaner concentrate being sent for regrind.

 Flotation tails are first screened at 1.5 mm then fed to primary (Kcvd 64) then cleaner (Kcvd 32) Knelson separators, both tails reporting to final tails the concentrate being sent for regrind. The bulk sulphide concentrate is fed to a 7mx4m regrind mill with a 25mm ball charge .The mill product is then subjected to primary then secondary cyclone banks the latter resulting in a product of 95% minus 45 microns, the underflows of both banks being returned to the regrind mill. The secondary cyclone bank product is then screened at 1.0mm before being sent to primary then cleaner knelson separators, the cleaner concentrate joining the final pyrite concentrate ,whilst both tailings combining together to form the pyrite/arsenopyrite differential flotation plant feed.

The differential arsenopyrite/pyrite float circuit consists of 3 stage conditioning followed by 9 x 5m3 Outotec tank Cells as the primary float cells, followed by a further 3 x 5m3 Outotec Tank Cells as cleaners and 2x 2 Outotec TC 1.5 – 2.0 standard cells as recleaners. The recleaned pyrite concentrate is then thickened in an Outotec 4m thickener, filtered in a Metso pressure filter and then despatched for roasting and subsequent final gold recovery at an external smelter. The cleaner and recleaner tailings rejoin the primary segregation tailings as the arsenopyrite concentrate.

The arsenopyrite concentrate grading approximately 50g/t Au and 16% sulphur is then thickened to approximately 55% solids by weight and fed to the first stage of the Leachox process consisting of three 2000 litre vertical stirred fine grinding mills to produce a product at 90% minus 4 microns from the 90% -45 microns feed. Pre-oxygentation then takes place in 4x $650m^3$ pre-oxygenation tanks .Each tank is run in tandem with two REA-Au 400 Aachen reactors). The slurry is then leached in a leach column (again in circulation with one Aachen reactor).Lime and cyanide are added to the leach feed at additions of 30kg/t and 1kg/t respectively. Oxygen is added to the Aachen reactors with an overall consumption of approximately 20kg/t .Diesel is also added to the leach feed to combat preg robbing .The subsequent downstream circuit for recovering the dissolved gold consists of a carbon adsorption and associated elution circuit. Final tails are subjected to a cyanide detoxification stage before pumping to the tailings facility. The Leachox section of the overall flowsheet is given in Figure 9.0 below and a detailed process flowsheet is given in Appendix 2.0.
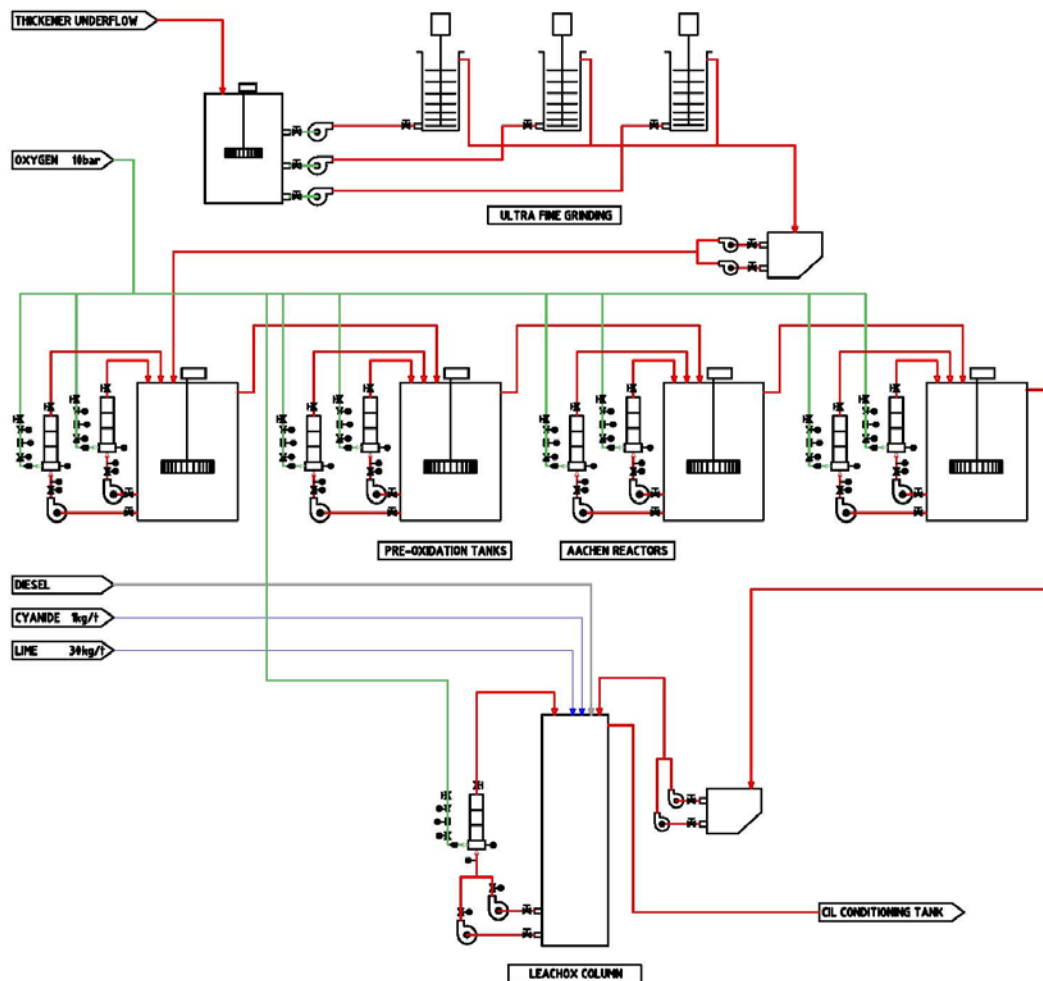

Figure 9.0 MMS Leachox Circuit at Vasilkovskoye

COMMISSIONING EXPERIENCE

The MMS Leachox plant was installed in August 2009. Whilst the plant is in partial operation commissioning has not been signed off at present due to:-

- Failure of and delays in commissioning upstream processes (in pit conveyors, ball mills, flotation circuit)

- Delay in the supply of essential control equipment outside of the Leachox contract scope i.e. density and flow control for UFG feed, non return valves, pressure gauges, flowmeters and regulators for the Aachen aerators

- Adverse weather conditions (coldest winter for several decades)

- Labour being drawn from the local community unfamiliar with operating process plants resulting in a long learning curve.

Initially the commissioning centred on preparing the hydraulic motors, power packs, and electronic control systems for the ultra fine grinding mills, the Aachen reactors having already been installed by the main engineering contractor TOMS. This progressed smoothly with the net result that by October 2009 all 3 mills had been water commissioned with a nominal bead load. Slurry feed commissioning commenced in November 2009 with a modified upstream circuit limited to only the primary roller press, ball mill combination running. This only provided sufficient feed to run one of the two banks of bulk sulphide rougher flotation cells, together with it's corresponding cleaner, and recleaner banks. The differential flotation circuit was not yet operational at this stage so the bulk sulphide concentrate was reground to a d95 of 45 microns, thickened then pumped directly to the fine grinding mills.

Initially the fine grinding mills were operated at approximately 110 rpm or 50% of their rated maximum together with a reduced bead load as modifications involving changing and reconfiguring the bearings in the bearing housings were deemed necessary by the supplier, with this work finally completed in December 2009. During March 2010 the fine grinding mill PLC's were connected up to the mine's Profibus system allowing full SCADA control of the mill.

The upstream secondary milling circuit only became operational during March 2010 as did the differential flotation circuit; however ongoing problems with the upstream ball mill gearboxes meant that the plants rated throughput of 1050 tons per hour was only recently achieved. These upstream problems have often resulted in a non steady state feed conditions to the Leachox circuit with large fluctuations being seen in both tonnage and density.

This situation has been further exacerbated by delays in essential control equipment such as flowmeters and density meters. Consequently to date the mills have not been run at their combined rated tonnage of approximately 50tph and therefore not yet fully tested however accepting this the designated grind of 90% minus 4 microns has easily been achieved even at these fluctuating tonnages.

In terms of the Aachen reactors commissioning the same problems were experienced with the non steady state feed conditions. Choking of the Aachens has occurred as expected as they were run without essential control equipment such as slurry non return valves, pressure regulators, flowmeters etc. The delivery of this equipment was outside of MMS's scope of supply and is currently still being installed with experience from other installations indicating that once installed as per specification this will resolve the problem.

As alluded to earlier there has also been a definite and steep learning curve due to the inexperienced plant personnel with the commissioning focus understandably being on upstream equipment and only more recently moving downstream to the Leachox process. Metallurgical leach test results are still awaited from Vasilkovskoye to provide base line recovery data and indications of improving leach efficiency as greater availability and increased steady state upstream operations are achieved. At present then it is premature to comment on the recovery increase directly attributable to Leachox although experience to date indicates that design recoveries should comfortably be achieved when the Leachox circuit is operated correctly.

SUMMARY

Historically mining companies have steered clear of refractory gold projects where possible due to a combination of perceived processing difficulties and poor economics associated with low recoveries. The paper has however illustrated that recent advances in technology integrated into the Leachox process have now eliminated these concerns with many companies now actively investigating new refractory projects and re-assessing old projects which were previously uneconomic.

The background to the Leachox process for treatment of refractory gold ores is detailed from initial process selection, through equipment selection and testwork to a commercial application resulting in a refractory gold concentrate treatment plant 50% larger than the current largest bi-oxidation plant in the world. In summary then Leachox provides a low operating and capital cost compared to other processes albeit at a marginally lower recovery resulting in improved overall economics.

REFERENCES

Botz, M.M., "Overview of Cyanide Treatment Methods", Mining Environmental Management, Mining Journal Ltd., London, UK, pp 28-30, 2001

"Leachox Modelling Testwork on Kazzinc Vasilkovskoye sample" internal communication, Maelgwyn Minerals Services Africa, South Africa 17 September 2007

Coetzee, L .L, Martin, G.J,"SGS Mineralogical Report no Min 0707/127"South Africa 26 September 2007

O'Connel , S.P,"SGS Report no Met 07/281-Quantification Diagnostic Leach Appraisal on a Maelgwyn Minerals Sample, 26th September 2007

Imhof, R.M., Battersby, M.J.G., Parra, F., Sanchez-Pino, S., (2005). The Successful Application of Pneumatic Flotation Technology for the Removal of Silica by Reverse Flotation at the Iron Ore Pellet Plant of Compania Minera Huasco, Chile. Centenary of Flotation Symposium, 5 – 9 June 2005, Brisbane, Australia.

Battersby, M.J.G., Fletcher, M.G., Imhof, R.M., Singh, A.A., Puder, F., (2005). The Advantages of the Imhoflot G-Cell Pneumatic Flotation Process with Centrifugal Froth Removal – Two Case Studies. Randol Innovative Metallurgy Forum, 21 – 24 August 2005, Perth, Australia.

Sanchez-Pino, S., Sanchez-Baquedano, A., Imhof, R.M., Battersby, M. J. G., (2006). Is Bigger Always Better in Flotation? The Imhoflot G-Cell. PROCMIN 2006 IV International Mineral Processing Seminar, 22-24 November 2006, Santiago, Chile.

Battersby, M.J.G., Imhof, R.M., Fletcher, M.G., Puder, F., (2007). The Efficient Recovery of Kaolin from a Hydrocyclone Plant Middlings Steam Utilising Imhoflot G-Cell Pneumatic Flotation. SME 2007 Annual Meeting 26-28 February 2007, St Louis, USA

Imhof, R.M., Fletcher, M.G., Singh, A.A., Vathavooran, A., (2007). Application of Imhoflot G-Cell Centrifugal Flotation Technology. SAIMM Conference, Flotation Cell Technology in the 21st Century, 20 June 2007, Johannesburg, South Africa.
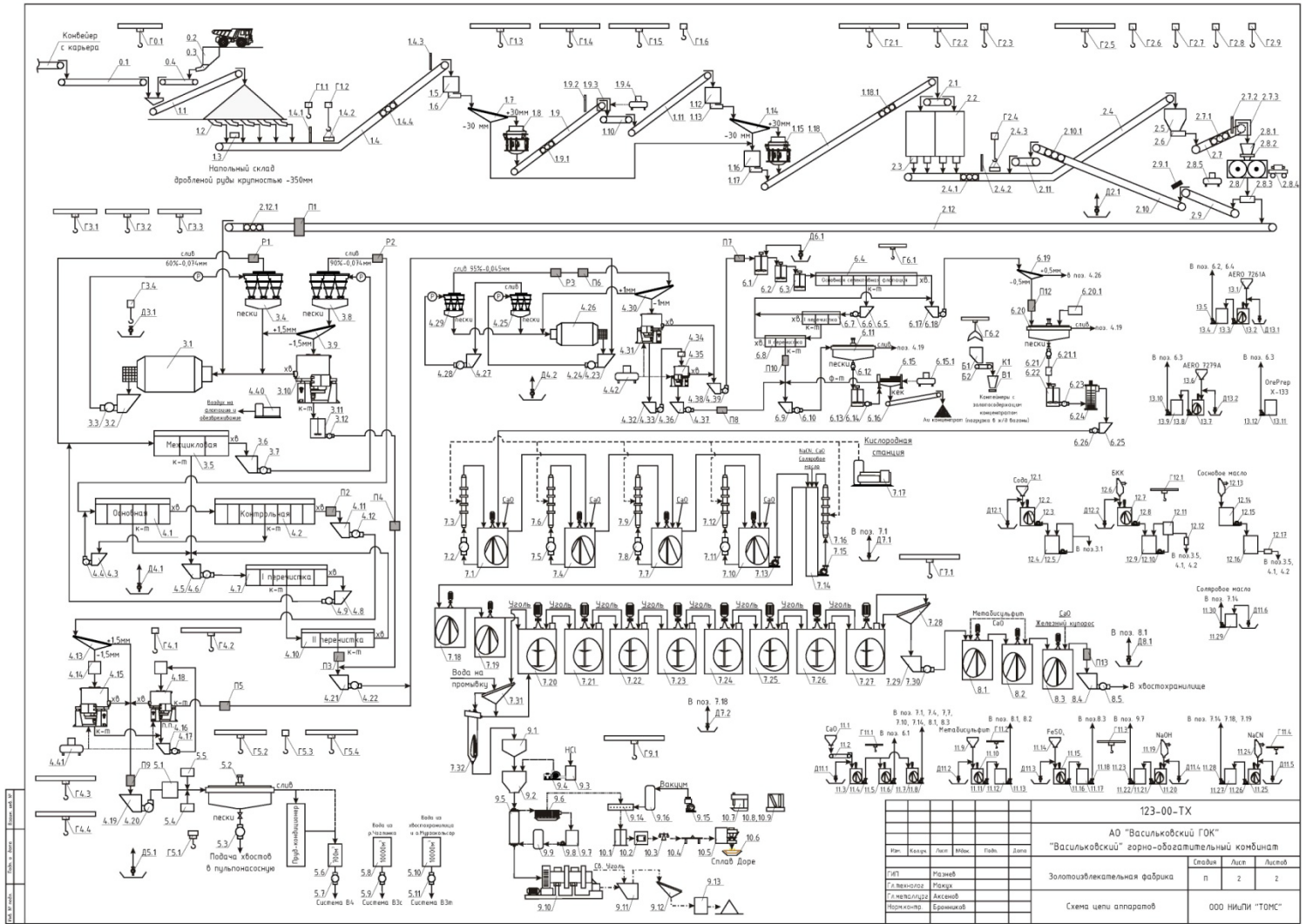
Sanchez-Pino, S., Sanchez-Baquedano, A., Imhof, R.M., (2008). Imhoflot G-Cell Pneumatic Flotation Technology for Fine Particle Applications. PROCMIN 2008 V International Mineral Processing Seminar, 22-24 October 2008, Santiago, Chile.

APPENDIX 1.0

Table 6.0 detailed leach results

	Grind d90 microns	Leach Time hrs	Au g/t	Au Rec %	%S	Oxygen kg/t	Lime kg/t	Temperature °C during Aachen Simulation
Head			48.40		15.5			
Leach Residue, with carbon	75	24	7.90	83.68	14.5	20	12	
Leach Residue, no carbon	75	24	9.50	80.37	14.7	20	12	
Leach Residue, Aachen simulation, with carbon	75	24	7.70	84.09	14.3	20	12	36.8
Leach Residue, Aachen simulation, no carbon	75	24	7.40	84.71	13.8	20	12	36.8
Leach Residue, with carbon	53	24	5.52	88.60	13	20	12	
Leach Residue, no carbon	53	24	9.66	80.04	15	20	12	
Leach Residue, Aachen simulation, with carbon	53	24	7.41	84.69	15.5	20	12	36.8
Leach Residue, Aachen simulation, no carbon	53	24	8.36	82.73	14.9	20	12	36.8
Leach Residue, with carbon	43	24	9.07	81.26	14.1	20	12	
Leach Residue, no carbon	43	24	15.90	67.15	14.4	20	12	
Leach Residue, Aachen simulation, with carbon	43	24	6.03	87.54	14.4	20	15	38.1
Leach Residue, Aachen simulation, no carbon	43	24	7.83	83.82	14.2	20	15	38.1
Leach Residue, with carbon	9	24	2.68	94.46	14.8	20	12	
Leach Residue, no carbon	9	24	5.00	89.67	14.8	20	12	
Leach Residue, Aachen simulation, with carbon	9	24	2.02	95.83	14	20	18	38.3
Leach Residue, Aachen simulation, no carbon	9	24	4.32	91.07	14.1	20	18	38.3
Leach Residue, with carbon	5	24	2.42	95.00	14.4	20	12	
Leach Residue, no carbon	5	24	4.22	91.28	14.5	20	12	
Leach Residue, Aachen simulation, with carbon	5	24	1.84	96.20	13.3	20	22	40.2
Leach Residue, Aachen simulation, no carbon	5	24	4.04	91.65	13.5	20	22	40.2
Leach Residue, with carbon	4	24	2.18	95.50	14.3	20	12	
Leach Residue, no carbon	4	24	4.11	91.51	14	20	12	
Leach Residue, Aachen simulation, with carbon	4	24	1.86	96.16	12.4	20	33	42.7
Leach Residue, Aachen simulation, no carbon	4	24	5.18	89.30	13.1	20	33	42.7
Leach Residue, Aachen simulation, carbon after 24hrs	4	48	1.88	96.12	13	20	33	42.7
Leach Residue, Aachen simulation, diesel conditioning after 24hrs, carbon after 24hrs	4	24	1.56	96.78	13.1	20	33	42.7