

Sulphide mineral flotation
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Titania AS 30/01/2009



Preface:

The research project described in this report was conducted in January 2009 at Titania AS in Norway. I first came into contact with Titania AS when they had a presentation at the Norwegian University of Science and Technology (NTNU). I was studying at the NTNU for one semester to fulfil a minor as part of my Bachelor degree in Applied Earth Sciences at Delft University of Technology.

I would first like to thank Titania AS for giving me an opportunity to do an internship. Furthermore I would especially like to thank Wolfgang Shubert my supervisor for organising the research project and sharing his expertise and knowledge with me.

The internship at Titania AS has been a great experience. I have learned a lot about conducting research both in practical and theoretical sense. Furthermore I now understand more how an entire mine and processing plant are operated. This new knowledge, I believe will be of great value for me in the future.

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1. Abstract

The report describes a research project conducted by Louis Carlier in January 2009 at Titania AS, ilmenite mine in Norway. This research project focuses on the first testing stage of three, namely: laboratory cell test → pilot cell test → full scale test. The purpose of this research project was to provide information on the flotation behaviour of sulphide minerals using potassium-amyl xanthate as a collector. The effect of Ph, time, and reagent regime were tested using a laboratory test cell. Some pilot cell tests were included to allow a cleaning stage and gain information on the continuous flotation process of sulphide minerals. The results of this project are recommendations for further pilot cell tests in the form of precise testing ranges for: time, Ph, reagent quantities and cleaning stage.

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3. Introduction:

Titania AS is a company which has been producing ilmenite from an open pit mine since 1960. The ore comes from a magmatic dyke deposit. The main minerals forming the deposit are ilmenite 39,9 % and plagioclase 36,9 % (see appendix, Minerals in the Tellnes ore). Sulphide bearing minerals present in the ore include:

- Pyrite
- Pyrrhotite
- Pentlandite
- Violarite
- Siegenite
- Millerite
- Chalcopyrite

In the process of enriching the ilmenite concentrate sulphides are currently separated from the Ilmenite. Sulphides are sold as a by-product. These sulphide bearing minerals also contain the elements nickel, copper and cobalt which have become more valuable throughout the years. The majority of the current world's production of nickel still comes from sulphidic sources.

The process to separate sulphide from the ilmenite is done stepwise. In general, most of the sulphide minerals travel with the ilmenite concentrate during the two parallel enrichment process (see appendix, processing flow chart, page 25): the ilmenite flotation with tall oil and the gravimetric separation. After these processes, the two concentrates are mixed and leached at pH 1 to get rid of the apatite. After the leaching, sulphides are floated to separate them from the ilmenite using Tall oil. However, tall oil residues in the sulphide by-product are not wanted by the customers as it turns the sulfuric acid black after calcinations.

The rising prices and higher quality demand have led to the idea of installing a sulphide floatation in front of the ilmenite floatation. The steps necessary to reach full scale production include: literature investigation, laboratory tests, pilot cell tests, full scale tests. Furthermore many different slurry streams can be tested.

The most suitable streams will be tested first. These are the FCS and FRS respectively Fine Clean Spiral and Fine Rougher Spiral coming from the gravimetric plant (see appendix, gravimetric plant, page 26). Both streams have an ideal grain size distribution of 60 to 150 μm . FCS concentrate has larger sulphide content but a smaller debit of 45 t/h versus 75 t/h for FRS.

The average concentration values in % for both streams are:

<i>feed averages</i>	TiO₂	S	Ni	Cu	Co
FCS	43.58	0.29	0.07	0.02	0.02
FRS	38.32	0.25	0.06	0.02	0.02

The investigations includes the testing of FCS and FRS streams according to their floatation behavior, time periods, pH values and reagents amounts are varied. Indicational pilot tests will also be conducted.

The goal of this research is to narrow down the testing range of different parameters for the Pilot cell tests. The results from the laboratory cell tests are used to gain further understanding of the process and form the base for the Pilot cell tests.

4. Company overview:

Titania AS is one of the largest ilmenite producers in the world. It was founded in 1902 and started continuous ilmenite production from the Storgangen deposit 3 km NE of Hauge in 1917. At this stage Titan CO A/S of Fredrikstad, now Kronos Norge A/S acquired share majority. The company was taken over by National Lead Company (based in the US), now NL Inc, in 1927. NL Inc now owns Titania AS through Kronos A/S.

The current production deposit "Tellnes" (see figure 1) was discovered by an aeromagnetic survey in 1954. Six years later in 1960 production started at the Tellnes deposit. In 1965 the operations at Storgangen were closed down and all the production facilities were transferred to Tellnes. Tellnes now has an annual production of approximately 550000

tonnes of Ilmenite. Except from providing [Figure 2 Tellnes deposit](#)

Kronos A/S with the ilmenite for its internal needs, Titania AS also sells ilmenite to third parties. These further process the black ilmenite to obtain a white pigment used in: paint,

rubber, paper, food, fibres, cosmetics, plastic. About 95% of the titanium is used as a white pigment in the form of titanium oxide. The remaining 5% are used in the metal industry. Although Titanium dioxide is often more expensive than other conventional pigments it has some favourable characteristics. It is non-toxic and has a superior quality in: whiteness, opacity, refractive index, light-scattering properties and UV-absorption.

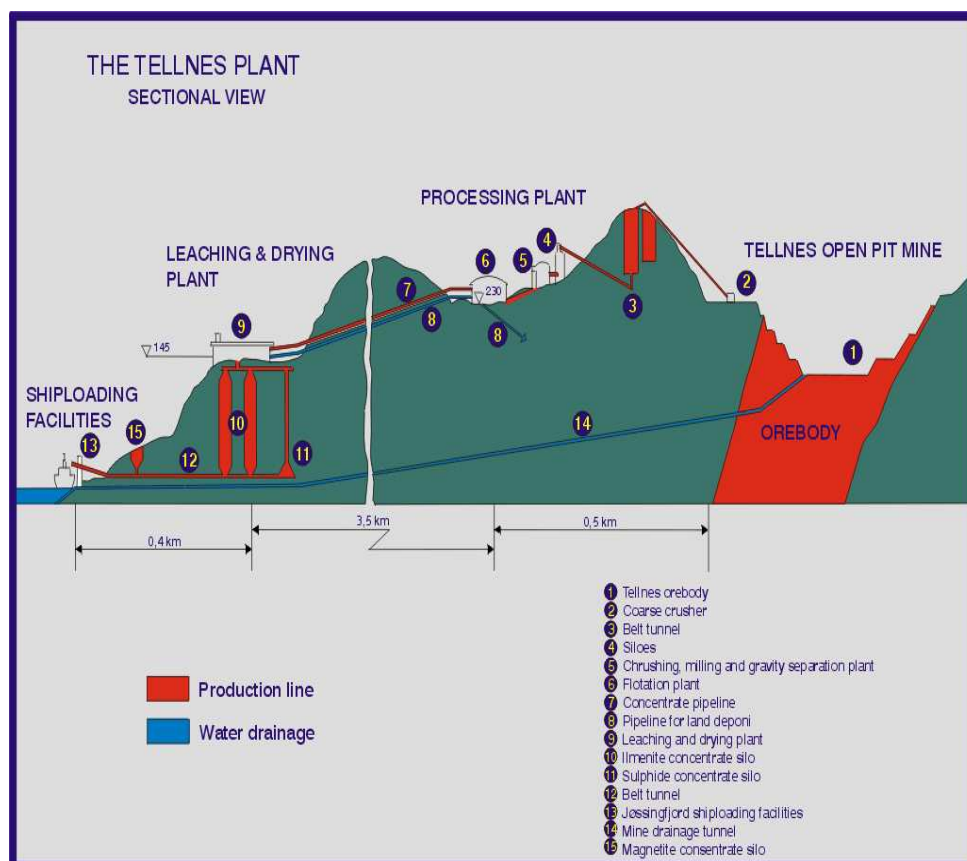
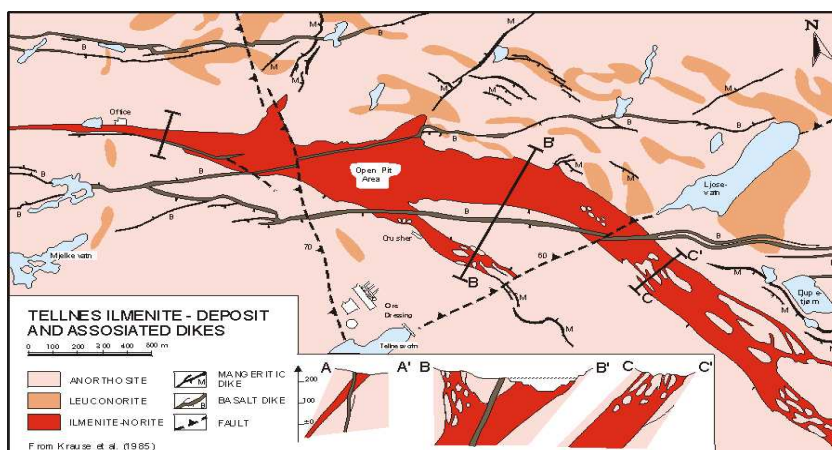


Figure 1 Titania AS production

Nowadays the Production can roughly be divided into 3 stages (see figure 2).

The first stage is the open pit mine from which the ore is transported to the primary crusher by trucks. From the primary crusher the ore is transported by a conveyor belt through a tunnel towards the processing plant (second stage). Here the ore goes through the fine crushing and grinding plant before reaching the separation plant where minerals are separated from each other using their specific characteristics such as specific gravity and flotation characteristics.

The concentrate is then transported through a 4 km pipeline to the drying plant (third stage) in Jøssingfjord. Here further treatment of the concentrate takes place. This includes: leaching of apatite, flotation of sulphide minerals and drying. The final product is loaded on ships for transport to clients.

5. Research method

Literature reviews provide general information about the reaction principles (see literature list), reagent characteristics and possible results. The testing of our ore can be subdivided into three stages. The principle for the use of three testing stages is to be more effective. By starting on a small scale many tests can be conducted within a relatively short time.

The results from the tests in one stage will be used to determine the testing condition for the following test stage.

The laboratory floatation cell is the smallest of all three. Although the full scale floatation condition will differ from the laboratory floatation condition, valuable information can be gained during these tests. The main purpose of the laboratory test is to narrow the range of values found in the literature. The new, more precise values will be the base for the pilot tests.

The pilot cell is the second testing stage. It has fewer restrictions than the laboratory cell and greatly resembles a full scale floatation cell. The greatest advantage compared to the laboratory cell is the ability to produce large amounts of concentrate and to perform a continuous process. It can be connected to the processing plant, allowing it to test many different streams directly. Again the pilot cell testing result narrow the testing ranges for the full scale tests. The pilot cell can effectively be used in combination with the laboratory cell. For example: the larger concentrate volume from the pilot cell make it possible to first float sulphide bearing minerals and clean them afterwards in the laboratory cell.

Once the last testing stage is reached only several streams will be tested in a precise range of condition. If the results are satisfactory the developing stage will be started.

The collector used is potassium amyl xanthate (PAX is the industrial name). In the report it will be referred to as “xanthate”. Xanthate is most stable and therefore effective in an alkaline environment. In acid solutions xanthate is decomposed. The process water in the pulp to be floated is in the pH range of 9 to 9.9.

Tests will be conducted to find out if xanthate is sufficiently effective at a PH value between 9 and 9.9 and if it shows different effectiveness if the PH changes. The PH will be adjusted using 10 % sulphuric acid (H_2SO_4)

Time dependence of the reaction will also be tested. Different retention times affect the recovery and concentrate quality. Reagents need some time to react with the different minerals. On the contrary, at longer time intervals the reagents could start to react with unwanted tailing minerals (all other non sulphide minerals including ilmenite) too as some reactions are time dependent. Intergrowth particles consisting of several minerals including tailing minerals would also be floated. This would lower the concentrate quality.

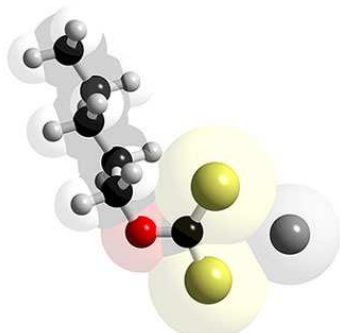


Figure 3, potassium amyl xanthate

The required amount of potassium amyl xanthate per amount of solid (g/t) should also be investigated as this is the most expensive chemical used in the process. Different concentrations amounts of xanthate may also affect the tailing minerals

In each test (time, PH, quantity of xanthate) all condition but one will remain constant. (see appendix for constant conditions). From each test the feed, concentrate and tailing will be analyzed by an XRF analyzer. From the XRF analysis results the specific recovery and enrichment of each element can be calculated. The concentrate quality will be pictured using the percentages of each element of interest present in the concentrate. The XRF results will provide information about elements and not minerals. This however, is a reflection of the mineral contents in the feed, concentrate and tailing.

Formulas:

a = element content of feed in %

b = element content of tailing in %

c = element content of concentrate in %

Enrichment $i \rightarrow i = \frac{c}{a}$

Recovery = $\frac{c(a-b)}{a(c-b)} \times 100\%$

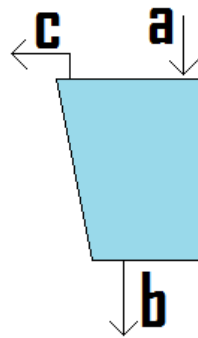


Figure 4, different streams

As can be seen the enrichment “i” is a ratio between two concentration and therefore dimensionless.

Although a total XRF element analysis will be obtained only several elements will be taken into account. First TiO_2 which is present in the ilmenite. In an ideal case it should be only be present in the tailing and not in the floated concentrate. The sulphide bearing minerals should be floated and therefore only be present in the concentrate and not in the tailing. To represent the sulphide bearing minerals the element S will be taken into account as well as Ni, Cu and Co. the last three elements are only present in sulphide bearing minerals in the Tellnes deposit (see appendix, Minerals in Tellnes ore). S, Ni, Cu and Co should therefore show a high correlation enrichment. Slight differences may occur due to the fact that potassium amyl xanthate is a better collector for the minerals: pyrite, pyrrhotite, pentlandite, violarite, millerit and chalcopyrite than for siegenite. The element Co is only present in siegenite and should therefore have lower recovery and enrichment values.

The results will be presented in the form of graphs. The graphs will indicate if any optimal condition is present within the tested range. In the process of analyzing the results we have to take into account that a small error will have a relatively higher effect at the laboratory test due to the smaller scale. It will therefore not be surprising to see deviations in the trend curves. For a list a possible inaccuracies/errors see appendix.

6. Result analysis

Time

Recovery

The first characteristic noticed in the Recovery % curves (graphs 1 and 10) is the good correlation between S, Ni and Cu element. The Co recovery curve does not show any correlation to the S, Ni and Cu curves. Furthermore the Co curve of the FCS test is exactly the opposite of the Curve in the FRS test. This indicates that potassium-amyI-xanthate is not optimized as a reagent for the floatation of siegenite (a Co mineral). As mentioned in the “research method” potassium amyI xanthate was a very good reagent for the floatation of pyrite, pyrrhotite, pentlandite, violarite, millerit and chalcopyrite which contain the elements S, Cu and Ni. These three elements have the highest recovery. The element Ni which is present in the minerals pentlandite, violarite, siegenite and millerite closely follows the trend of the S and Cu elements.

The gradient of the recovery % curves represents the recovery rate. The highest recovery rate for FCS is between the 2 and 4 minute measurement (7.2 % difference in recovery). This indicates that there is still sufficient sulphide bearing minerals to be floated and the sulphide bearing particles did not all have the time to react (come into contact) with the collector and the air bubbles. The curves for FRS do not show such great differences in recovery (only 0.9 % difference in recovery between the 2 and 4 minutes intervals). This is caused by fact that the FRS slurry has a lower sulphide concentration than the FCS slurry. In contrast to FCS most sulphide bearing particles will have been floated effectively within 2 minutes. As a result the FRS recovery curves are relatively horizontal and do not show great variations.

An imprecision is present in both the FCS and FRS curves. In both the FCS and FRS the curves representing S bearing minerals recovery show a dipping gradient at certain intervals.

Taking the FCS “S” curve to illustrate this error: between 2 and 4 minutes S bearing minerals are being floated resulting in a rising recovery. The recovery values at 6 and 8 minutes are lower than those at 4 min which means S bearing particles have sunken from the formed froth layer. The 10 minute interval shows the highest S recovery. This indicates that S bearing particles have been floated again. The float→sink→float sequence is improbable. Furthermore the values between the 4 minute and 10 minute measurement all vary within a range of 3,25 % which is relatively limited range.

According to the probability Formula (Ullmann’s Encyclopedia of Industrial Chemistry) it



Figure 5, gold colored spots on surface

would be more likely to have a rising curve which would reach a maximum without dipping at any moment (Will’s Mineral Processing Technology, B.A. Wills and T.J.Napier-munn, page 295). Longer time interval would mean higher chance for particles to react with reagent and collide with air bubbles (higher P_c and P_a) leading to higher chance of successful floatation and ultimately higher recovery. In both FCS and FRS we could say maximum recovery has been reached. The maximum recovery can be recognized during the

test as the moment at which the gold colored spots cease to appear on the surface (see figure 5). These gold colored spots are sulphide bearing particles forming clusters. For FCS the maximum sulphide recovery reached is 95,16% and for FRS 93,05 %.

Enrichment

At the 2 and 4 minutes measurements only a thin froth layer (C) is present (see figure 6). Due to the continuous agitation of the suspension (in layer A and B) tailing particles reach the bottom of the froth layer.

A = agitated suspension with air bubbles.

B = intermediated state between fully agitated suspension and froth layer. This has a high air content but still has the characteristics of a liquid.

C = froth layer

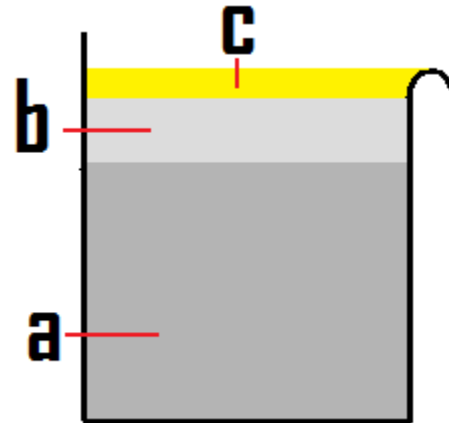


Figure 6, flotation cell stratigraphy

Small amounts of fluid containing tailing particles are skimmed out of the cell together with the thin froth layer. These tailing particles lower the enrichment. After 4 minutes a thicker froth layer is formed. Now only the top of the froth layer is skimmed out of the cell and reduced amounts of tailing particles reach the concentrate. this results in higher enrichment values (peak at 6 min).

In both FCS and FRS a clear peak is present at the 6 minute measurement (FCS $i=129.9$, FRS $i=114.8$ for the S curves) (graphs 2 and 11).

After 6 minutes the enrichment drops which means the ratio between floated sulphide bearing minerals and floated tailing minerals drops. Three possibilities can be mentioned.

- 1) According to the recovery curves a maximum recovery has already been reached at 6 minutes (horizontal trend of the curves). After this, nearly no sulphide bearing particles are floated. A constant amount of invaluable particles are still reaching the concentrate as the surface is skimmed constantly. Result: relatively less sulphide bearing particles reach the concentrate compared to the constant amount of invaluable minerals → enrichment drops.
- 2) In the beginning of the floatation the easiest particles float (well liberated particles containing sulphide minerals). At longer intervals intergrowth particles composed of different minerals start to float. Result: enrichment drops.
- 3) Some minerals will react with reagents only after a certain time period. In our case it means that after 6 minutes some tailing minerals start to react with the xanthate and start to float, adding to the constant flow of tailing minerals reaching the concentrate. As a result the enrichment of sulphide bearing minerals will drop.

Concentrate quality

The concentrate quality (graphs 3 and 12) closely follows the trends of the enrichment curves. A clear inverse correlation between the TiO_2 and S curve is visible. Also here a peak in valuable element concentration in the concentrate is present at the 6 minute measurement. Although Cu has a higher enrichment than Ni the concentrate has a higher Ni percentage than Cu. This is caused by the original feed composition in which the Ni concentration is higher than the Cu concentration. In the FCS feed, Ni has a concentration of 0,07 % in the feed and Cu 0,02 %. A higher Cu enrichment is not enough to overcome the original concentration difference in the feed both in the FCS and FRS.

pH

Recovery

The recovery of both FCS and FRS is not significantly affected by the different PH conditions (graphs 4 and 13). FCS sulphide recovery values vary between 91.8 % and 97.1 While FRS values vary between 90.7 % and 93.3 %.

The S, Ni and Cu curves show the same stable trend while Co shows more variance, with value ranges 42.1-55.5% (FCS) and 38.0-50.8 (FRS). Again the elements S and Cu have the highest recovery closely followed by the element Ni.

These results do not show any loss in effectiveness of the collector (potassium amyl xanthate) in the acidity range. The reason for this may be:

- 1) Xanthate decomposition reaction. The decompositions of Potassium Amyl Xanthate could be a slow process and the time period involved in the laboratory tests too short. It is important to investigate this during continuous floatation with the pilot cell tests, as the collector may stay longer in the floatation cell.
- 2) Quantity of xanthate. According to the FCS xanthate tests 50 g/t (33 g/t for FRS) is enough to reach the maximum recovery within 6 minutes at PH 7. Although the xanthate may gradually be decomposed enough intact xanthate would remain to react with the sulphide minerals.

Enrichment

In the enrichment curves a certain general trend is visible in both FCS and FRS (graphs 5 and 14). The S curves have a peak (FCS $i=122,6$, FRS $i=128,26$ for the element S) at pH 7. On both sides of the peak in the acid and alkali range a minimum is present. For FCS the minimums are at PH 4 ($i=78,3$) and 9 ($i=103,09$), for FRS the minimums are at PH 4 ($i=103,96$) and 8 ($i=105,88$) for the element S. FRS also has a minimum at PH 6. This one however is not as significant as the ones at PH 4 and 8. It is also not present in the curves representing Ni and Cu enrichment. It is known from literature that xanthate is more effective under alkaline conditions. However the recovery values stated the xanthate was still effective in acid conditions down to PH 3 (read recovery part).

Another way in which xanthate may lose its effectiveness is by losing its selectiveness. Minerals that do not react with xanthate at a certain pH may be floated at another PH. (see: Principles of Mineral Dressing, A.M. Gaudin, critical PH). Because the recovery curves representing S, Ni,Cu and to a certain extend Co are rather constant the recovery effectiveness of xanthate is not depending on the pH (under the standard testing conditions). The difference in enrichment is therefore likely to have been caused by a loss in

selectiveness at pH 4 and 8-9. Under these pH conditions apparently other minerals present in the slurry float. This as a result lowers the enrichment of sulphide, nickel, copper and cobalt.

Concentrate quality

A clear inverse correlation is visible between TiO_2 and S (graphs 6 and 15). In the FRS results at pH 7 the TiO_2 curve does not show any link to the dip in the S curve. This could support the statement that the dip at pH 7 in the S curve is an imperfection in the measurement/error. Both concentrates have the best quality at pH 7: FCS 36,9 % (S) and 5,4 % (TiO_2), FRS 30,0 % (S) and 5,3 % (TiO_2).

Xanthate

Recovery

Both the FCS and FRS recovery curves reach a maximum (graphs 7 and 16). In the FCS curves the elements related to sulphide bearing minerals have a peak recovery at 58 g/t. The greatest recovery differences are in the lower range from 17 g/t up to 50 g/t. The Time recovery curves showed a maximum recovery had already been achieved at a 4 minute interval (with 50 g/t). During the xanthate test the time was set at 6 minutes in all measurements (see appendix: standard conditions). Any difference in recovery must therefore be related to the added quantity of xanthate. Up to 50 g/t the added xanthate quantity has a limiting effect on the recovery.

In the FRS results this limiting effect is less visible although it can be recognized in the S curve in the range from 17 g/t to 33 g/t and to a less extent up to 50 g/t. Because the concentration of sulphide bearing minerals in the FRS stream is lower less xanthate is needed to achieve a maximum recovery.

The relation between sulphide bearing mineral recovery and g/t can be explained with the probability formula (Ullmann's Encyclopedia of Industrial Chemistry). The easiest particles will be floated first. These are the particles composed of only one sulphide bearing mineral or where the sulphide bearing mineral is dominant. Higher g/t values enhance the chance of particles to come into contact with the reagent (higher chance of particle reagent bonding = higher P_a). This will cause particles in which sulphide bearing minerals are not dominant to be floated successfully as well. A peak recovery is achieved when no particle remains that can effectively be floated even with higher values of g/t. The concentration of valuable minerals in the remaining particles is so low that not enough air bubbles can bind to overcome the gravitational force.

Enrichment

Great variations are visible in enrichment of both the FCS and FRS streams (graphs 8 and 17). In the FCS sulphide curve the values vary between $i=81.0 - 152.4$. In the FRS sulphide curve the values vary between $i=89.2 - 135.7$. Not only is the enrichment range extensive, but the FCS and FRS curve show different patterns of maximums and minimums.

The highest recoveries are achieved in the lower values of g/t. As the g/t rises the general trend of the curves is dropping. This is in line with the higher "recovery=lower enrichment principle" (B.A. Wills and T.J. Napier-munn, Will's Mineral Processing Technology, page 295). A low recovery indicates only the easiest particles have yet been floated. These are the particles that consist of nearly pure sulphide bearing minerals. A higher recovery means more particles with partially tailing minerals are being floated. As a result the enrichment of the concentrate drops. This is only true if the reagent used for the floatation is known to be an effective collector for the minerals present in the slurry.

FCS and FRS will be discussed separately.

FCS

Between 17 g/t and 33 g/t the recovery curve has the highest positive gradient (graph 8). In these ranges the easier particles are floated and only a minor constant amount of tailing

material reaches the concentrate. The recovery rises and as a result the enrichment does so too. At 33 g/t peak enrichment is reached. After this peak the recovery keeps rising however more difficult particles start to be floated resulting in more invaluable material reaching the concentrate. An inverse correlation between recovery % and enrichment (i) is reached. (see description above.)

FRS

It was stated before that the FRS recovery curve at 33 g/t (graph 17) are much closer too their maximum recovery than the FCS recovery curves under the same conditions. (due to lower concentration of sulphide bearing minerals in the feed). The FRS enrichment curve therefore immediately follows a inverse correlation with the recovery % curve. The 67 g/t measurement remains inexplicable. Further tests could be done to find out if we are dealing with an exception/error.

Concentrate quality

Again concentrate quality and enrichment curves are highly similar in trend (graphs 9 and 18). Both FCS and FRS have the best quality in the lower g/to values. 33 g/t for FCS and 17 g/t for FRS.

7. Pilot cell tests

For the first tests with the pilot cell the most ideal stream was used, FCS. The first test was a batch test with the same standard conditions used in the laboratory tests. The propeller speed and air supply was adjusted to the point at which all particles were kept in suspension avoiding excessive turbulence that would destroy the froth layer. No preconditioning was used.

Batch

The concentrate from the first 4 minutes was taken for further analysis. The amount of the concentrate was such that a cleaning stage with the laboratory cell could be conducted. A problem with an evacuation valve made it impossible to recover the tailings from the pilot cell. Although it is not possible to calculate the recovery, the obtained values for concentrate quality and enrichment provide valuable information on the floatation process. All enrichments have been calculated with the individual concentrates and the original feed to the pilot cell (not laboratory cell)

Table A

	TiO₂	S	Ni	Cu	Co
Byproduct average	3.82	30.21	4.01	1.615	0.83
laboratory average	5.7	35.86	4.79	1.48	0.75
Pilot	8.26	34.16	4.24	1.35	0.96
Cleaning	1.04	48.26	5.84	2.11	1.3

Table A shows the obtained quality of the pilot cell concentrate comes close to the average FCS laboratory quality and the current sulphide by-product quality. The drawback is the TiO₂ content of 8,26 %. This is more than double the current by-product TiO₂ content.

The cleaning process with the laboratory cell has proven to be successful. No extra collector was added for the cleaning stage. The concentrate from the first 4 minutes was sampled. The differences in element content between current sulphide by-products, pilot and first concentrate from cleaning are:

	TiO₂	S	Ni	Cu	Co
Pilot- Byproduct average	4.44	3.95	0.22	-0.27	0.14
Cleaning - Byproduct average	-2.78	18.05	1.83	0.49	0.48

Not only is the S concentration 18.1 % higher in the cleaned 1 concentrate, but the TiO₂ content has dropped to 1.0 % which is 2.8 % lower than the current byproduct value. The lower TiO₂ content is of great importance because TiO₂ remains the main product of Titania AS. 8.3 % of TiO₂ in the sulphide product (pilot concentrate) means a great loss of valuable material.

Continuous

During the first continuous test the reagent dosage was not precise. By diluting the reagents we were able to minimize the error. The second test was successful. The quantities of concentrate were high enough to include a cleaning stage with the pilot cell.

concentrate quality	TiO ₂	S	Ni	Cu	Co
Byproduct average	3.82	30.21	4.01	1.62	0.83
Laboratory average	5.7	35.86	4.79	1.48	0.75
Pilot	16.88	23.24	3.31	1.028	0.70
Cleaning	0.87	48.77	7.71	3.66	1.26

The results from the continuous pilot cell test do not differ significantly from the batch test. Sulfide bearing minerals were floated successfully, (S = 48.7 % in the cleaning concentrate). Again the TiO₂ concentration in the pilot cell concentration is too high. In the continuous test it is twice as high as in the batch test. The cleaning stage is a good solution to avoid loosing too much TiO₂ (from 16.88 % to 0.87 % TiO₂ due to cleaning stage).

(figure 7 and 8 clearly show the difference in the first flotation stage and the cleaning stages. The cleaning stage has a much thicker and compact froth layer than in the first concentrate)



Figure 7, first concentrate



Figure 8, cleaning stage

8. Conclusion

General observations conclude that potassium amyl xanthate is not optimized as a collector for siegenite. Chalcopyrite is the most effectively floated mineral. Although the Cu enrichment is higher than the one for Ni, it is not high enough to overcome the concentration differences in the feed. TiO_2 and S show an inverse correlation in concentration as expected. Furthermore the principle of high recovery low quality is especially visible in the xanthate tests. Although some optimum values can be defined the reaction does not seem to be very dependent on either the reagent amount, pH value or retention time. No extreme variations are present in the results.

Both the FCS and FRS concentrates were relatively close to the present by-product. Only the TiO_2 content remains high. Even if the FRS average concentrate has a lower S content it still has a decent quality.

Cobalt should not be taken into account too seriously in further result analysis as this element shows no clear pattern.

The FCS reached a maximum recovery at 4 minutes the enrichment and concentrate quality show a peak at the 6 minute measurement. The 6 minute interval can therefore be regarded as the optimal time interval. Pilot cell tests should be conducted within the range of 4 to 7 minutes.

The xanthate quantity tests showed a maximum recovery had been reached at 50 g/t The highest enrichment and concentrate quality were at 33 g/t. More laboratory test should be done to investigate if the result truly have so much variations or if errors are responsible for the maximums and minimums. Now the best range to test with the pilot cell would be from 25 to 58 g/t This range includes all three peak values for recovery, enrichment and concentrate quality.

The PH did not affect the recovery effectiveness of xanthate. It may however have affected the selectiveness which could be an explanation for the minimum values in the enrichment and concentrate quality curves. A general trend shows the xanthate is more effective from PH 6 and higher. The recommended testing range for the pilot cell tests would be from PH 6 to 9. The FCS stream should also be tested under original PH.

Except from the Co curve for recovery the FRS time tests results showed great similarity with the FCS results. The recovery curve for sulphide is relatively horizontal indicating a maximum recovery may already have been reached at a 2 minute interval. Highest enrichment and concentrate quality were obtained at the 6 minute measurement. The recommended time range would be from 4 to 7 minutes, similar to FCS.

A higher xanthate quantity enables more difficult particles (intergrowth particles) to be floated as well. The enrichment is the highest in the lower range of added xanthate quantities. The enrichment and recovery are inversely correlated. Only the measurement at 67 g/t is inexplicable. Further test could reveal if this is an error or not. The general concentrate quality trend shows the best concentrate quality is reached with xanthate amounts of 17 to 33 g/t. The recommended range for the pilot cell tests would be from 17 g to 50 g per ton. An optimum between recovery and quality is more likely to be found in this range than with 67 g/t, making further investigation to a possible error at 67 g/t of limited importance.

The PH does not really affect the recovery effectiveness. A loss of selectiveness may be the cause of the minimum values in the enrichment and concentrate quality curves (similar to FCS). At PH 6 an error may be present. A small minimum is present in the S curve but no maximum is visible in the TiO_2 curve which is inversely correlated to the TiO_2 concentrate quality curve. The more promising range to test would be from PH 5 to 8. The slurry should also be tested with the original PH.

The Pilot cell installation has been completed and the first tests have brought promising results. The machine is working. The batch and continuous tests do not differ significantly from each other. Both are of higher quality than the current by product. The cleaning stage is an effective process to reduce the high TiO_2 content in the pilot cell concentrate. The tailings from the cleaning process should be recirculated to avoid losing the remaining sulfide minerals and TiO_2 . The process now has to be optimized taking into account the recommendations based on the laboratory test results.

9. Literature list

Arthur F. Taggart, Handbook of Mineral Dressing, ores and industrial minerals.
Fourth printing, September 1950.

A.M. Gaudin, Principles of Mineral Dressing.
First edition
Page 367-399

Baki Yarar, Ullmann's Encyclopedia of Industrial Chemistry flotation report.
Sixth edition.

B.A. Wills and T.J.Napier-munn, Will's Mineral Processing Technology
Seventh edition, 2006.
Page 267-296

10. Appendices

Testing equipment and standard conditions for laboratory cell tests

Floatation Cells

	Laboratory cell	Pilot cell	Full scale cell
Cell volume (l)	3	2*28	1000
Conditioning tank	No	Yes	Yes
Continuous process possible	No	Yes	Yes
Propeller RPM	1500	1000-1500	230
Air inflow regulation	No	Yes	Yes
High concentrate volume	No	Yes	Yes
Possible cleaning of concentrate	Yes (only concentrates from other cells)	Yes	Yes
Mobility	High	Moderate	None

Reagents

H₂SO₄ → To adjust PH.

CuSO₄ → Catalysator.

potassium amyl xanthate → Collector.

Pine-oil → Frother/Foamer.

Standard conditions laboratory cell

Batch process.

1\3 cell volume solid material (1.8 kg). As long as no deposition takes place.

2\3 process water.

PH around 7 (variance up to 0.2 is allowed)

20 mg copper sulphate.

50 g/t, potassium amyl xanthate.

4 drops of pine oil.

6 minutes time interval.

Directly after the pine oil has been added the air supply will be turned on and the time will start. The air supply will gradually be raised to maximum avoiding excessive overflow of the tank.

If the water level drops due to skimming of the surface more process water will be added to keep constant water level.

Possible errors/inaccuracies

Too much solid material to keep in suspension. Deposition will take place.

Difficult to keep PH at 7 (neutral) due to logarithmic scale.

Large air bubbles at the surface. May be caused by deposited solid or too much material in suspension. (not enough agitation of slurry).

A constant amount of tailing mineral will always reach the concentrate, this is due to: the intergrowth characteristics of particles and tailing minerals reaching the froth layer. Also while skimming manually fluid from under the froth layer may be skimmed out of the cell and reach the concentrate. This fluid contains tailing minerals.

In pilot cell, inaccurate dosage of reagents.

If the concentrates are left too long in the oven, they will oxidize. This can ruin the concentrate.

Floatation probability formula

(Ullmann's Encyclopaedia of Industrial Chemistry)

The ultimate objective of a flotation process is the selective removal of solid particles from the aqueous medium which is accomplished by the adhesion of air bubbles to the hydrophobic particles/collector. Particle floatability can be treated as a probability.

$$P_f = P_c * P_a * P_s$$

P_f = probability of flotation

P_c = probability of particle – bubble collision

P_a = probability of particle – bubble adhesion

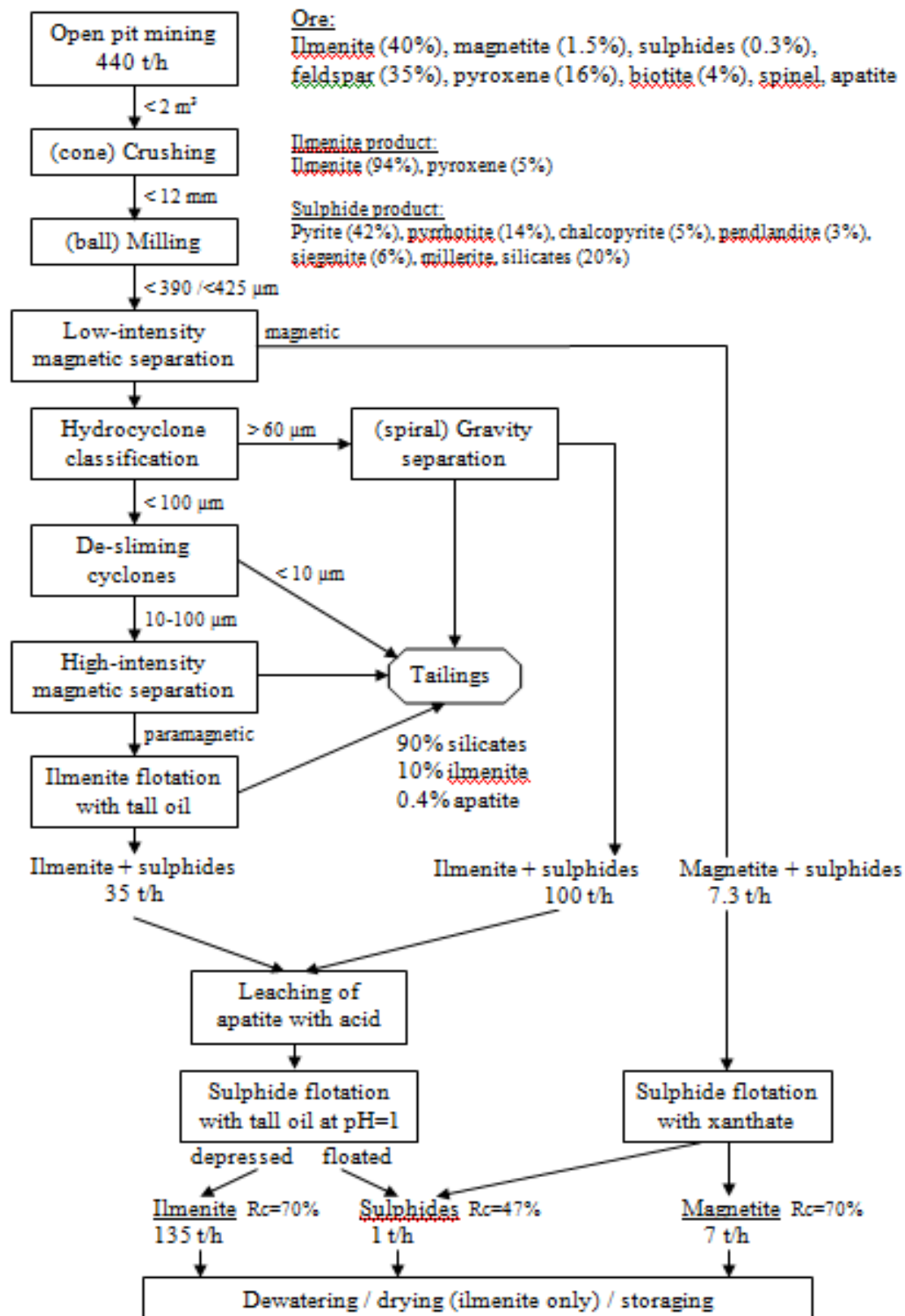
P_s = probability of formation of a stable particle – bubble aggregate

Minerals in the Tellnes ore

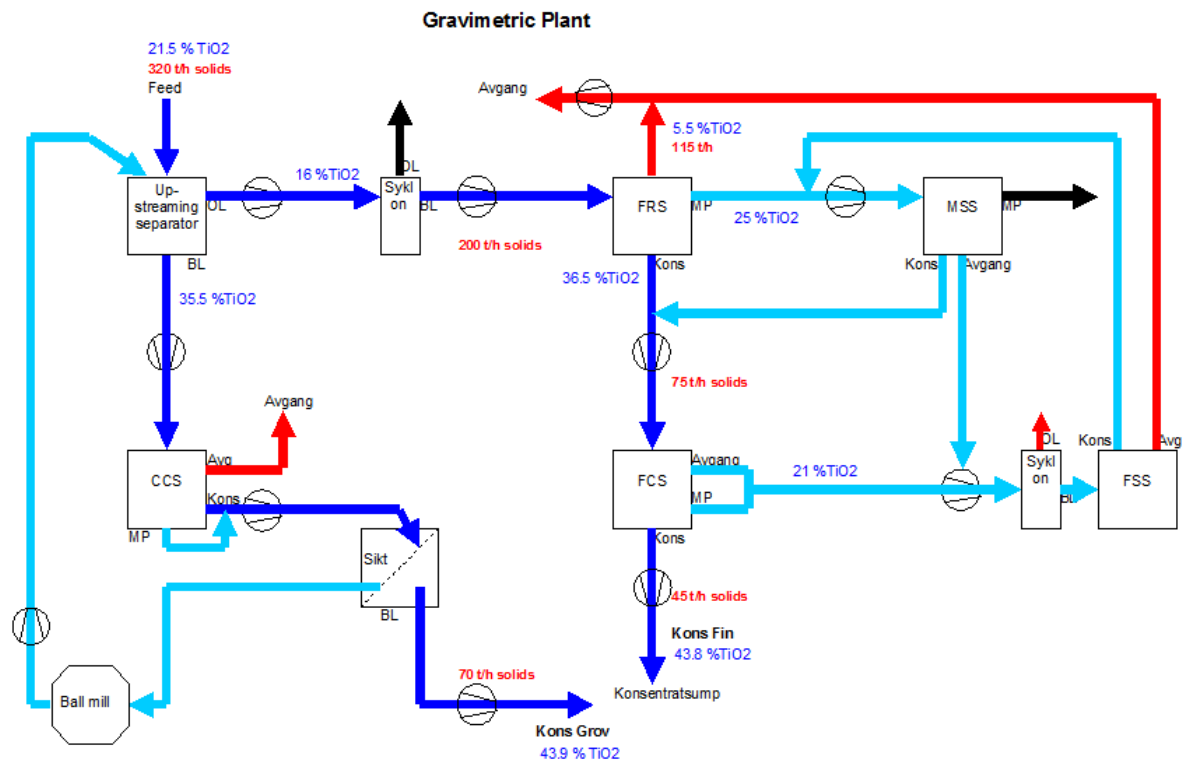
Minerals in the Tellnes-ore			Distribution		
Composition		Density	Ore	Grav. Cons	Acidplant kons
Ilmenite	FeTiO ₃ , (FeO, MgO)TiO ₂ (Hem13)	4.7	39.9	93.3	94.3
Hematite	Fe ₂ O ₃	5,2 - 5,3			
Magnetite	Fe ₃ O ₄ (Ilmenomagnetite)	5.2	1.5	0.4	0.3
Sulphide	Several, ref eget oppsett	4,2 - 5,0	0.3	0.6	0.1
Rutile	TiO ₂	4.2			
Clinopyroxene	(Ca(Mg, Fe ²⁺)+AlFe ₃ +Ti {(Si, Al) ₂ O ₆ })	ca. 3,4	6.5	1.5	0.9
Orthopyroxene	(Mg, Fe)SiO ₃ (En77-75)	ca. 3,5	8.6	2	3.3
Plagioclase	(NaAlSi ₃ O ₈)(CaAl ₂ Si ₂ O ₈) (An45-42)	2.6	36.9	1.3	0.5
Biotite	K(Mg)3{AlSi ₃ O ₁₀ }(OH,F) ₂	3.0	4.2	0.2	0.2
Amfibol *	NaCa ₂ (Mg, Fe, Al) ₅ (Al, Si) ₈ O ₂₂ (OH) ₂	3,3 - 3,5			
Apatite	Ca(PO ₄) ₃ (F,Cl,OH)	3.2	0.6	0.2	0.04
Spinel (several)	(Mg,Fe,Zn)Al ₂ O ₃	3,6 - 4,0	1.5	0.5	0.4
clay minerals					
Chlorite	(Mg,Fe,Al) ₆ (Al,Si ₄)O ₁₀ (OH) ₈	2,6 - 3,4			
Talc	Mg ₃ Si ₄ O ₁₀ (OH) ₂	2.8			
		SUM	100.0	100.0	100.0

Composition of sulphide minerals in the Tellnes-ore								
Sulphides								
		Ni/S	% S	% Fe	% Ni	% Cu	% Co	Ni/S *
Pyrite	FeS ₂	< 0,01	53.45	46.55				
Pyrrhotite	Fe _{1-x} S	< 0,01	37.67	62.33				
Pentlandite	(Fe,Ni) ₉ S ₈	1.03	33.23	32.56	34.21			1.03
Violarite	(Fe,Ni) ₃ S ₄	0.69	42.54	18.52	38.93			0.92
Siegenitte	(Ni,Co) ₃ S ₄	0.38	42.12		43.36		14.51	1.03
Millerite	NiS	1.83	35.33		64.67			1.83
Chalcopyrite	CuFeS ₂	< 0,01	34.94	30.43		34.63		

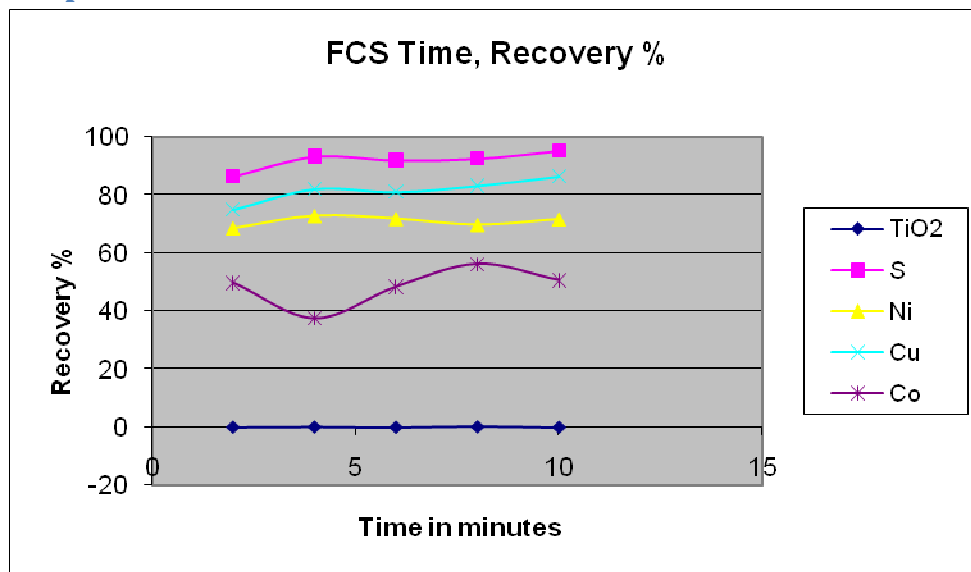
Processing flow chart



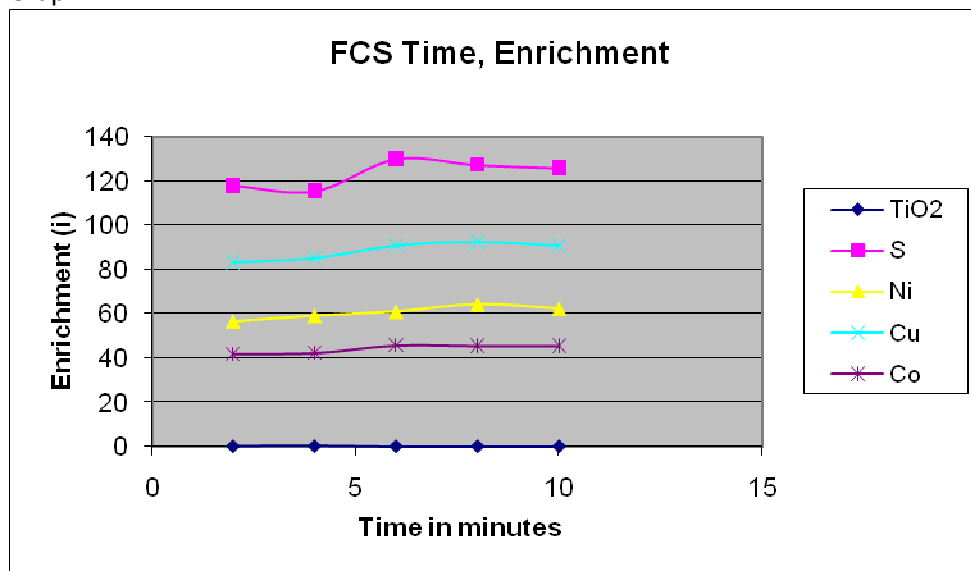
Gravimetric plant



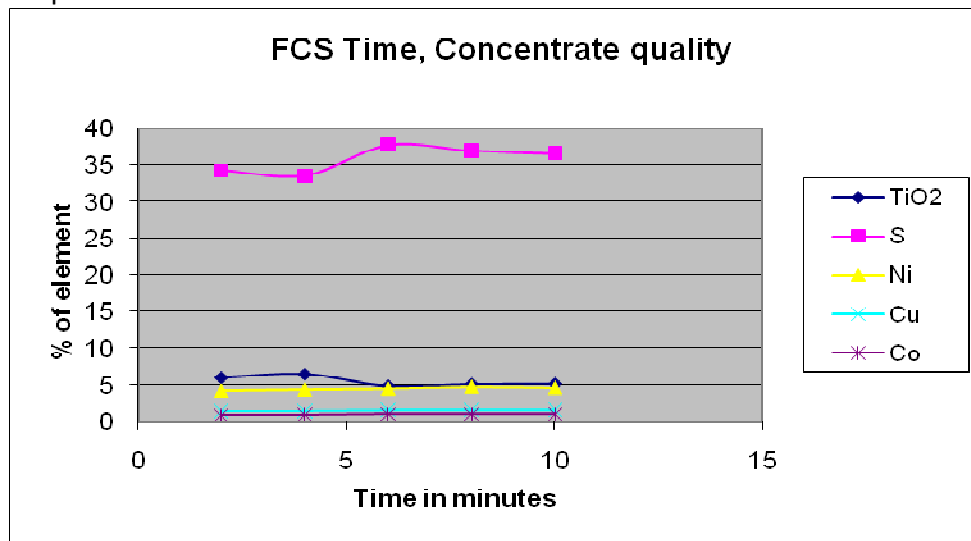
Graphs



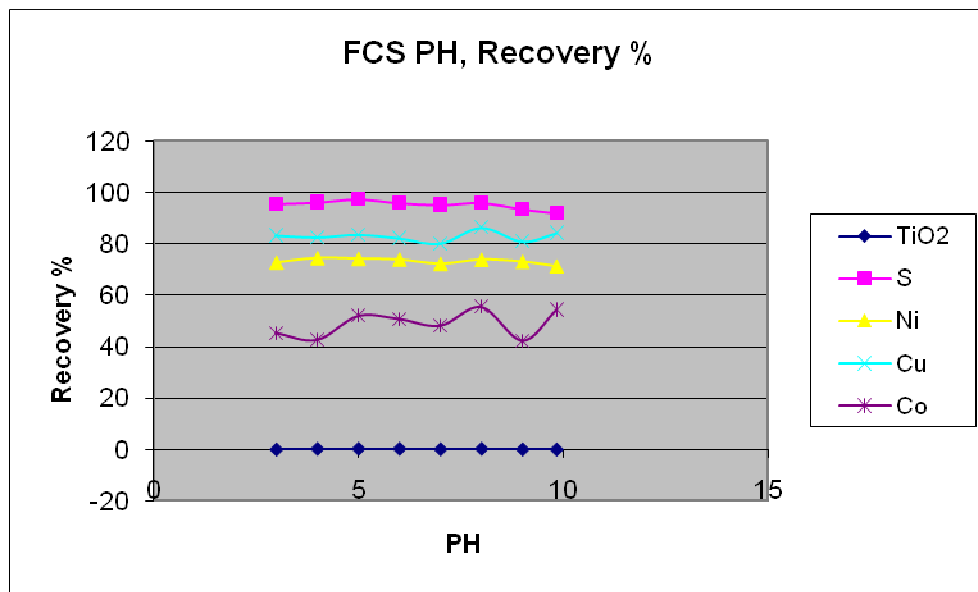
Graph 1



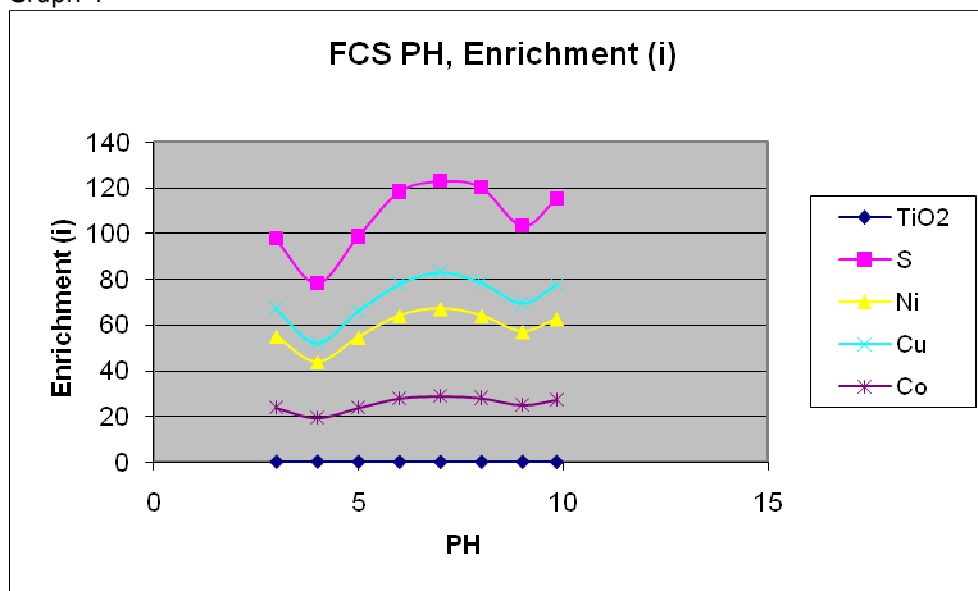
Graph 2



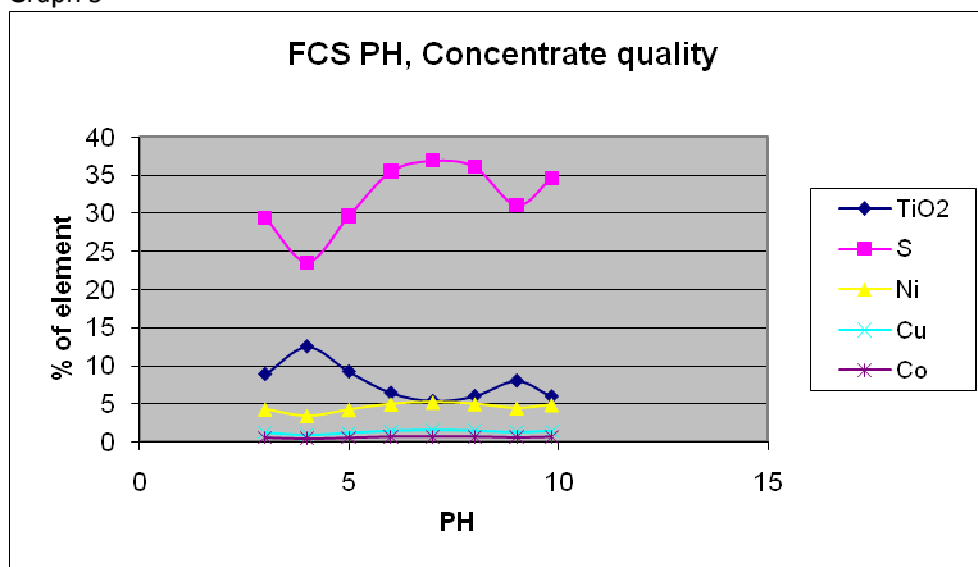
Graph 3



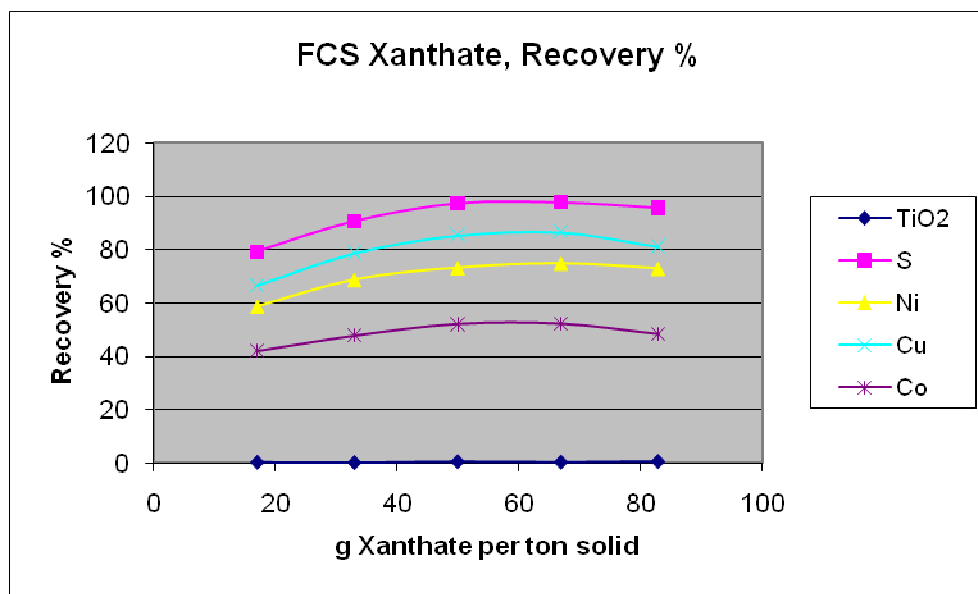
Graph 4



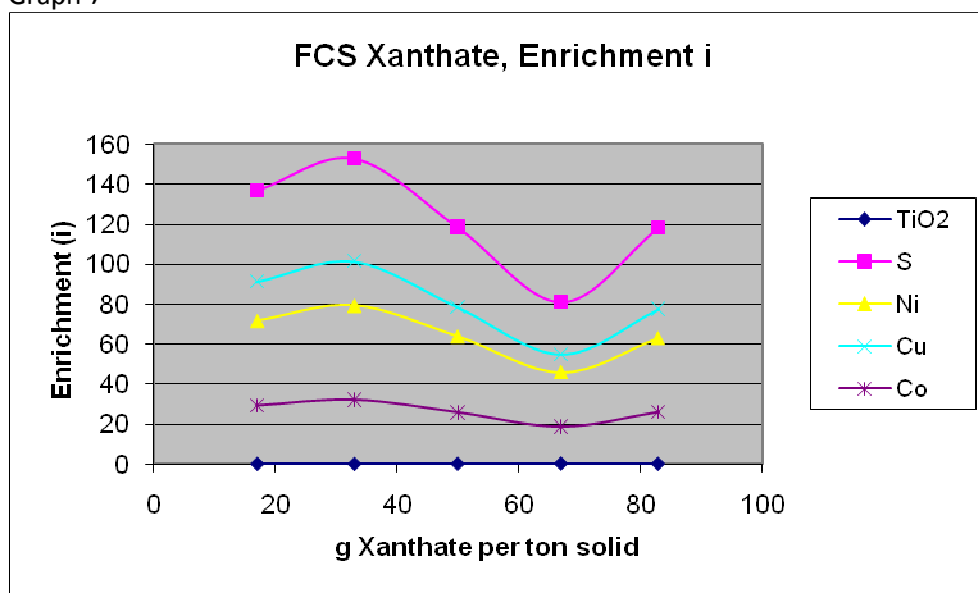
Graph 5



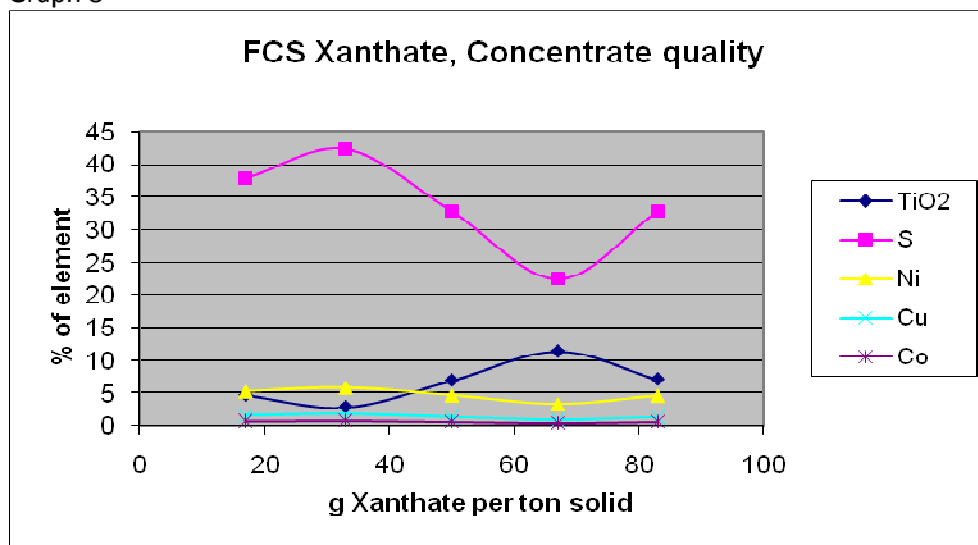
Graph 6



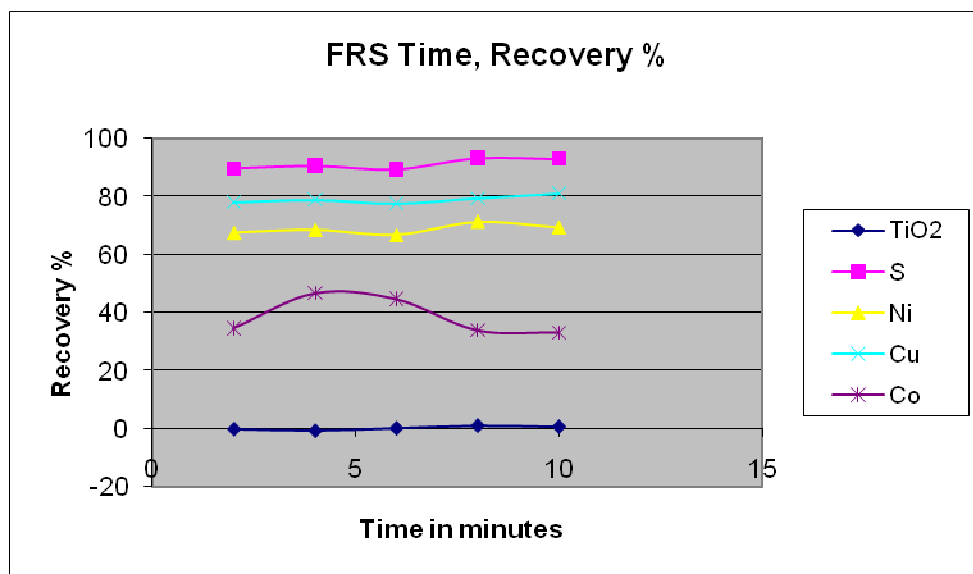
Graph 7



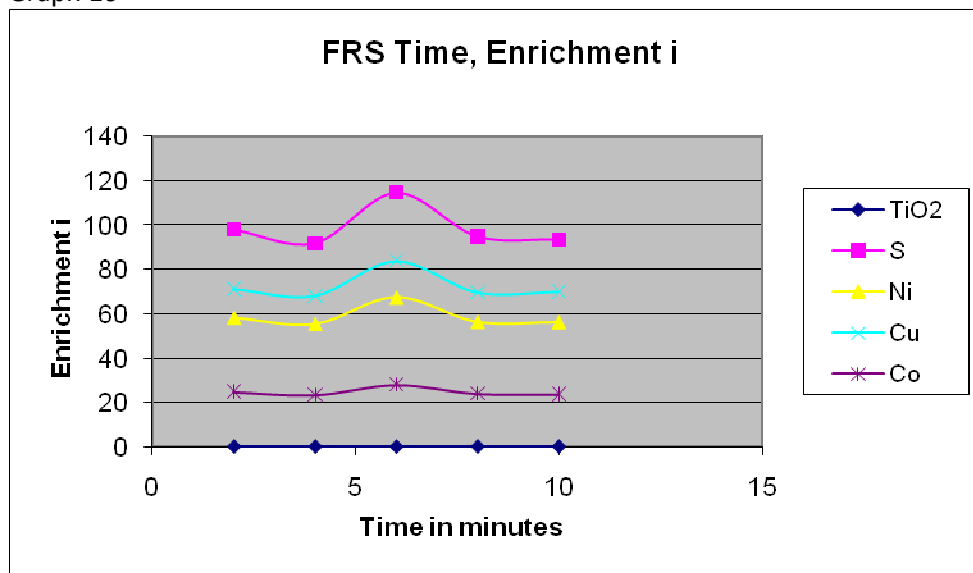
Graph 8



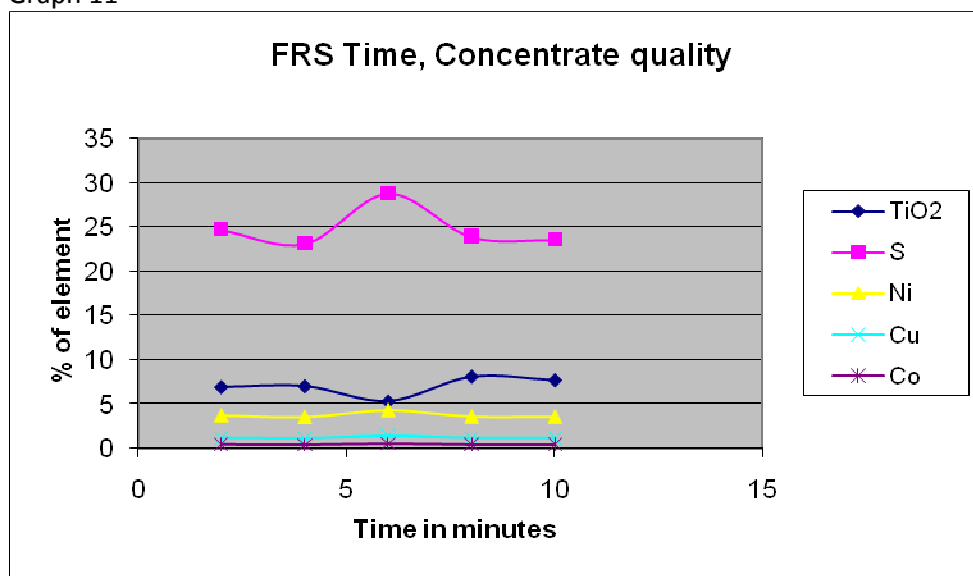
Graph 9



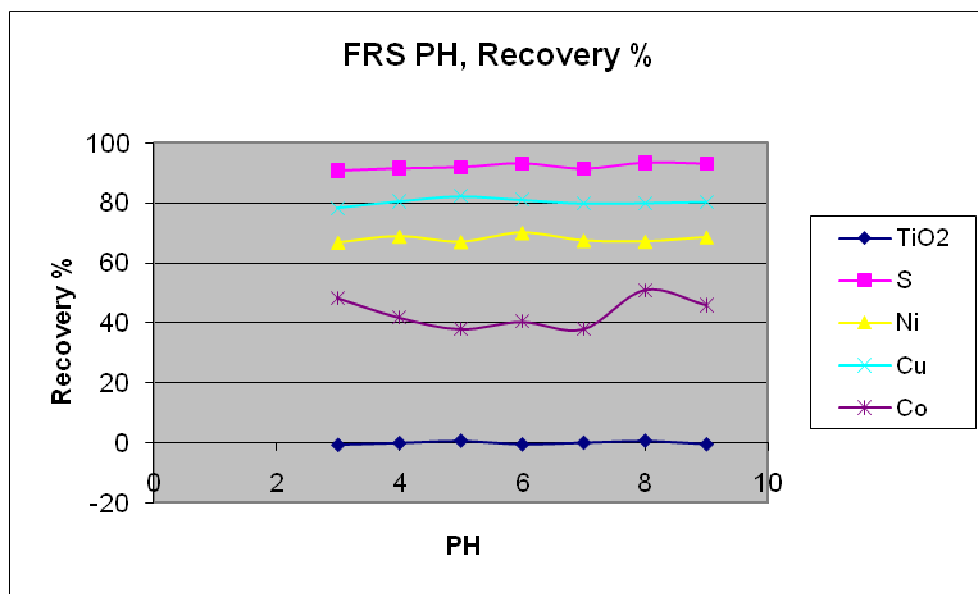
Graph 10



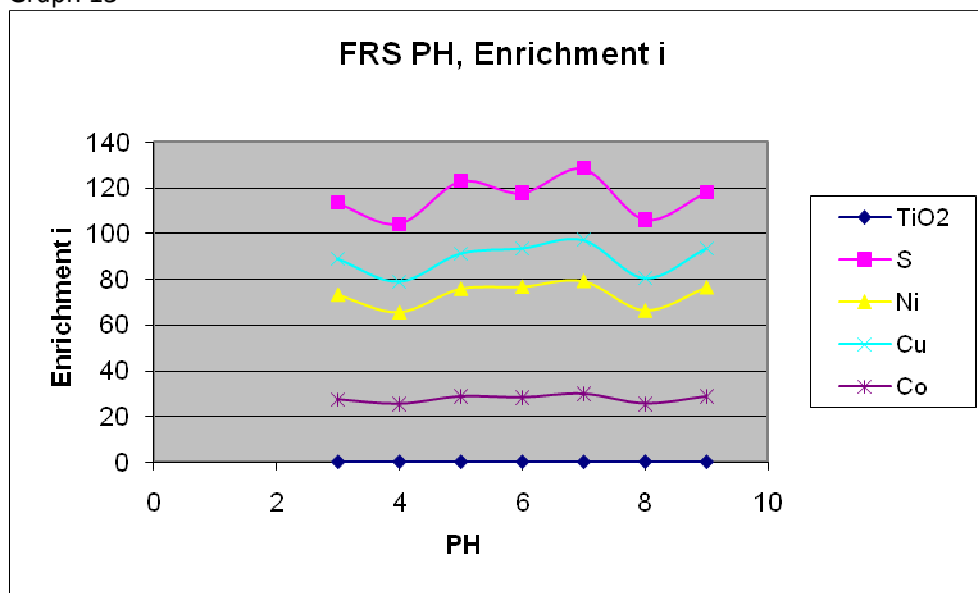
Graph 11



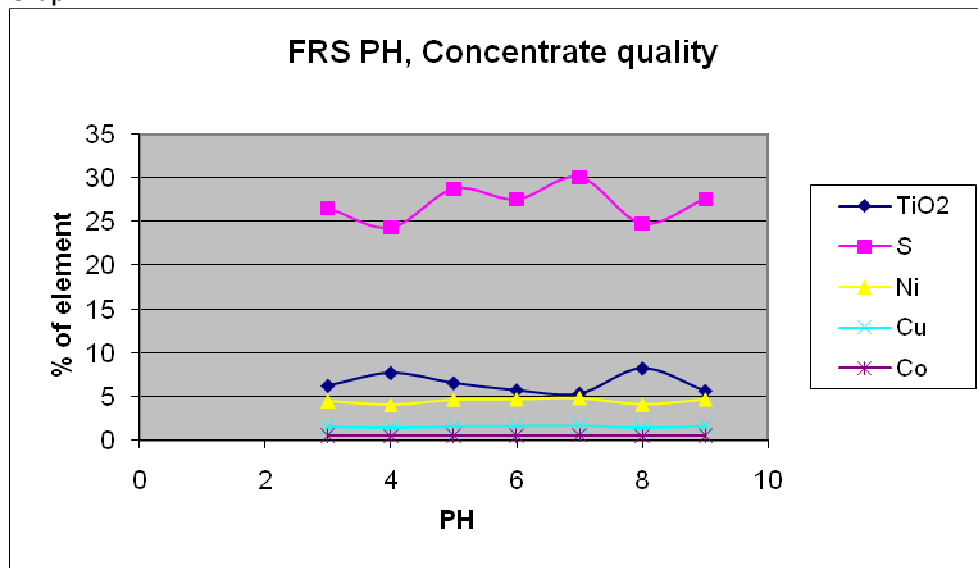
Graph 12



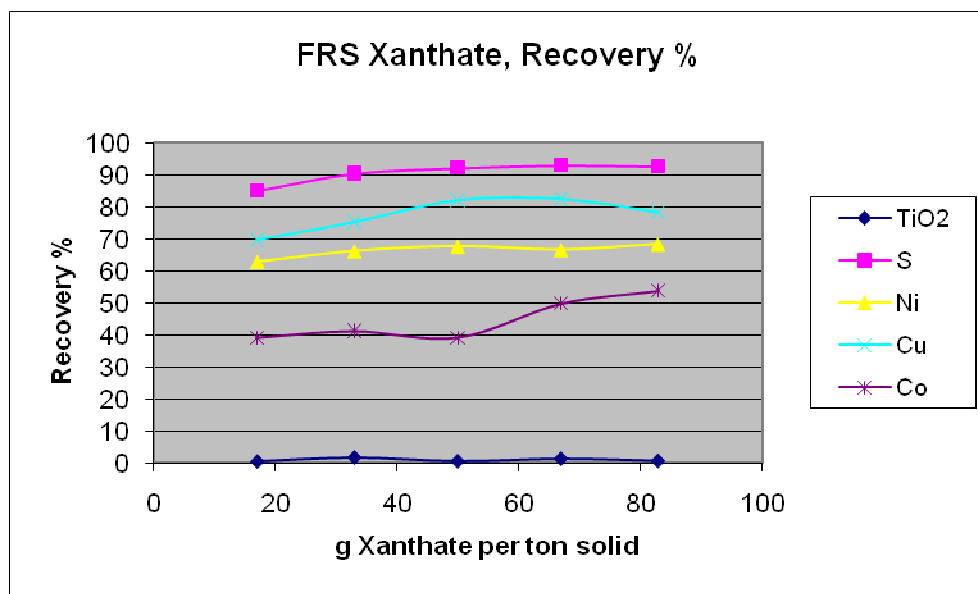
Graph 13



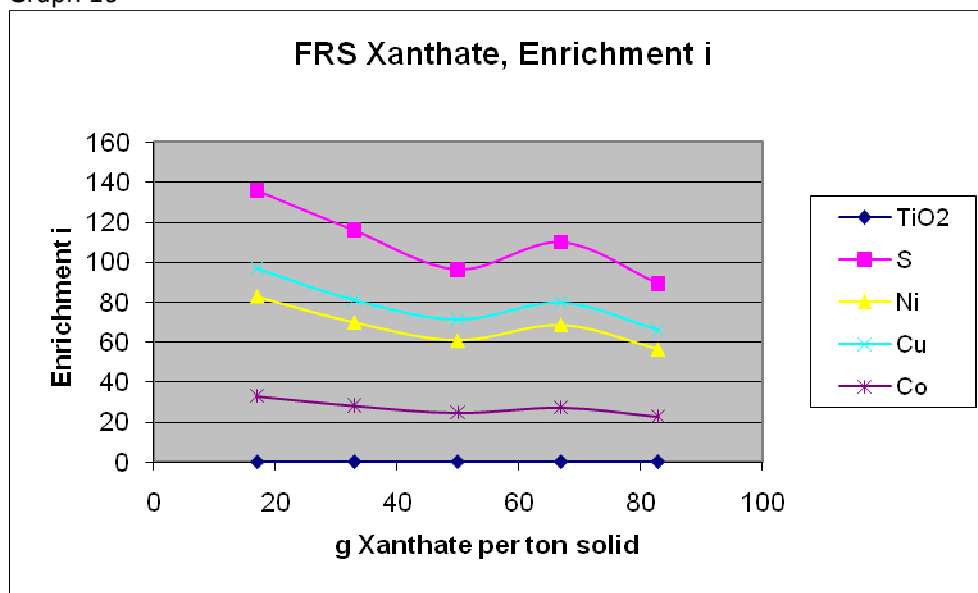
Graph 14



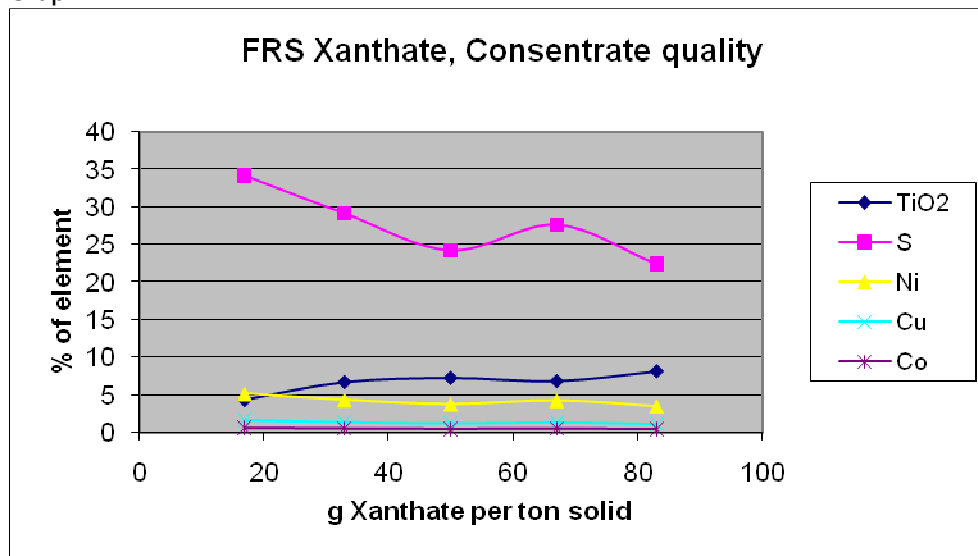
Graph 15



Graph 16



Graph 17



Graph 18