Recovery of Ultra Fines Using Imhoflot Pneumatic Flotation – Two Pilot Plant Case Studies Recovering Nickel and Zinc from Tailings Streams

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ABSTRACT

It has been well documented that conventional tank flotation is inefficient in the recovery of metallic fines and very inefficient in the recovery of ultra fines. In contrast Imhoflot pneumatic flotation, with the combination of pico bubble generation and order of magnitude increase in energy utilization in the collection zone over tank flotation has the ability to recover such ultra fines.

This paper details two separate site based pilot plant investigations into the ultra fine losses occurring into tailings by tank flotation and the ability of Imhoflot to recover these ultra fines. The first case study looks at a nickel operation in Europe where pilot plant work using a three stage Imhoflot G-Cell plant demonstrated its ability to recover approximately 30% nickel from the plant final tails, predominantly in the minus eleven micron fraction size range. The second case study investigated losses at a zinc operation where the liberation size of the zinc mineralisation was deemed to be around seven microns and this was the target grind size of the operation. A two stage Imhoflot G-Cell pilot plant achieved a 20% zinc recovery from the final cleaner tailings stream.

KEYWORDS

Froth flotation, pneumatic flotation, ultra-fines beneficiation, nickel sulphide, zinc suphide

INTRODUCTION

Froth flotation became the dominant method for the recovery of metallic minerals from ores shortly after its initial development now well over a hundred years ago (Lynch 2010). Since its early development the use of agitated tanks and compressed air became the standard method of effecting the collection of the metallic minerals. Such “tank” flotation still dominates the market in new beneficiation plants with estimates in excess of 95% market share; hence such tank flotation is also referred to as “conventional” flotation.

However over those 100 years numerous technical papers have identified conventional flotation as been inefficient at both extremities of the particle size range. Figure 1.0 indicates the standard view of almost all tank flotation plants (Pease 2004). In the coarse sizes (say +150 microns) this may be attributable to liberation issues but also the inabilities of the froth to carry liberated particles to a concentrate due to the competing tasks of originally also trying to maintain such particles in suspension and having them contact with bubbles. In the fine size range (say -30 microns) conventional flotation can be said to be inefficient and in the ultra fine size range (say -15 microns) extremely inefficient. Since the early days very few developments have been made to address these issues. The tanks originally being square are now generally round and, in an attempt to increase bubble – particle contact for the ultra fines, in some operations much more energy is inputted into the system by the installation of larger motors. However putting more energy into the tank system also compromises coarse particle recovery as mentioned above.

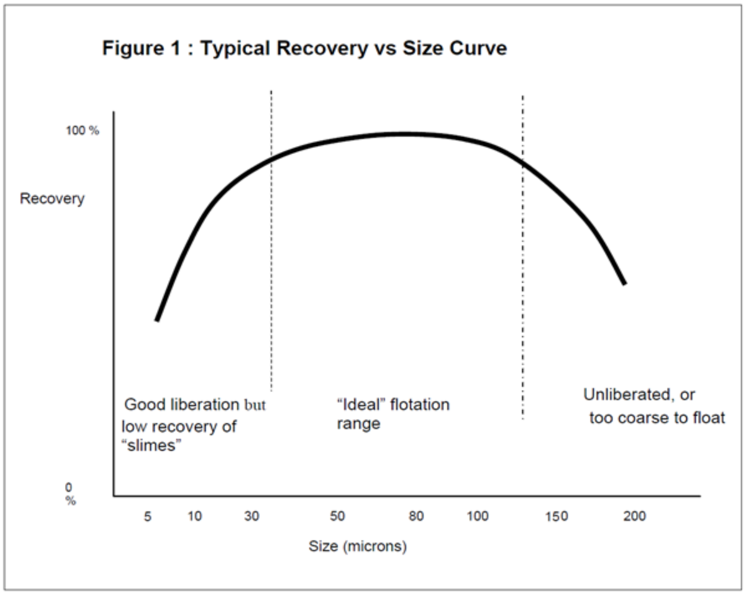


Figure 1.0 Generalized Metal Recovery Verses Size Curve (after Pease 2004)

Historically ore bodies have been mined that are coarse grained and thus give good recoveries in the tank flotation cell size range. If the ore is more fine grained or highly disseminated, operations have been made to be economic with lower recoveries by generally operating at higher throughputs to reduce unit operating costs. It is a generalization that in recent times the "good" orebodies have largely been depleted and those remaining are lower grade, highly disseminated ore bodies that require finer and finer grinding for optimum liberation but yield lower recoveries due to the inherent losses in the tank flotation process. Consequently much attention has been directed to the area of fine and ultra fine metals recovery. This is in both areas of liberation with ultra fine grinding technology and then in the metals subsequent recovery by flotation or other methods.

There have been attempts to improve recoveries with tank flotation by various flowsheets such as split streaming into different size fractions before flotation. At the Mt Keith nickel operation in Western Australia that treats a low grade pentlandite ore the flotation feed is split into coarse, fine and slimes fractions where flotation takes place in three separate lines. Recovery must be economic for the operation to continue but owners reported nickel recovery of only 60% in 1994 the year after start up, increasing to 69% in 2003. Another flowsheet concept is to try and first recover coarse particles by normal flotation tanks at the start of the flotation process followed by additional tanks with higher power inputs to try and recover fines. Both these flowsheets will increase some recovery of fines but at considerable capital cost of the extra equipment.

ALTERNATIVE FLOTATION METHODS

Column Flotation

Commercially developed in the late 1970's, column flotation offers high selectivity over tank flotation that can give higher grade concentrates. However, columns, and their derivatives that include internal or external sparging, like all counter-current flotation mechanisms suffer from low unit metal recoveries. This generally makes them unattractive for roughing and main stream flotation recovery applications.

Pneumatic Flotation

Pneumatic flotation started to be commercialized in the early 1980's and originates out of research work done by Professors Bahr and Simonis in Germany. If nomenclature is simplified by "tank" flotation and "column" flotation, pneumatic flotation can be described as "pipe" flotation.

IMHOFLOT PNEUMATIC FLOTATION

The history and development of Imhoflot pneumatic flotation has been well documented in many technical papers (Brown 2001, Imhof 2003, Battersby 2005, Sanchez-Pino 2009) and so an overview will only be given here. Maelgwyn Mineral Services (MMS) has two different types of pneumatic flotation cells. These are the vertically fed (V-Cell) and tangentially fed (G-Cell). Pneumatic flotation differs from conventional flotation in that the bubble particle contact takes place outside of the cell itself, within the aerator. The associated pulp then enters the cell, or a better nomenclature would be the froth separation tank as the mineralisation of bubbles has now already taken place in the aerator. Separation of these two vitally important aspects of flotation namely bubble particle contact and mineral collection has the advantage that both can now be individually optimised rather than having a trade off situation inherent to conventional flotation. The method of self aspiration of Imhoflot flotation is based on the well known venturi principal. However, unlike other pneumatic flotation systems it is not just a simple venturi. Its patented design has a complex system of nozzles, impingement plates and gas hold up mechanisms that generate a spectrum of bubble sizes from pico through nano to milli sizes. Made from advanced ceramic materials the aerators are non blinding or blocking and do not allow the build-up of scale. There are no moving parts in the Imhoflot cell where all the energy for mineral collection comes from the feed pump, with collection/flotation occurring in milliseconds. There is an order of magnitude utilization of this energy in the collection zone compared with tank flotation.

In Imhoflot pneumatic flotation the residence time of any particle within the aerator is a function of the feed slurry velocity which is normally kept within the range 15-18m/sec. This results in a very short residence time (flotation time) in the aerator of a few milliseconds. However, importantly the majority of the feed pump energy is now focused into the restricted volume of the aerator. The net effect of this is very high air utilisation rates up to an order of magnitude higher than conventional flotation (Hay 2009). The ensuing enhanced kinetics are key to the superior performance of the Imhoflot pneumatic flotation process. In terms of cell geometry for both cell types the aerator is common. However the design of the separating zone differs for each cell.

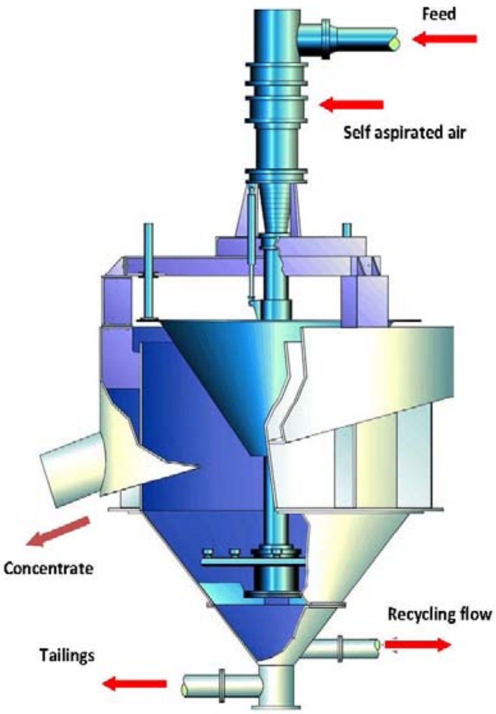


Figure 2.0 Schematic of an Imhoflot V-Cell

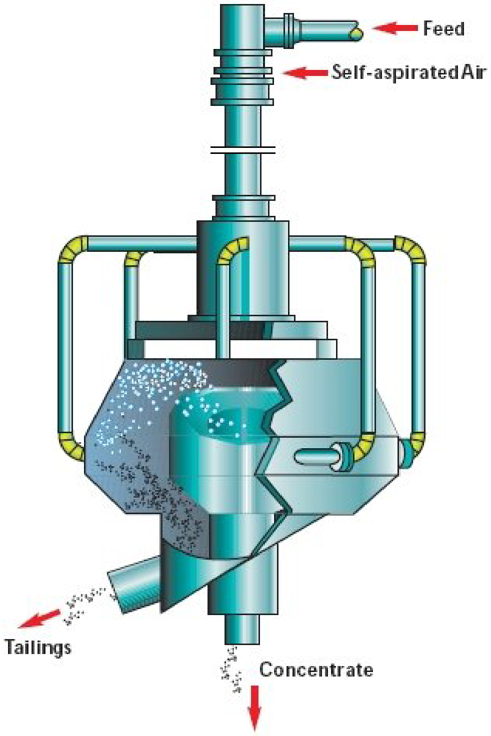


Figure 3.0 Schematic of an Imhoflot G-Cell

In the vertical type – Imhoflot V-Cell (Figure 2.0) the separating zone is essentially a vertical cone. Slurry is injected through nozzles upwards from the base of the cone i.e. co-current with bubble movement. There is no rotor-stator mechanism as in a conventional cell as the bubble particle contact has already taken place in the aerator resulting in a lower energy requirement. Sizing of the Imhoflot V-Cell vertical type is dictated by the time required for efficient froth separation and corresponds to approximately two and a half minutes (Imhof, 1993) contributing to an overall very small footprint.

The Imhoflot G-Cell (Figure 3.0) was developed after the Imhoflot V-Cell. Due to the bubble-particle contacts happening instantaneously in the aerator, collection is complete in the pipe system. The cell is just for separation of the mineral rich bubbles from the tailings gangue. This can be optimized completely independently of the flotation process and energy is not required to maintain the pulp in suspension or generate more collisions. The G-Cell was developed that embraces centrifugal froth separation. It accelerates the separation of fine and ultra fine particles. It has much shorter residence times of between 30 and 60 seconds, depending on the size of cell (Battersby 2005). This equates to an even smaller froth separation vessel and consequently plant foot print. The shorter residence time is thought to reduce gangue entrainment in the froth phase. Whilst mineral recovery as a function of size varies across different mineral classes the Imhoflot V-Cell is generally used for normal and coarse particle flotation, with the Imhoflot G-Cell being favoured for very fine to normal sized particles.

RECOVERY OF ULTRA-FINES

Imhoflot has been installed commercially in plants treating a wide range of minerals such as gold sulphides, base metals, iron ore, coal, potash, industrial minerals etc. where it is believed enhanced recovery was obtained in the ultra fines by the nature of Imhoflot. However no studies have been undertaken confirming this. The work on these two pilot plant operations therefore specifically related to the recovery of ultra-fine material.

CASE STUDY 1

AGUABLANCA NICKEL MINE, SPAIN

The Aguablanca nickel – copper mine in Spain had identified a problem with high nickel losses to the tails. Previous QEMSCAN mineralogical analysis had indicated that these losses were predominantly associated with the very fine fractions. MMS were contracted to run a three stage Imhoflot G-Cell pilot plant on various streams of the plant to demonstrate the abilities of Imhoflot to the owners of the mine and to determine if any improvements could be made to the Aguablanca circuit. This pilot plant work included treating the final tails of the plant as previous work had indicated the G-Cells ability to recover metal fines not recovered by conventional tank flotation. However, prior to the decision to test Imhoflot, Aguablanca had already determined that the losses may have been attributable to a simple residence time constraint and had therefore purchased new tank flotation capacity in the form of four, 30m³ Outotec tank cells. After the commissioning and optimisation of four new tank cells the G-Cell pilot plant was again run on the new tailings.

BACKGROUND

The Aguablanca nickel – copper mine which is part of the Lundin Mining group is situated approximately 70 Kilometres north of Seville in Spain. Flotation testwork was carried out on the mine using a 3x G12 (1.2m diameter) Imhoflot G-Cell pilot plant. The initial objective of the test work was to investigate whether the G-Cells could increase nickel recovery to above the current 82%. QEMSCAN analysis of plant tailings, shown below, indicated significant amounts of liberated nickel sulphides in the minus 9 micron fraction which were not being recovered by the plants tank cells.

The Aguablanca Mine treats material from the Monaguera nickel- copper sulphide orebody. The open cast mine produces about 160,000 tpm of run of mine (ROM) material which is treated by a 260 tph flotation plant (up -rated from the original 190 tph plant).The head grade is highly variable but is approximately 0.6% nickel, 0.4% copper, with the sulphide mineralisation being pyrrhotite, chalcopyrite, pyrite and the nickel – iron sulphides, pentlandite and violerite, and the nickel sulphide millerite. Trace amounts of nickel also being present in the pyrite and pyrrhotite with the pyrite averaging 1.15 % Ni.

ROM material is fed through a primary jaw crusher producing a d80 of 100mm followed by a secondary cone crusher with a d80 of 50mm.The grinding circuit is made up of a SAG mill with pebble cone crusher followed by a ball mill, the cyclone overflow being 80% minus 90 µm. A flowsheet of the flotation plant is given in Figure 5.0 and the G-Cell pilot plant in Figure 4.0 below. The flotation circuit was originally designed to produce separate nickel and copper concentrates. These are however now mixed to form a bulk sulphide concentrate averaging 7% Ni and 6% Cu. Rougher flotation consists of 11x $40m^3$ Wemco tank cells split into four sections, the first two cells are referred to as pre-float as they were originally designed to float off talc (MgO) which is a major contaminant, attracting smelter penalties if above 5% in the final concentrate. These are now used as copper roughers together with a third cell, which was originally intended as the only copper rougher. This concentrate is cleaned in 4 x 12 m^3 tank cells becoming part of the final concentrate, the cleaner tailings reporting back to the first nickel cleaner cell.

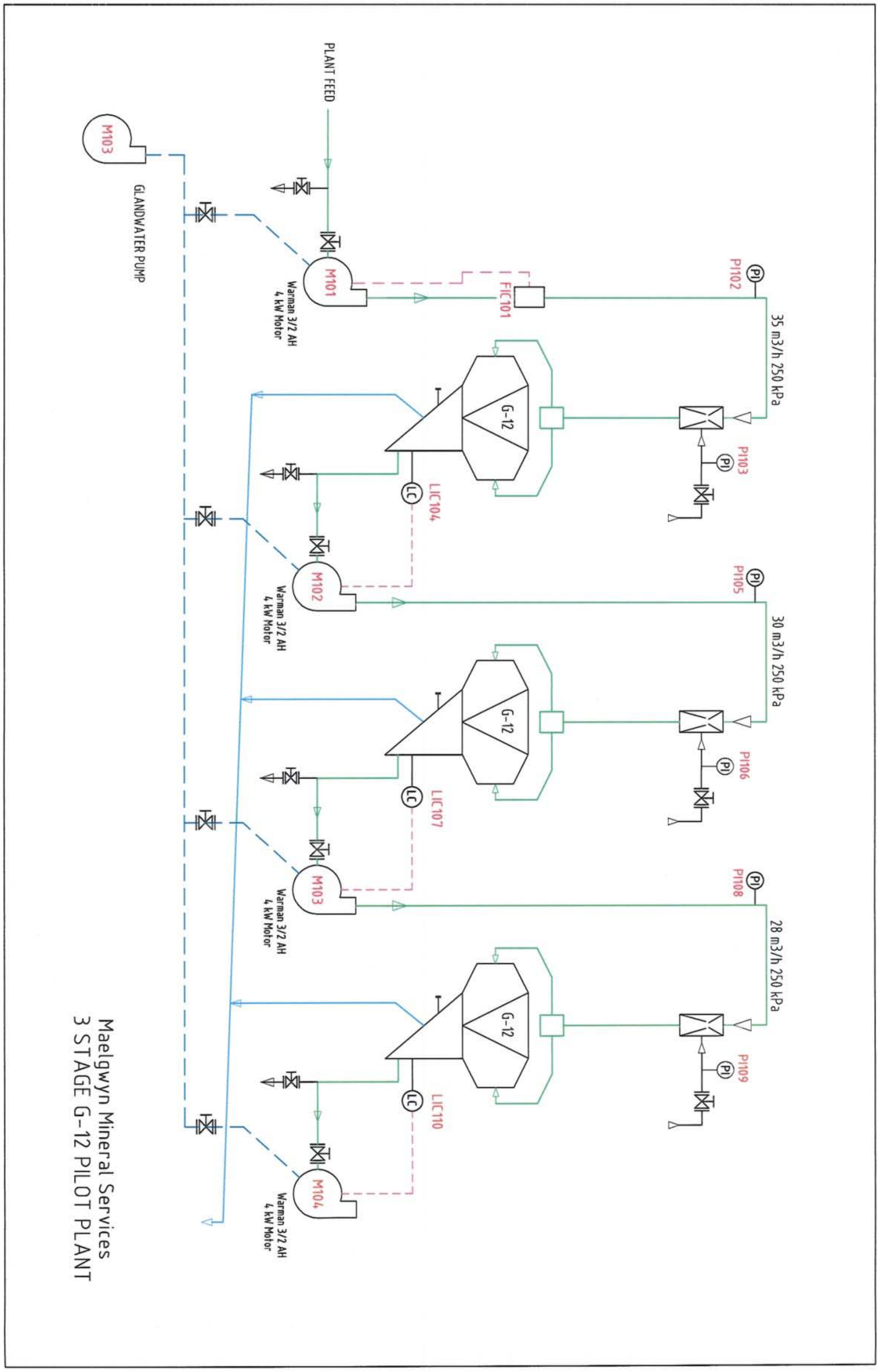
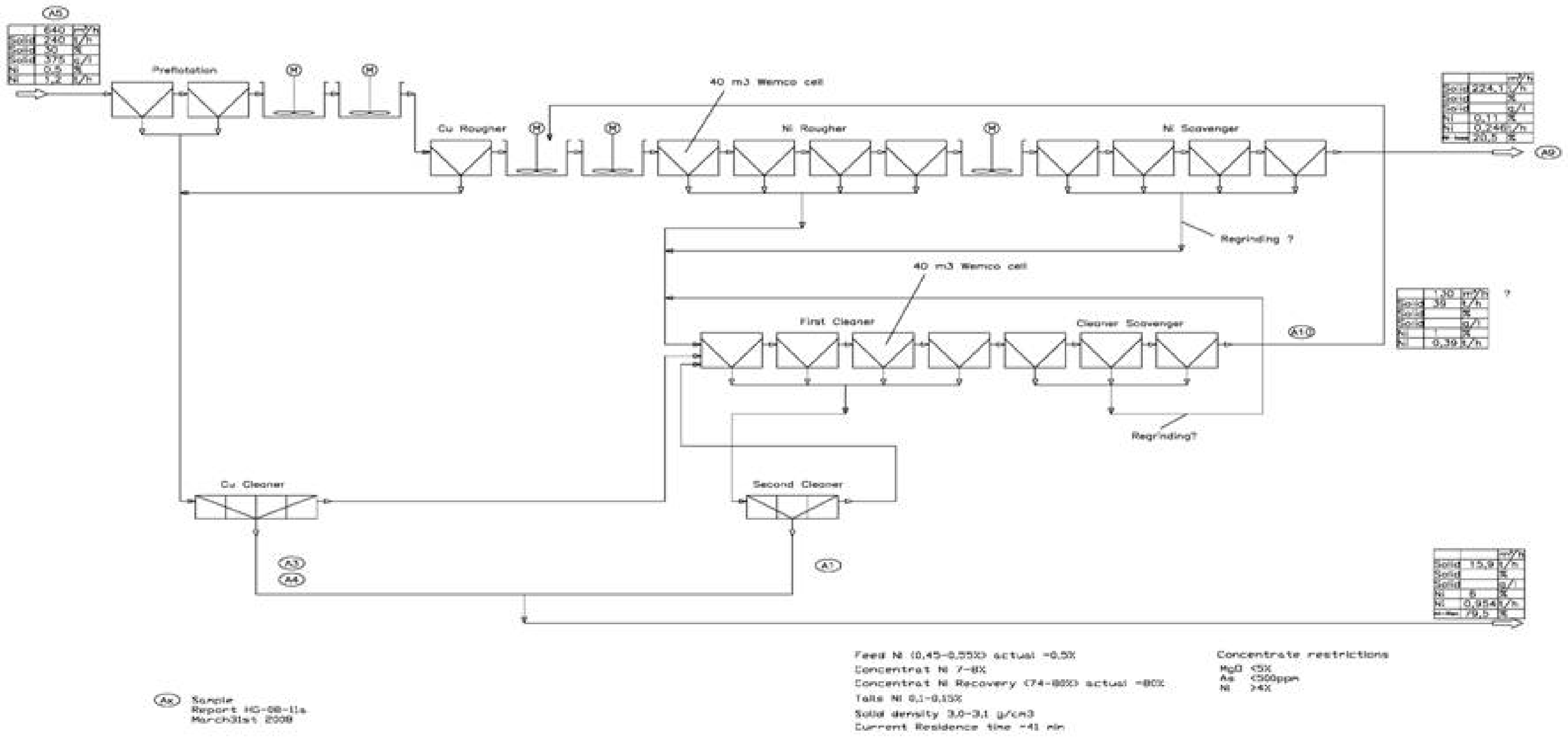


Figure 4.0 Imhoflot 3 stage G-12 Pilot Plant Flowsheet

Figure 5.0 Aguablanca Plant Flowsheet



Feed Ni (0,45-0,55%) actual ~0,5%
Concentrat Ni 7-8%
Concentrat Ni Recovery (74-80%) actual ~80%
Tails Ni 0,1-0,15%
Solid density 3,0-3,1 g/cm3
Current Residence time ~41 min

Concentrate restrictions
MgO <5%
As <500ppm
Ni >4%

(Ax) Sample
Report HG-08-11s
March31st 2008

The concentrates of four nickel roughers and four nickel scavengers feed the first, then second nickel cleaners, the nickel scavenger tails represent the final tails and was the G-Cell pilot plant feed. The first cleaner tails feeds the scavenger cleaner whose concentrate is recycled back to the first cleaner, cleaner scavenger tails reports back to the nickel roughers. The second nickel cleaner tails return to the first cleaner.

Due to the complex and variable mineralogy of the ROM feed together with throughput changes the reagent suite is varied on a day to day basis but the main collector used is a Xanthate mix (67% SIBX , 33% PAX- 140g/t) added in stages where required. Aero 3894 (25g/t) is added at the cyclones as a copper promoter together with Aero 412 (10g/t) some of which is added down the rougher circuit. Copper sulphate (33g/t) is added in the nickel roughers and scavengers as an activator for pyrite, pyrrhotite and the nickel sulphides. Large amounts of a talc specific depressor mix (70% Depramin, 30% Finnfix BW,-1000g/t) is added sequentially throughout the circuit, as is a frother mix (currently 33% Oreprep 591, 33%MIBC, 33% Pine Oil- 120g/t). The rougher circuit is normally run at pH 8.5, whilst the cleaners are run at up to pH 11.5.

Various mineralogical testwork was performed by Aguablanca on a composite sample of the current plant tailings including mineral distribution versus size and liberation analysis by QEMSCAN. The metal distribution according to size results are tabulated in Figure 6.0 below and indicate that approximately 50% of the nickel in the tailings is associated with ultrafine particles i.e. minus 9 microns. The QEMSCAN testwork results are summarised in Figure 7.0 which essentially illustrates a plot of the percentage of the particles for each size fraction against their percentage liberation. For example, for the +75 micron size fraction approximately 65% of the particles are 10% liberated and the remaining 35% are 20% liberated. As expected as the size fractions become finer a higher degree of liberation is apparent but this is limited to approximately 30% liberation part from the minus 9 micron fraction. The conclusion from all of the above work is that there are liberated nickel particles which can be recovered but are not recovered by the existing tank flotation.

Product (µm)	Weight (%)	Assay				Distribution			
		Ni (%)	Cu (%)	Fe (%)	S_{TOT}(%)	Ni	Cu	Fe	S_{TOT}
+ 75	35.48	0.069	0.045	2.85	0.71	19.6	41.0	24.3	25.7
-75+38	21.32	0.097	0.029	5.31	1.73	16.6	15.9	27.2	37.7
-38+9	22.81	0.072	0.018	3.39	0.91	13.2	10.6	18.6	21.2
-9	20.39	0.310	0.062	6.12	0.74	50.6	32.5	30.0	15.4
Head	100.00	0.125	0.039	4.164	0.979	100.00	100.00	100.00	100.00
Assayed Head		0.120	0.039	4.310	1.120				

Figure 6.0 Analyses of Aguablanca Tailings Size Fractions

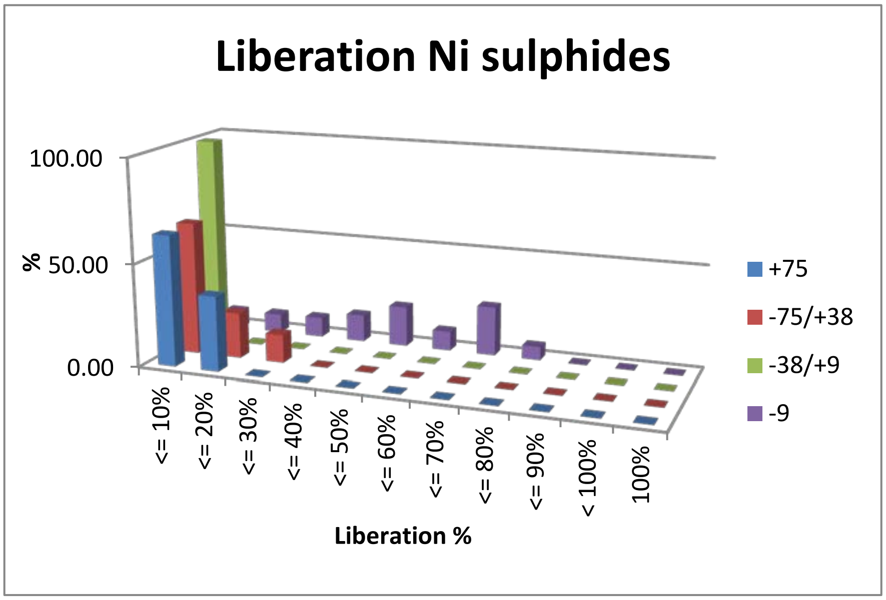


Figure 7.0 Graphic Showing the Liberation Characteristics of Ni Sulphides Versus Size

DESCRIPTION OF THE IMHOFLOT G-CELL PILOT PLANT

The G-Cell pilot plant is illustrated in Figures 8.0 and 9.0. Each G-Cell incorporates a self-aspirating aerator, normally operating between 1.5 and 2.5 bar gauge pressure. This provides highly dispersed fine bubble froth, with relatively low energy input. The feed pump feeds the aerator, which discharges to the distributor and in turn the four tangential separator nozzles. Froth separation takes place as the fluid and froth spin in the outer bowl of the cell, with flow of froth to the central froth funnel. All feed pumps are equipped with frequency converters. The speed of the first pump can adjusted according to the desired flow rate and the pressure at the aerator. The second pump speed is then automatically controlled by the level of the pulp of the first G-Cell. The third and fourth pumps are controlled in the same manner. The airflow to the aerators can be adjusted by a manual valve, which throttles the flow to the self aspirating aeration units. A pressure gauge in front of the aerator indicates the vacuum. The pulp level in the separation part of the G-Cell is measured by level probes and controlled by the pulp flow to the next cell. The froth depth can be varied by means of the level control probes pressure sensors and a separately controlled movable central cylinder whose top is the froth launder. Consequently the froth height can be controlled above the fixed pulp surface. The pilot plant was controlled by means of a laptop connected to a Siemens PLC system with readings of throughput, pump speeds, aerator pressures, venturi suction vacuums, and pulp depths being shown. The main variables that can be used by the operator to adjust performance are, feed pressure and volume, pulp relative density (solids content), aeration rate, and froth depth.

The pilot plant has a footprint of approximately 5500mm x 2100mm and an installed height of approximately 3600mm. The diameters of the cells are 1.2m and the plant requires 3 phase, 400V 40KW power. The plant is designed to fit into a standard 20 foot container for transportation.

Fig 8.0 Schematic of Imhoflot 3 Stage G 12 Pilot Plant

Figure 9.0 Imhoflot G1.2 Three Cell Pilot Plant at Aguablanca

TESTWORK

The testwork was carried out on pulp piped from the two discharge columns of the last Wemco nickel scavenger cell. The pulp was gravity fed through two HDPE tubes which joined together through a Y piece and then passed over a feed box containing a 5mm tramp screen before being connected directly to the inlet of a Warman 3/2 feed pump. In addition there was an inline screen situated on the 1st G-Cell feed line to stop tramp material entering the aerator. This could easily be opened for cleaning.

The pilot plant was designed with a maximum throughput of 35 m^3/hr (flowmeter limit) and a minimum of 20 m^3/hr, with feed control through ball valves on the plant feed pipes in front of the feed box. The speed of the first pump is set at the PLC which then controls the aerator pressure in the first cell. The venturi suction pressure for each cell is controlled manually by adjusting the inlet valve next to the aerator. Each inlet had its own control valve.

Each test run was conducted after a start up procedure during which the plant was started and flowrates, pump speeds, and pulp levels slowly increased until steady state is achieved at the required test setting. During test runs the following samples were taken every 15 minutes; feed, tailings, concentrate from each cell and a composite concentrate sample taken from the main launder. All sensor readings were recorded, together with reagent additions, pulp density readings taken using a Marcy scale, and composite concentrate flowrate. Sampling continued as long as the plant maintained steady state. One of the major problems encountered during the test work was that the main plant feed was sometimes highly erratic in both flowrate and percent solids which made maintaining a steady state in the pilot plant difficult. The averaged results of the first set of testwork are summarised in Figure 10.0 below.

Feed			Concentrate			Tailings			Recovery	Mass Pull
Ni %	Cu %	MgO %	Ni %	Cu %	MgO %	Ni %	Cu %	MgO %	Ni %	%
0.102	0.037	10.95	1.227	0.206	8.44	0.072	0.032	11.12	29.9	2.6

Figure 10.0 - Averaged Results of the 1st Final Tailings Testwork.

The first set of tailings testwork ran for approximately three weeks on days when the main plant was in operation and the feed was relatively stable. The testwork was then halted whilst the Aguablanca mine installed the four new 30 m^3 tank cells as final tailings scavengers. The detailed results of the initial tests have been summarised in Appendix 1.0. Accepting the varying feed conditions and pilot plant test settings the G- Cell pilot plant was able to recover approximately 30% of the nickel content of the tailings at a mass pull of 2.6%. This would represent an over 5% increase in overall nickel recovery for the operation. The corresponding Nickel concentrate grade was 1.2%. Given the G-Cells well known ability to recover fines which are not recovered by tank flotation in conjunction with the previously provided tails size versus grade information it was considered highly probable that a high proportion of the recovered nickel in the G-Cells is associated with the fines not being recovered by tank flotation. However there was the possibility that the G-Cell pilot plant was recovering the extra nickel due to a shortage of residence time in the main plant rather than being a unique feature of Imhoflot. As previously mentioned Aguablanca mine had already thought that increased residence time in the current circuit would increase recovery. The new tank cell scavenger cells increased residence time by 31% which together with a planned move to cut feed from 260tph to 175tph to extend the mine life increased residence time by 95%. This however did not result in the expected higher recoveries.

Further G-Cell pilot plant testwork on the "new" plant final tails or rather extended roughing time tailings produced average recoveries of 25 % at relatively low mass pulls indicative of possible still higher recoveries. Results are summarised below in Figure 11.0 and detailed in Appendix 2.0.

Feed			Concentrate			Tailings			Recovery	Mass Pull
Ni %	Cu %	MgO %	Ni %	Cu %	MgO %	Ni %	Cu %	MgO %	%	%
0.100	0.029	13.48	0.569	0.127	14.41	0.077	0.027	13.40	25.9	4.9

Figure 11.0 - Averaged Results of Final Tailings Testwork After New Tank Cells Commissioned.

Samples of pilot plant feed, concentrate and tailings were taken and subjected to Cyclosizer size classification followed by assay of the fractions. The results of these tests are shown in Figure 12.0 below, and appear to confirm previous Aguablanca testwork indicating that a large majority of the nickel present in the final plant tails (pilot plant feed) is in the minus 9µm fraction. A significant proportion of this material is recovered by the G-Cells. The results are also shown graphically in Figure 13.0.

	Calculated Feed (100%)			Concentrate (5%)			Tailings (95%)		
Size Fraction	Mass %	Grade % Ni	Dist %	Mass %	Grade % Ni	Dist %	Mass %	Grade % Ni	Dist %
+44µm	16.24	0.04	100.00	0.28	0.541	21.33	15.96	0.04	78.67
+33µm	27.93	0.03	100.00	0.19	0.315	6.81	27.74	0.03	93.19
+23µm	12.45	0.03	100.00	0.29	0.138	10.09	12.16	0.03	89.91
+15µm	5.47	0.03	100.00	0.28	0.091	14.47	5.19	0.03	85.53
+11µm	6.56	0.04	100.00	0.48	0.097	20.34	6.08	0.03	79.66
-11µm	31.35	0.16	100.00	3.49	0.544	38.01	27.87	0.11	61.99
Head	100.00	0.07	100.00	5.00	0.444	30.14	95.00	0.05	69.86

Figure 12.0 Table Indicating Percentage Nickel in Each Size Fraction

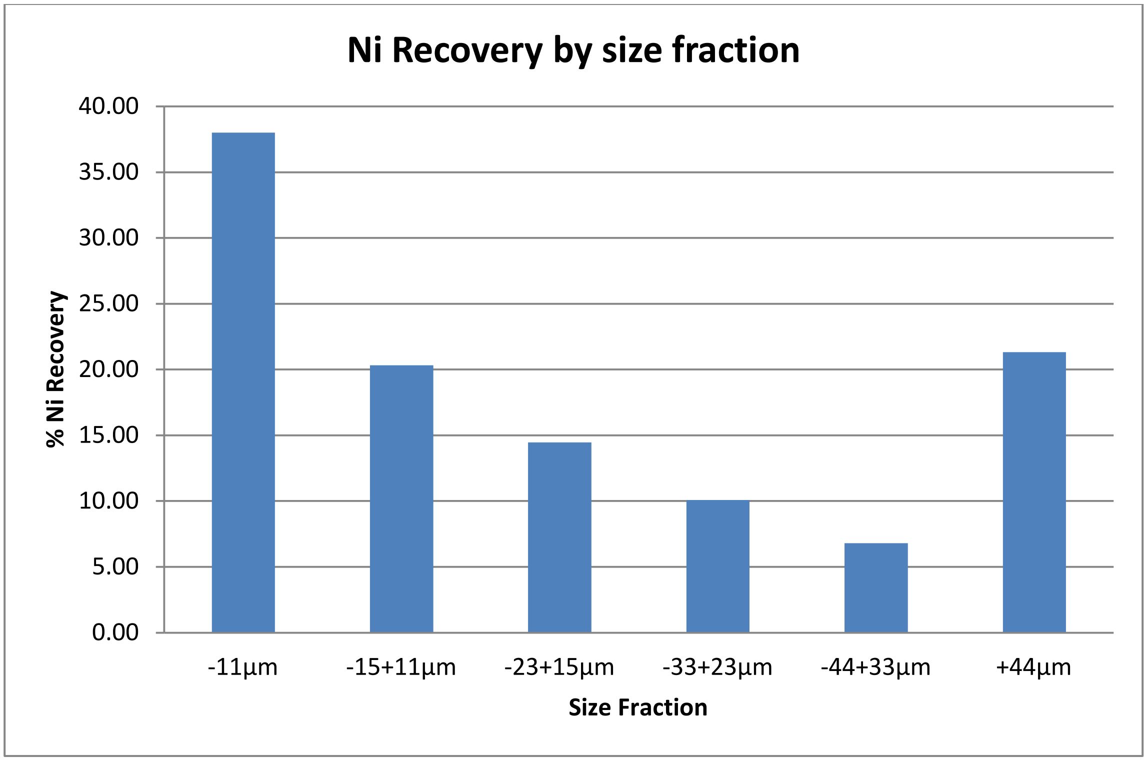


Figure 13.0 Nickel Recoveries by Size Fraction in Concentrate for G-Cell Pilot Plant Testwork

In December 2010, during the pilot plant testwork, heavy rain in the area caused a major pit wall slippage at the mine. This halted production and consequently any further pilot plant testwork at the operation. The plant is currently on care and maintenance. The mine owners advise that they plan to start production again at the end of 2012.

One of the issues identified with the ability of pneumatic flotation to recover ultra fines from the final tails is that they would be lost again if put back into the existing tank flotation cleaner circuit. The benefit of extra ultra fines recovery by such scavaging would only be realised if it was cleaned in a bespoke Imhoflot cleaner circuit that could produce a product that could be blended into the final concentrate. This was one of the proposed tasks of the pilot plant testwork which was put on hold by the mine closure. However

Imhoflot is also efficient in cleaning applications so it thought that a high grade concentrate would be produced with acceptable impurity limits.

CASE STUDY 2

To further highlight the abilities of pneumatic flotation in recovering ultra-fines a second pilot plant study is reported here which details results of treating a cleaner circuit tail that was a final plant tail.

BACKGROUND

This mine exploits one of the words largest zinc/lead deposits. The treatment plant produces a bulk concentrate with a required minimum zinc grade of 40%, there are penalties for excessive iron and silica in the concentrate, and with the silica at a threshold of 3.7% the most important. The finely disseminated nature of the mineralisation requires an ultrafine grind of below 7 microns to achieve liberation and to reduce entrainment of silica. This is achieved using six, 1.1 MW ISA mills. Significant amounts of zinc are lost to tailings from the cleaner circuit, parts of which are open circuit, as ultra fines, and the objective of this testwork was to determine whether any of the cleaner tailings streams could be retreated by the G-Cell pilot plant to produce a final grade concentrate.

Processing Plant

The Processing plant treats approximately 1.8M tons of high grade ore per year. The head grade averages 12.6% Zinc, 5.5% Lead, and 57ppm Silver, and the plant produces about 320,000 tons of bulk flotation concentrate per year at an average 45% zinc. The crushing circuit product feeds the milling circuit which consists of a 4MW SAG mill in closed circuit with a 935kW Vertimill. The primary cyclones are set to produce a 75µm product with the underflow feeding the Vertimill. The rougher flotation circuit consists of 4 x 16m^3 tank cells, with the concentrate being fed to cyclones with a set point of 7µm, the overflow of which feeds Cleaner 3. The cyclone underflow is fed to 6 x 1.1 Mw ISA mills in closed circuit with another 7µm cyclone pack the overflow of which feeds Cleaner 1. The concentrate from Cleaner 1 is fed to Cleaner 2, the concentrate from which together with that from Cleaner 3 feeds Cleaner 4.The tailings from Cleaners 1 to 4 report to final tailings. Cleaners 5 to 7 are in close circuit with the concentrates reporting to the next cleaner and tailings reporting to the previous one. Cleaner 7 producing the final concentrate. The zinc grade of the concentrate averages 45% at 13% moisture with a minimum smelter requirement of 40% this would allow for a degree of blending with G-Cell concentrates if necessary. The basic plant flowsheet is given in Fig 14.0 below.

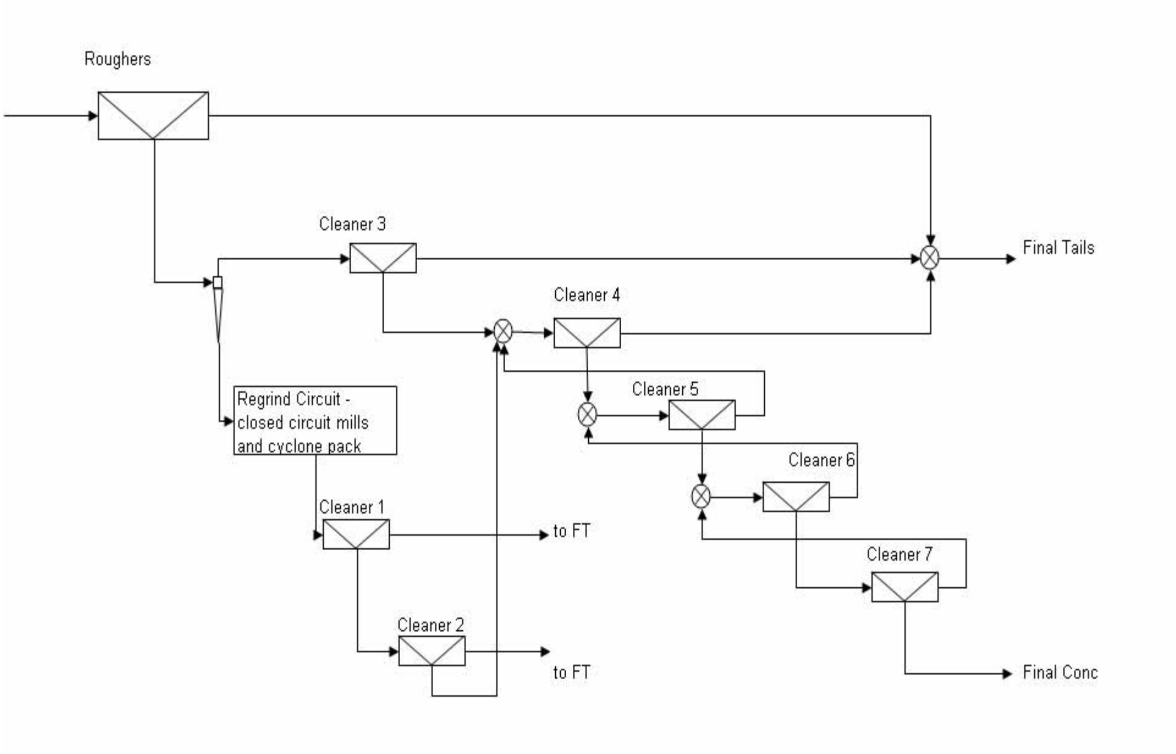


Fig 14.0 Flotation flowsheet

PILOT PLANT AND TESTWORK DESCRIPTION

This pilot plant was a two stage self contained unit consisting of two, G10 (1m diameter) Imhoflot G-Cells. The pilot plant is shown in Figure 15.0. The units operated in series with the tailings of the first G-Cell feeding the second G-Cell. The pilot plant is able to treat up to 23 m^3/hr. For this testwork the feed percent solids was very low and therefore the plant operated at around 1 t/hr during the tests. Testwork was performed upon several of the cleaner streams but results are only reported for the Cleaner 4 tailings stream as this stream currently reports to final tailings with the results indicating that a product suitable for blending into the final concentrate can be produced. Zinc recovery varied from 9.2% to 20.2% with associated grades of 37.9% and 35.6% zinc. The results are summarised in Figure 16.0.

Figure 15.0 Imhoflot G10 Two Cell Pilot Plant in Operation

With this extra recovery from a final tails a simple payback exercise indicated a payback of between 4 and 9 months depending upon the assumptions used. The high upgrade of zinc and downgrade of silica at 3.7 and 2.7 respectively indicate that at least some of the zinc is found as liberated material, probably ultra fines, as it would be impossible to achieve these results from middlings. This would also explain the inability of the current cells to adequately treat this material. It should also be noted that there were no reagent additions made during these tests.

	Zinc Feed	Zinc Recovery	Zinc Grade	SiO_2 Feed	SiO_2 Grade	Mass Pull
	(%)	(%)	Conc (%)	(%)	Conc (%)	(%)
GA1	10.40	9.2	37.91	12.61	4.69	2.5
GA2	9.91	20.2	35.61	13.67	5.16	5.6

Fig 16.0 Selected Results From Cleaner 4 Tailings Stream

The results from individual cells shown below in Figure 17.0 infer that there is not a significant drop in zinc grade or rise in silicon grade between cells in either test which indicates that a third cell could be added to increase recovery without seriously affecting the combined grade. It is also likely that a third cell would enable the mass pulls to be better ballanced to optimise the grade recovery relationship.

Test	Cell 1			Cell 2		
	Zinc Recovery %	Zinc grade Conc %	SiO2 grade Conc %	Zinc Recovery %	Zinc grade Conc %	SiO2 grade Conc %
GA1	5.7	38.1	4.7	3.5	37.6	4.7
GA2	13.9	36.3	4.9	6.3	34.2	5.7

Fig 17.0 Breakdown of Results on an Individual Cell Basis

CONCLUSIONS

The testwork case studies confirmed the well documented inefficiency of conventional tank flotation in the recovery of ultrafine particles. In contrast Imhoflot G-Cell pneumatic flotation at Aguablanca was able to recover up to 30% of these fine particles from existing plant tailings. It did this from plant tailings that had already been processed by two different manufacturer's types of tank flotation cells with an inordinate amount of excess flotation residence time in the circuit.

Whilst the above recovery improvements alone may provide sufficient justification for the consideration of G-Cells for new base metal flotation circuits, additional benefits may accrue for any flotation circuit where the valuable component is poorly liberated when G-Cells are combined with ultra fine grinding. Further testwork is still required to confirm this, however given their small footprint, high throughput and other operational benefits Imhoflot G-Cell technology clearly represents a major breakthrough in flotation plant circuit design.

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Appendix 1.0 Sampling Campaign 1. G-Cell Pilot Plant Flotation of Existing Aguablanca Plant Tailings

RUN	Feed				Concentrate				Tailings				Recovery Ni %	Mass pull % Pull %
NUMBER	solids %	Ni %	Cu %	MgO %	Solids %	Ni %	Cu %	MgO%	Solids %	Ni %	Cu %	MgO%	Ni %	
RUN 1	36.9	0.109	0.039	11.25	19.4	1.385	0.263	6.66	31.0	0.060	0.024	11.78	47.0	3.7
RUN 2	35.1	0.116	0.036	11.13	15.1	1.167	0.182	8.27	30.7	0.046	0.016	11.37	62.8	6.2
RUN 3	38.4	0.071	0.025	10.20	10.3	1.592	0.232	5.84	36.1	0.061	0.023	10.26	14.6	0.7
RUN 4	37.1	0.068	0.031	11.11	8.8	1.282	0.200	8.64	35.9	0.057	0.032	10.63	16.9	0.9
RUN 5	37.0	0.118	0.042	11.56	20.5	1.392	0.213	8.60	31.5	0.091	0.065	11.51	24.5	2.1
RUN 6	37.2	0.087	0.039	11.49	12.3	1.377	0.262	11.85	33.1	0.066	0.032	11.49	24.4	1.6
RUN 7	33.8	0.085	0.029	10.95	14.2	0.964	0.137	8.24	24.0	0.070	0.025	11.16	19.0	1.7
RUN 8	34.6	0.078	0.042	12.31	12.0	1.082	0.259	11.6	39.8	0.067	0.037	11.45	15.0	1.1
RUN 9	34.9	0.132	0.047	11.68	20.2	1.181	0.263	9.07	31.3	0.104	0.036	11.69	23.3	2.6
RUN 10	35.3	0.074	0.033	11.49	13.1	0.718	0.191	12.82	35.6	0.060	0.031	11.47	20.6	2.1
RUN 11	39.4	0.107	0.032	10.81	13.4	0.983	0.161	8.43	34.9	0.083	0.026	10.93	24.5	2.7
RUN 12	35.3	0.085	0.039	8.46	12.2	1.343	0.237	6.39	34.6	0.064	0.034	11.01	25.9	1.6
RUN 13	36.3	0.094	0.048	11.15	15.0	1.264	0.251	6.73	36.3	0.068	0.042	10.49	29.2	2.2
RUN14	37.3	0.077	0.036	11.40	13.8	1.256	0.257	7.23	37.1	0.053	0.035	10.76	32.5	2.0
RUN 15	39.3	0.077	0.031	10.04	15.1	1.079	0.164	10.23	34.0	0.037	0.020	10.33	53.8	3.8
RUN 16	32.7	0.101	0.032	10.21	16.0	1.268	0.154	6.54	30.9	0.081	0.029	10.64	21.2	1.7
RUN 17	31.2	0.125	0.042	10.19	13.7	1.347	0.242	6.61	28.3	0.093	0.038	10.98	27.5	2.6
RUN 18	34.2	0.122	0.042	10.80	18.5	1.331	0.165	5.82	29.7	0.069	0.035	11.52	45.8	4.2
RUN 19	31.4	0.097	0.033	11.45	23.8	1.236	0.177	9.87	29.5	0.079	0.029	11.11	19.8	1.6
RUN 20	29.7	0.141	0.039	11.27	15.4	1.282	0.149	8.54	27.3	0.085	0.025	11.74	42.5	4.7
RUN 21	35.5	0.136	0.041	11.76	17.9	1.021	0.168	10.00	32.0	0.112	0.033	12.23	19.8	2.6
RUN 22	31.1	0.139	0.040	10.10	19.8	1.438	0.196	7.68	28.9	0.077	0.033	10.00	47.1	4.6
Average	35.2	0.102	0.037	10.95	15.5	1.227	0.206	8.44	32.4	0.072	0.032	11.12	29.9	2.6

Appendix 2.0 Sampling Campaign 2. G-Cell Plant Flotation of Aguablanca Tails After Extension of Plant Roughing Time

RUN No	Feed				Concentrate				Tailings				Recovery Ni %	Mass Pull Ni %
	Solids %	Ni %	Cu %	MgO %	Solids %	Ni %	Cu %	MgO %	Solids %	Ni %	Cu %	MgO %	Ni %	Pull %
RUN 1	32.7	0.101	0.032	10.21	16.0	1.268	0.154	6.54	30.9	0.081	0.029	10.64	21.2	1.7
RUN 2	31.2	0.125	0.042	10.19	13.7	1.347	0.242	6.61	28.3	0.093	0.038	10.98	27.5	2.6
RUN 3	34.2	0.122	0.042	10.8	18.5	1.331	0.165	5.82	29.7	0.069	0.035	11.52	45.8	4.2
RUN 4	31.4	0.097	0.033	11.45	23.8	1.236	0.177	9.87	29.5	0.079	0.029	11.11	19.8	1.6
RUN 5	29.7	0.141	0.039	11.27	15.4	1.282	0.149	8.54	27.3	0.085	0.025	11.74	42.5	4.7
RUN 6	35.5	0.136	0.041	11.76	17.9	1.021	0.168	10.0	32.0	0.112	0.033	12.23	19.8	2.6
RUN 7	31.1	0.139	0.040	10.10	19.8	1.438	0.196	7.68	28.9	0.077	0.033	10.00	47.1	4.6
Average	35.2	0.102	0.037	10.95	15.5	1.227	0.206	8.44	32.4	0.072	0.032	11.12	29.9	2.6