UNIVERSIDADE FEDERAL DE MINAS GERAIS

Curso de Pós-Graduação em Engenharia Metalúrgica e de Minas

Walter de Moura

ESPECIAÇÃO DE CIANETO PARA REDUÇÃO DO CONSUMO NO CIRCUITO DE LIXIVIAÇÃO DE CALCINADO DA USINA DO QUEIRÓZ

CYANIDE SPECIATION TO DECREASE CONSUMPTION IN THE CALCINE LEACHING CIRCUIT AT QUEIRÓZ PLANT

Dissertação de Mestrado apresentada ao Curso de Pós-Graduação em Engenharia Metalúrgica e de Minas da Universidade Federal de Minas Gerais

Área de Atuação: Hidrometalurgia
Orientadora: Profa.Virginia Ciminelli

Belo Horizonte

Escola de Engenharia da UFMG

2005

Agradecimentos

Meus agradecimentos vão primeiro para AngloGold Ashanti Mineração, especialmente a Gerência de Metalurgia, pelo suporte físico e financeiro para a execução deste trabalho, sem o qual nada poderia ter sido feito.

Profundos e sinceros agradecimentos a equipe de técnicos que trabalharam, de forma séria e profissional na execução destes trabalhos, mesmo com as limitações de equipamentos e espaço: Altair Souza Sérgio, técnico de processo da usina do Queiróz, líder da equipe, José Marcelino dos Reis, José Carlos de Oliveira, também técnicos de processo contratados e Fábio José do Patrocínio, operador da usina do Queiróz. Os três primeiros são, além de profissionais, amigos de longa data, aos quais admiro.

Agradeço a Álida Diegues e a equipe do laboratório químico da planta do Queiróz que realizaram as análises químicas, fora da rotina diária, e que tiveram que agüentar minhas reclamações e exigências pelos resultados.

Agradeço também a minha mulher Claudia Márcia pelo suporte emocional e pela paciência de me agüentar nos momentos de mal humor e cansaço depois de me esperar nas noites que tive que ficar escrevendo este documento.

Summary

List of Figures

List of Tables

Resumo

A usina do Queiróz, localizada na cidade de Nova Lima, no estado de Minas Gerais, trata o minério semi refratário da mina Cuiabá com teor de ouro em torno de 7 g/t. O minério é moído e flotado para produzir um concentrado de pirita com teor de ouro da ordem de 30 g/t. A quantidade de concentrado gerado depende do teor de enxofre, que varia de 3 a 9% no minério. Este concentrado é ustulado para liberar o ouro ocluso na pirita. Quando a produção de concentrado é maior do que a capacidade do ustulador este é classificado por ciclonagem. A fração grossa do cyclone alimenta o ustulador e os finos, que contém pirita e pirrotita, vão diretamente para a lixiviação. O ouro é recuperado por cianetação de uma mistura de calcinado, produzido no ustulador, e dos finos do concentrado de sulfetos. Devido à presença de quantidade significativa de sulfetos no material, o consumo de cianeto é elevado para materiais oxidados - da ordem de 2,5kg/t -, o que resulta em um custo aproximado de US$ 300.000/ano. Uma investigação foi feita para a redução desse consumo. Os resultados mostraram que o cianeto é consumido através da formação de cianocomplexos metálicos (cerca de 50% do cianeto total da saída da cianetação), principalmente de zinco e de cobre, e da formação de tiocianato (cerca de 31%). Conforme demonstrado em ensaios de laboratório, a presença de cianeto de zinco, (o zinco é proveniente do minério), mostrou-se pouco deletéria à cianetação, uma vez que o complexo atua com lixiviante do ouro. A perda irreversível de cianeto, devida à formação de tiocianato, é atribuída à presença de "*by-pass*" de finos de sulfetos para os tanques de lixiviação. Para o minério de Cuiabá, a adição de nitrato de chumbo não se mostrou eficaz para a redução da formação de tiocianato ou do consumo de cianeto. Os resultados mostraram que aumentando o tempo de "pré-lime" para 10 a 16 horas e aumentando a dosificação de cal para a faixa entre 10 a 13 kg/t, é possível reduzir o consumo de cianeto entre 22 a 45%, dependendo do material tratado. Este resultado foi associado à redução da formação de cianocomplexos metálicos e implica em uma economia da ordem de US$ 15.000 a US$ 63.000/ano (5-20% do gasto anual com cianeto). Maiores reduções poderiam ser obtidas com a eliminação de ambas operações: da lixiviação direta de finos de sulfetos e o uso de sulfato de cobre como ativador na flotação.

Abstract

Queiróz Plant, located in Nova Lima City, Minas Gerais State, Brazil, treats a semi refractory gold ore of approximately 7 g/t of gold from the Cuiabá Mine. The ore is milled, 30% of the gold is recovered by gravity concentration, and floated to produce a pyrite flotation concentrate with a gold grade around 30 g/t. The final tonnage of concentrate depends on the sulfur grade, which varies from 3 to 9% in the ore. The concentrate is roasted to liberate the gold locked in pyrite. When the amount of concentrate produced is higher than the roaster capacity it is sent to cycloning. The underflow goes directly to roasting and the overflow, which contains pyrite and some pyrrhotite, goes directly to the leaching circuit. The gold of these fines are recovered by cyanidation in a blending with the calcine produced in the roaster. The cyanide consumption is high, approx. 2.5kg/t of calcine, costing about US$ 300.000/year. An investigation was carried out to reduce cyanide consumption in this circuit. The results showed that cyanide is consumed metal cyanocomplexes (50% of the cyanide consumed), mainly zinc and copper, and thiocyanide formation (31%). As demonstrated in laboratory, the presence of the zinc cyanocomplex is not deleterious to cyanidation, since the complex is an effective leaching reagent for gold. The irreversible loss of cyanide as thiocyanide is associated to the by-pass of the sulfide fines directly to the leaching circuits. Addition of lead nitrate did not reduce either the thiocyanide formation neither the cyanide consumption. The results also showed that increasing the pre-lime time to the range of 10 to 16 hours and increasing the lime addition in the pré-lime to the range of 10 to 13 kg/t of calcine, it is possible to reduce the cyanide consumption between 22 to 45%. This result was correlated to the reduction of cyanocomplexes formation and implied in operational savings between US$ 15,000 to US$ 63,000/year (5 - 20% of the total costs with cyanide). Further savings would require the elimination of both situations: fresh sulfides by-pass the roaster and copper sulfate addition in the flotation circuit.

1. Introdução

Minas Gerais, o maior centro de exploração aurífera do Brasil, começou a ser ocupada no final do século XVII, quando por volta de 1690, nas regiões de Sabará e de Ouro Preto, foram feitas as primeiras descobertas de veios de ouro e iniciada a extração do minério.

Por volta de 1725, a exploração de ouro começou a ser feita na Mina de Morro Velho, situada no Arraial de Congonhas do Sabará, utilizando-se processos bastante primitivos. Até as primeiras décadas do século XIX, a mina pertenceu à família do Padre Antônio de Freitas que realizou de forma irregular a extração do ouro pelos processos rudimentares da época, utilizando-se de mão-de-obra escrava. A história continua:

- Em 1830, a propriedade foi vendida ao Capitão George Francis Lyon, antigo Superintendente da Mina de Gongo Soco, na região de Caeté.

- Em 1834 a "*Saint John D'el Rey Mining Company"* adquiriu a Mina de Morro Velho das mãos do Capitão Lyon, e a companhia transferiu, então, suas atividades para Nova Lima, deixando definitivamente o município de São João Del Rei, onde iniciou suas atividades de produção de ouro, no Brasil.

- Em 1960, os ingleses transferiram o controle acionário para o grupo brasileiro liderado pelo Unibanco e a empresa passou a se chamar Mineração Morro Velho S.A.

- Em 1975, ocorreu a associação com a *"Anglo American Corporation"*, então a maior empresa de mineração de ouro do mundo.

- Em 1980, o Grupo Bozano Simonsen adquiriu a participação nacional da Morro Velho e foi quando ocorreu a transferência das ações do Unibanco para o Grupo Bozano Simonsen.

- Em 1985, entra em operação a usina do Queiróz tratando inicialmente o minério proveniente da mina Cuiabá.

- Em 1986, entra em operação o segundo circuito da usina do Queiróz que tratava o minério da mina de Raposos.

- Em 1993, como parte da reestruturação da *"Anglo American Corporation"*, o grupo transferiu os ativos brasileiros para o seu braço europeu, a Minorco.

- Em 1996, é fechada a antiga usina de Nova Lima, que tratava os minérios provenientes das minas Grande, Velha e de minas menores como Faria e Bicalho. A mina Grande, a mais profunda e antiga do mundo, é também fechada bem como as minas menores e o minério da mina Velha passa a ser tratado na usina do Queiróz.

- Em 1999, a AngloGold adquiriu todos os ativos de ouro da Minorco. A AngloGold se instala, assim, no Brasil já tendo uma experiência acumulada de mais de três décadas de atuação no país. Como um dos maiores produtores de ouro do mundo, a AngloGold responde por 10% da produção do metal.

- Hoje, a Mineração Morro Velho mudou de nome para *Anglogold Ashanti Mineração Ltda*, subsidiária da *AngloGold Ashanti South America*, que opera três outras operações, duas no Brasil e uma na Argentina.

Ao longo de sua história, a Mineração Morro Velho já produziu mais de 470 toneladas de ouro. Até pouco tempo atrás, operava em três minas próximas a Belo Horizonte: Mina Velha (Nova Lima), Mina Cuiabá (Sabará) e na Mina Engenho D'água (Rio Acima). Essas três unidades produziam, anualmente, uma média de 6,5 toneladas de ouro. Atualmente somente a mina Cuabá está operativa. O alto grau de pureza (99,99%) confere ao ouro produzido pela Morro Velho o Certificado *"Good Delivery"* da Bolsa de Londres. É uma garantia da sua aceitação no mercado mundial e um reconhecimento da longa tradição da empresa no mercado de ouro.

A AngloGold foi formada em janeiro de 1999 após uma fusão da divisão aurífera da *Anglo American* com suas afiliadas. Hoje a empresa, com sede na África do Sul, tem ações negociadas nas bolsas de Johannesburgo, Londres, Nova Iorque, Sydney, Bruxelas e Paris. Detendo integralmente o capital da Mineração Morro Velho, a AngloGold produz anualmente um total aproximado de 241 toneladas de ouro. A empresa opera em quatro continentes, com uma combinação de minas subterrâneas e a céu aberto. Está presente na África do Sul, Argentina, Austrália, Brasil, Estados Unidos, Mali, Guiné, Gana e Namíbia.

A Anglogold Ashanti Mineração Ltda está localizada entre as cidades de Raposos e Nova Lima, no estado de Minas Gerais, Brasil, como mostra a Figura 1.1. Como já foi dito, a Usina do Queiróz começou a operar em 1985 tratando 420.000 t/a de minério (toneladas por ano) e desde então vem aumentando sua capacidade de tratamento ano a ano chegando a 830.000 t/a em 2003 como mostra a Figura 1.2. Esse aumento de tonelagem fez com que a usina chegasse ao seu limite operacional de tratamento usando o circuito original. Foram feitos, então, ao longo desses anos, várias modificações no circuito. Entretanto, o ustulador passou a ser o “gargalo” o que tornou necessária a introdução de um circuito chamado “circuito de finos” que será discutido em detalhe adiante.

A usina do Queiróz apresentava originalmente dois circuitos:

(i) Circuito de Raposos – Este tratava o minério da mina de Raposos e de outras minas de minério não refratário. Tratava-se de um circuito composto de moagem, concentração gravítica, lixiviação convencional e CIP (*“Carbon In Pulp”*), eluição e eletrorecuperação. A mina de Raposos foi fechada em março de 1998 e o circuito desativado.

(ii) Circuito de Cuiabá – Este trata o minério da mina Cuiabá, que fica a 16km da usina. O minério é britado na superfície da mina e transportado por teleférico para a usina do Queiróz. O circuito de Cuiabá é semi-refratário, ou seja, parte de ouro pode ser recuperado por lixiviação direta com cianeto (70%), entretanto, outra parte (23,5%) encontra-se incluso em pirita (FeS_2) e não pode ser recuperado sem pre-tratamento. O circuito é composto de moagem, concentração gravítica, flotação, ustulação, neutralização, lixiviação do calcinado, CIP, eluição e electrodeposição. Os gases gerados na ustulação (SO_2) produzem ácido sulfúrico e os rejeitos de flotação são depositados em barragem.

A despeito da complexidade do circuito, este é bastante convencional para este tipo de minério. Entretanto, o circuito da Usina do Queiróz tem certas peculiaridades que o fazem diferente de outros em operação no mundo.

Nova Lima está a 20km de Belo Horizonte, capital do estado de Minas Gerais

Figure 1.1 - Localização da Cidade de Nova Lima e da Usina do Queiróz

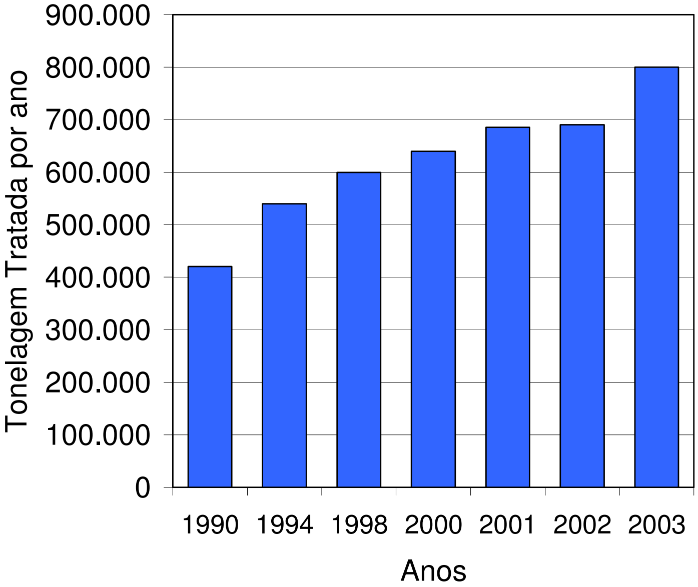


Figure 1.2 - Aumento do Minério Tratado na Usina do Queiróz desde 1990.

O teor médio de enxofre do minério que alimenta a usina é de 6,5%, variando de 3,5 a 9,5%. O teor de enxofre do concentrado de pirita, que alimenta a ustulação, é de 32% de enxofre, com um mínimo de 28%. Atualmente, quando a usina é alimentada com a média de 6,5% S é gerado em torno de 471t/d (toneladas por dia) de concentrado, que é o limite de tratamento do ustulador. Entretanto, a produção de concentrado pode variar de 254 a 689t/d, dependendo do teor de enxofre na alimentação. Quando a produção de concentrado é maior que 471t/d torna-se necessário a entrada em operação do "circuito de finos". Trata-se de um ou dois ciclones que classificam o concentrado em "finos" e "grossos". O ouro contido nos finos é mais fácil de recuperar que o contido nos grossos, podendo estes (os finos) serem desviados diretamente para o espessador de calcinado, desaguados e enviados para a lixiviação, juntamente com o calcinado. Ao mesmo tempo, nas duas primeiras células de flotação, chamadas de "células de grafite", é flotado o concentrado de grafite, que contém grafite e o ouro fino não recuperado na concentração gravítica. Este concentrado é também enviado para o espessador de calcinado onde se junta ao calcinado e aos finos. A Figura 1.3 mostra o circuito simplificado.

O calcinado - agora junção de calcinado puro vindo da ustulação com os finos do ciclone e com o concentrado de grafite - alimenta o circuito de lixiviação com 50% de sólidos depois de passar pelo espessamento, neutralização e "pré-lime", onde o pH é corrigido de 1,2 para 11,5 com cal virgem. Após a lixiviação, o calcinado é filtrado, a torta é repolpada com solução estéril da precipitação e enviada para o CIP, que recupera o ouro já solubilizado mas não recuperado na filtragem. A solução da filtragem é clarificada, desaerada e enviada para a precipitação do ouro com zinco. A solução estéril é enviada de volta à filtragem para lavagem dos filtros e para repolpamento da torta. O carvão do CIP é eluido e a solução rica da eluição se junta à solução da filtragem para cementação com pó de zinco.

Baseado na explicação acima se verifica o quanto é difícil controlar o consumo de cianeto neste circuito. A Figura 1.4 mostra a variação do consumo de cianeto de janeiro de 2002 até março de 1993. Quando o consumo de cianeto, com base no calcinado, é transformado em consumo de cianeto com base no minério que alimenta a usina, este é bastante reduzido, como mostra a barra menor na Figura 1.4, que é de certa forma “mascarado”, tendo em vista que a massa de calcinado é somente 15% da massa inicial de minério que alimenta a usina.

Usualmente, o consumo de cianeto nas usinas varia de 200 a 400g/t. Algumas operações de lixiviação em pilha, trabalhando com minérios oxidados e com poucos consumidores de cianeto, podem ter consumos da ordem de 30 a 40 g/t. O consumo de cianeto do circuito de Cuiabá, com base no minério que alimenta a planta, pode ser considerado baixo se comparado com outros minérios contendo pirita. Entretanto, o consumo é muito maior do que esperado (da ordem de 2500 g/t), considerando que se trata de calcinado que contém basicamente hematita que, em principio, deveria apresentar poucos consumidores de cianeto.

A Figura 1.4 mostra os consumos médios mensais de cianeto com base no minério alimentado na usina e com base no calcinado tratado. O alto consumo de cianeto mostrado em maio de 2002 pode ser parcialmente explicado pela parada do ustulador para manutenção. Durante este período o concentrado de flotação foi diretamente lixiviado e depositado em bacias para retomada posterior. Entretanto, os altos consumos nos outros meses não estão bem explicados.

O objetivo deste trabalho é reduzir o consumo de cianeto no circuito de calcinado da usina do Queiróz. Para se atingir este objetivo propõe-se:

(i) Caracterizar os consumidores de cianeto no circuito de calcinado
(ii) Determinar a origem desses consumidores de cianeto
(iii) Avaliar o balanço de cianeto para se verificar a confiabilidade das análises

Há várias razões que justificam a relevância do trabalho proposto:

(i) Primeiro, a otimização do consumo de cianeto reduzirá o custo operacional da usina. A usina do Queiróz gasta em torno de US$ 300.000/ano com cianeto, isso significa em torno de US$ 0,36/ton ou 2,76% do custo total.
(ii) Segundo, esse trabalho trará um maior entendimento do processo através da identificação das espécies de cianeto em solução que podem ter efeitos sobre a recuperação do ouro. Além disso, o trabalho vai ao encontro ao recente Código Internacional de Gerenciamento de Cianeto que exige estudos para a otimização dos consumos de cianeto nas usinas de ouro.

(iii) Terceiro, a redução do consumo de cianeto reduz um possível impacto ambiental, reduzindo desta forma o risco ambiental da usina.

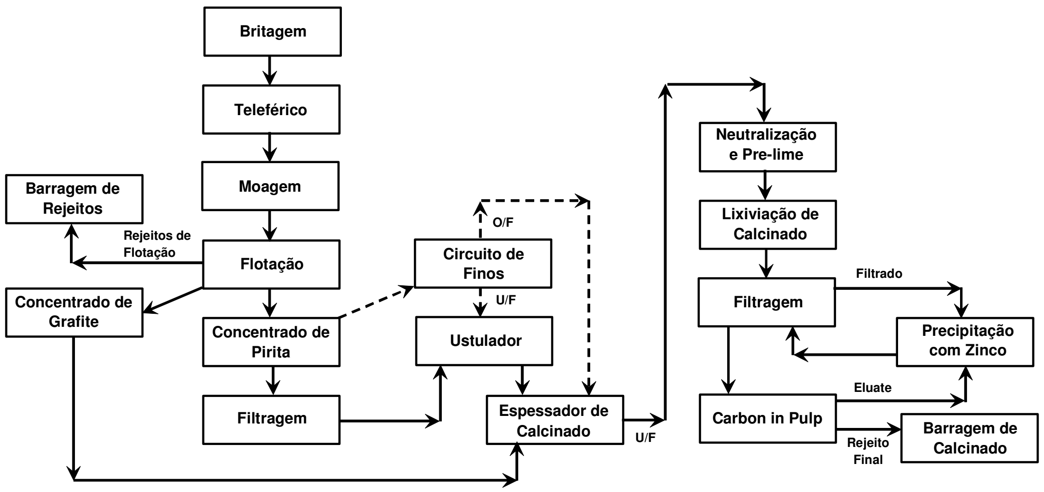


Figure 1.3 – Fluxograma de Blocos Simplificado da Usina do Queiróz Indicando o Circuito de Finos.

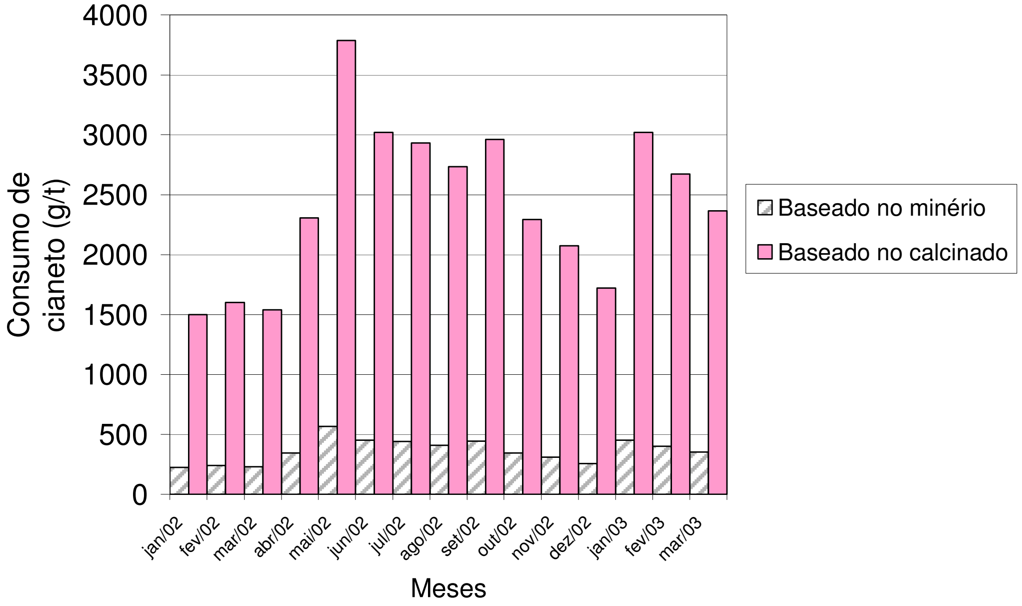


Figure 1.4 – Consumo de Cianeto na usina do Queiróz entre jan/02 e mar/03, com Base na tonelagem Tratada de Calcinado e com Base no Minério Alimentado na Planta.

Trabalhos preliminares mostraram que cobre, zinco e tiocianato são os maiores consumidores de cianeto – cianicidas - do circuito de lixiviação de calcinado. Entretanto, não há um entendimento mais amplo da origem e formação dos cianocomplexos e de como os mesmos podem ser reduzidos ou eliminados do circuito. A especiação é uma ferramenta poderosa para se entender os mecanismos da lixiviação e a influência desses complexos na recuperação do ouro.

A revisão bibliográfica será focada nos mecanismos de formação dessas espécies e nos métodos de análises. Finalmente, a preocupação da comunidade internacional com o uso do cianeto vem aumentando e é bem provável que em futuro próximo, as usinas sejam impedidas de descarregar cianeto para as barragens, como se faz hoje, ou seja, deverão ter descarga zero. Desta forma quanto menos cianeto for utilizado no processo menor será o custo de destruição do mesmo.

2. Literature Review

The consumption of cyanide is a significant operating cost for most gold plants, and should be closely regulated for safety, environment and economic reasons. At Anglogold Ashanti, the control of cyanide addition is based on the concentration at the first Pachuca tank in the circuit. There is no comprehensive study about the cyanide consumers; this will be the first cyanide speciation survey to help the processors to minimize cyanide consumption in this plant.

Only around 0.5 to 1% of the added cyanide is consumed to leach gold. Once in contact with the ore, cyanide can form a variety of metal complexes with different stabilities, and can also react with sulfur species to form thiocyanide, oxidize to cyanate and decompose to CO_2 and ammonia. Cyanide can also precipitate as ferrous-copper-cyanide, be adsorbed on the ore or carbon surface or even volatilize to the atmosphere, always depending on the leaching conditions. Each gold ore has a particular leaching behavior, due to its unique mineralogical composition and origin. Process differences include different “free” cyanide concentration and particle size distribution required for optimum gold leaching results.

A cyanide speciation survey can provide an accurate picture of how cyanide is being consumed in a gold leaching circuit. By determining the various solution species, the cyanide complexes can be determined or directly analyzed to quantify the amount of cyanide lost through the different reaction pathways. Depending on the relative proportions of the solution species, how they vary throughout the circuit and over time, recommendations can be made to reduce cyanide consumption. Cyanide addition and pH control should also be checked and recommendations can be made to improve it. Strategies for pre-treatment or different reagent addition points can also be investigated to avoid the formation of a particular cyanicide product.

Some of the delivers of this work are the indication of how effectively the plant is running and suggestions for improvements of how the plant should be running. Both, reduction of cyanide consumption and gold recovery optimization, can bring not only a financial result but also environmental savings considering less cyanide at the tailings pulp. Very few works have been done about this issue are found in the technical literature; most of the articles cover cyanide speciation at the laboratory conditions where many parameters are fixed due to the complexity of the an industrial circuit.

Considering the characteristics of the ore being treated at the Queiróz Plant, this review will focus on studies on copper, zinc, pyrite, pyrrhotite leaching and metal-cyanide complexing processes.

2.1. The Cyanide Ion

The calcine leaching at the *Queiróz* Plant is carried out by sodium cyanide, NaCN. This reagent, produced by Degussa, is a crystalline salt, which in solution dissociates to a positively charged sodium cation and a negatively charged cyanide anion, eq. (1). Only the cyanide ion is of major importance for cyanidation.

$$NaCN = Na^{+} + CN^{-} \qquad (1)$$

The cyanide ions contain a carbon and a nitrogen atom and an overall negative charge, since it has one excess electron. Theoretically, the negative charge can be dislocated either to the carbon or to the nitrogen atom (Lorosch, 2001). However, under the gold leaching conditions (aqueous solution) the negative charge is exclusively located at the carbon atom. The carbon atom has a lower electronegativity than nitrogen (lower tendency to attract electrons), which is preferred when making available electrons for bonding between cyanide and metal in transition metal complexes. The cyanide ion is the ionic or alkaline form of hydrogen cyanide or hydrocyanic acid (HCN), displaying the typical properties of a weak acid. The relative proportion of these two forms depends upon the pH of the system. The reaction between cyanide ion and water is expressed by eq. (2) (Smith and Mudder, 1991):

$$HCN = H^{+} + CN^{-} \text{ or } HCN + OH^{-} = CN^{-} + HOH \qquad (2)$$

At any particular pH and temperature the equilibrium of the system and the relative amounts of each species can be determined based on the eq. (3).

$$K_a = \frac{[H^+][CN^-]}{[HCN]} = 2.03 \times 10^{-10};\ pKa = 9.31 \text{ (at } 20^{o}C) \qquad (3)$$

Figure 2.1 shows the relationship between HCN and CN^- with pH in graphical form. Below pH 9,2 most of the cyanide present in solution is the HCN form. This species is extremely soluble in aqueous solution, with aqueous HCN in equilibrium with gaseous HCN (Adams, 1994). The hydrogen cyanide thus formed will volatilize to some extent.

$$HCN(aq) = HCN\ (g) \tag{4}$$

$$(\log\ [HCN(aq) = 1.394 + \log p_{HCN}) \tag{5}$$

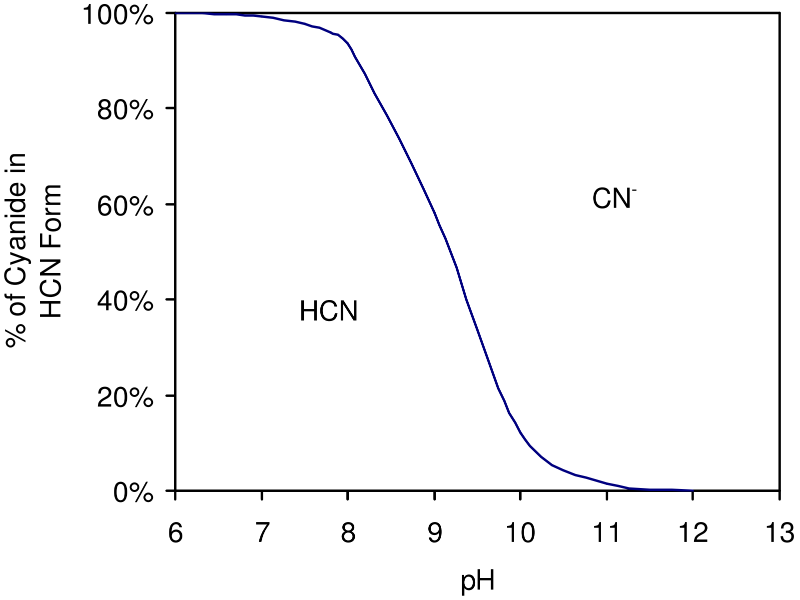


Figure 2.1 - Relationship Between HCN and CN^- with pH

2.2. Metal Complexes

To fully understand the chemistry of gold leaching it is necessary to understand the meaning of a metal complex. Lorosch (2001) in his book has made a very good summary of this matter, showing that cyanidation is mainly the chemistry of transition metal complexes and explaining the basic chemistry of these complexes.

“Metals like gold, silver, copper, zinc, iron, nickel, among others, are elements of the transition metals group in the periodic table. These elements are characterized by incompletely filled outermost electron sub shells, which are the d-sub shell for the above-mentioned metals. Since the distribution of electrons in the outer shell is preliminary responsible for the chemical properties of elements, the behavior of the transition group elements differ from the chemistry of other elements, such as the alkaline earth metals.

A chemical system always attempts to reach the most stable state. The most stable electron configuration for transition metals is the one of the noble gases of the same period. Therefore, transition metals show the preference to use free electron pairs of other molecules or ions to complete their outer electron shell and to reach a more stable electron configuration. In such a system, the electron pair donators are called ligands and the combination of the central atom or ion with the ligands forms a complex.

In aqueous solution, metal ions are solvated (solvation = formation of a complex using water molecules as ligands) caused by the ion-dipole interaction between the transition metal and water. During complexation, the water molecules are replaced by ligands, which become chemically bond to the metal atom or ion. By convention, the solvation or hydration is not included in the discussion of complex formation for simplification purposes.

Usually, a transition metal M^{n+} reacts with a certain number x of ligands L^{m-} according to the following eq.:

$$M^{n+} + xL^{m-} = ML_x^{(n+ +xm-)} \tag{6}$$

n = charge of the central atom or ion (usually n = 0, 1, 2, 3)
m = charge of the ligant (usually m = 0, 1, 2, 3)

Transition metal complexes can be anionic (e.g. $Zn(CN)_4^{-2}$), neutral (e.g. $Cu(CN)_2$) or cationic like the gold complex with thiourea ($Au[SC(NH_2)_2]_2^+$). However, all cyanide complexes occurring in the cyanidation process are anionic complexes".

2.3. How does Cyanide Leach Gold ?

Patented by MacArthur *et al.* in 1888, the gold leaching process by cyanide has now more than one century and has been the most important and usual process for gold recovery from ores. The process relies on the fact that gold dissolves in aerated cyanide solution to produce the gold cyanide complex (Jefferey and Breuer, 2000). Elsner's eq. (7) shows the stoichiometry by which this reaction occurs:

$$4\,Au + 8\,CN^- + O_2 + 2\,H_2O = 4\,Au(CN)_2^- + 4\,OH^- \tag{7}$$

In a relatively simple system, free gold quickly dissolves (Hedley and Tabachnick, 1968) – the requirements are that the gold be free and clean, that the cyanide solutions contain no impurities that might inhibit the reaction, and that an adequate supply of oxygen is present in the solution throughout the reaction period. In spite of the relative specificity of the gold-cyanide reaction, other metals and inorganic constituents react both with cyanide and to a certain extent with hydroxide, which must be present to maintain cyanide in its ionic form (Adrian and Miller, 1991). Many of the ore constituents such as quartz, silicate minerals, alkali metal carbonates are relatively inert to cyanide solutions. Other constituents, however, may react to a greater or lesser extent; such minerals are present in the ore in amounts several thousand times more than gold. This is particularly true now when more and more complex ores have been treated. Thus, cyanide consumption is very dependent on the ore mineralogy. Table II.1 shows the minerals usually found in association with gold.

2.4. Gold Leaching Kinetics

The gold leaching rate from ore, in practice, is a very complex kinetic problem involving a number of chemical, mass transport and mineralogical factors. It is therefore understandable that no fully adequate theoretical approach to the kinetics of the gold ores leaching has appeared so far. An empirical simple expression proved adequate for some purposes, has evolved from experience at the Council for Mineral Technology (MINTEK, South Africa) (Nicol, 1985) in the leaching of a number of South African ores. The expression has the following form:

$$-\frac{dC_S}{dt} = k_P (C_S - C_S^e)^2 \tag{8}$$

where:

C_S	Concentration of gold in the ore at time t
C_S^e	Concentration of gold at the infinite time (i.e. the minimum achievable in the residue)
k_P	Rate Constant

Table II.1 - Minerals Associated with Gold in a Sulfide Ore (Smith and Mudder,1991)

Element		Sulphide	Arsenides	Antimonides	Selenides	Tallurides
Iron		FeS Pyrrhotite FeS_2 Pyrite/ marcasite	FeAsS arsenopyrite			
Cobalt			CoAsS Coballite			
Nickel		$(Fe,Ni)_9S_8$ pentlandite				
Gold	Au native gold Au,Ag Electrum			$AuSb_2$ aurosibite		$AuTe_2$ krennerite/ calaverite
Silver	Ag native silver Ag,Au electrum	Ag_2S argentite (Pb,Ag)S argentiferous galena	Ag_3AsS_3 prousite $(Cu,Fe,Ag)As_4S_{13}$ argentiferous tennantite	Ag_3SbS_3 pyragyrite $(Cu,Fe,Ag)Sb_4S_{13}$ argentiferous tetrahedrite	Ag_2Se naumannite	Ag_2Te hessite
Mercury		HgS Cinnabar				
Copper	Cu Native copper	Cu_2S chalcocite CuS covellite Cu_5FeS_4 bornite $CuFeS_2$ Chalcopyrite	$CuAsS_4$ enargite $(Cu,Fe)As_4S_{13}$ tennantite	$(Cu,Fe)Sb_4S_{13}$ tetrahedrite		
Lead		PbS Galena				
Zinc		ZnS Sphalerite				
Carbon	C graphite/ Amorphous C					
Arsenic		AsS realgar As_2S_3 Orpiment				
Antimony		Sb_2S_3 Stibnite				
Bismuth	Bi native bismuth	Bi_2S_3 bismuthinite				

Integration of this expression gives:

$$\frac{1}{(C_S - C_S^e)} = k_P t + \frac{1}{(C_S^o - C_S^e)} \tag{9}$$

The constant k_P can be obtained from the results of batch leaching testwork by plotting the left-hand side of this expression versus time. It should be noted that C_S^e is the ultimate gold tailings but can also be an adjustable parameter that should be varied for maximum linearity of the plot.

2.5. Behavior of Pyrite and Pyrrhotite in Cyanide Solution

In the Cuiabá ore, part of the gold is found free while the other part is mainly locked in pyrite and pyrrhotite. Pyrite (FeS_2) is a more inert sulfide than pyrrhotite (FeS), which is very reactive, thus causing problems for gold dissolution due to oxygen and cyanide consumption under the leaching conditions. Dunn *et al.* (2000) studied the dissolution of the hexagonal and monoclinic pyrrhotite in cyanide solutions. It has been reported in earlier studies that pyrrhotite liberates a sulfur atom, which could react with cyanide to form thiocyanide as described by the eq. (10):

$$Fe_nS_{(n+1)(s)} + CN^-_{(aq)} = nFeS_{(s)} + SCN^-_{(aq)} \tag{10}$$

Oxidation and complexation of ferrous sulfide by oxygen and cyanide ions then yields sulfate and ferrocyanide ions, according to the eq. (11):

$$FeS_{(s)} + 2\ O_{2(aq)} + 6\ CN^-_{(aq)} = [Fe(CN)_6]^{4-}_{(aq)} + SO_4^{2-}{}_{(aq)} \tag{11}$$

Dunn *et al.* (2000) suggested that under cyanide leaching conditions, regardless the formation of ferrous cyanide complex being favored by thermodynamics, pyrrhotite consumes cyanide ions and oxygen to form instead iron (III) hydroxide:

$$4\ FeS_{(s)} + 3\ O_{2(aq)} + 4\ CN^-_{(aq)} + 6\ H_2O = 4\ Fe(OH)_{3(s)} + 4\ SCN^-_{(aq)} \tag{12}$$

The formation of iron (III) hydroxide during pyrrhotite oxidation has been supported by Morro Velho practice and recent works, although it was inferred that sulfate ions are also liberated. *Morro Velho's* practice so far is the use of “pre-lime” or “pre-aeration”, where the iron liberated due to pyrrhotite oxidation is precipitated as hydroxide to form a passive layer on the surface of unreacted pyrrhotite, thus inhibiting further oxidation of the mineral according to the eq. (13):

$$FeS_{(s)} + 9\ O_{2(aq)} + 10\ H_2O = 4\ Fe(OH)_{6(s)} + 4\ SO_4^{2-}{}_{(aq)} + 8\ H^+{}_{(aq)} \tag{13}$$

Dunn *et al.* (2000) concluded that crystallography is also important to understand the behavior of pyrrhotite in leaching solution. Classification as a generic term like "pyrrhotite" is not enough, hexagonal ($Fe_{0.90}S$) and monoclinic ($Fe_{0.88}S$) pyrrhotite showed appreciable difference in the oxidative dissolution behavior when leached in cyanide solution over a 24-h period. The rate of dissolution increased with an increase in oxygen partial pressure, decrease in particle size, and increase in temperature. Under all sets of experimental conditions, the monoclinic form exhibited greater reactivity than the hexagonal form. In particular, a fraction of monoclinic pyrrhotite of particle size 25 – 45 μm caused severe depletion of dissolved oxygen in solution, an effect that was not observed with the hexagonal form. The conclusion was that the difference in reactivity is due to the nature of the iron-oxyhydroxide product layers formed during the oxidation. One of which offers some protection to dissolution under all conditions used in that study, while the other offers protection only at elevated temperatures.

An interesting behavior of pyrite is its capacity to adsorb gold in oxygen-free cyanide solutions (Rees and van Deventer, 2000). The authors found that more than 98% of the gold from an oxygen-free cyanide solution and low free cyanide can be adsorbed on the pyrite surface, however increasing the free cyanide concentration the adsorption decreases to values close to zero. If activated carbon and free cyanide are present, the preference is gold adsorption on the carbon.

2.6. Behavior of Copper in Cyanide Solution

In a system where gold and copper are present, both are leached, but copper and complexed cyanide are not recovered after gold is removed from solution (Dreisinger *et al.*, 2002). Copper, if present in high concentrations in solution, also disturbs the conventional zinc precipitation process increasing zinc consumption. This leads to a significant operating cost penalty by excess of cyanide consumption, loss of a valuable copper by-product and significant cost in cyanide destruction in the tailings. In the Anglogold's calcine, copper is not present in significant concentration. However, preliminary tests showed that this might depend from where the ore comes from. In some periods, cyanide consumption was found to increase due to the presence of copper in the calcine.

Copper cyanide can be dissolved in excess cyanide to form cyanocuprate ions $Cu(CN)^{2-}$, $Cu(CN)_3^{2-}$ and $Cu(CN)_4^{3-}$ in aqueous solution. Dreisinger *et al.* (2002) made a very complete investigation about thermodynamics of the aqueous copper-cyanide system and confirmed that theses species undergo to the following successive equilibrium steps in the reaction with free cyanide and undissociated hydrocyanide acid:

$$CuCN = Cu^{+} + CN^{-} \qquad K_{sp} \tag{14}$$

$$CuCN + CN^{-} = Cu(CN)_2^{-} \qquad K_2 \tag{15}$$

$$Cu+ + 2CN^{-} = Cu(CN)_2^{-} \qquad \beta_2 \tag{16}$$

$$Cu(CN)_2^{-} + CN^{-} = Cu(CN)_3^{2-} \qquad K_3 \tag{17}$$

$$Cu(CN)_3^{2-} + CN^{-} = Cu(CN)_4^{3-} \qquad K_4 \tag{18}$$

$$HCN = H^{+} + CN^{-} \qquad K_a \tag{19}$$

There is a good agreement for the HCN dissociation constant (K_a) amongst the published data; the recommended value for K_a at 25°C is $10^{-9.21}$. However, the equilibrium constants for copper cyanide complexes differ between authors due to the different methods of measurements and processing the data. A more complete discussion about these discrepancies is beyond the scope of the present work.

Total cyanide concentration, pH, cyanide-copper molar ratio and temperature determine distributions and equilibrium potentials of copper cyanide species. By increasing CN:Cu molar ratio, the distribution of copper cyanide species shifts more completely to the highly coordinated complex ($Cu(CN)_4^{3-}$). Increasing temperature, results in decreasing the stability constants and thus the distribution of copper species shifts to the lowly coordinated complexes.

2.7. Behavior of Zinc in Cyanide Solution

Some preliminary tests showed zinc as one of the strongest cyanide consumer at the calcine leaching circuit. The behavior of zinc has not been well understood yet, so it is difficult to fully understand the leaching mechanism.

Gold bearing ores may contain a variety of zinc minerals, which are able to react with cyanide. The amphoteric characteristic of zinc, which allows its dissolution in acids as well as in alkalis, is the reason that cyanicidic actions of dissolved zinc usually are suppressed efficiently (Lorosch, 2001). Gold losses, can be caused by an accumulated zinc concentration in the cyanidation circuit. Table II.2 shows the solubility of zinc minerals in cyanide solution.

Once the minimum cyanide concentration required for gold extraction and oxygen are present, a cyanicide influence of zinc on gold dissolution is rare. However, if silver is present as argentite, the zinc cyanide has a strong detrimental effect on its dissolution due to the necessity of high free cyanide concentration in solution to dissolve silver minerals.

Zinc cyanide complexes is formed from metallic zinc and zinc minerals, as showed before, and the most stable complex under gold leaching conditions is the $Zn(CN)_4^{2-}$ species. Metallic zinc is readily soluble in cyanide solution with the reaction being essential for gold (and precious metals) cementation (Lorosch, 2001). However, at high zinc concentration the dissolution can became difficult. Since metallic zinc does not occur as a mineral in gold bearing ores, the relevant chemistry is discussed below.

Table II.2 - Solubility of Zinc Minerals in Cyanide Solutions (Hedley and Tabachnick, 1968)

Mineral	Formula	Zinc Dissolution (%)
Willemite	Zn_2SiO_4	13.1
Calamine (Hemimorphite)	$H_2Zn_2SiO_5$	13.4
Sphalerite	ZnS	18.4
Franklinite	$(Fe,Mn,Zn)O\ (Fe,Mn)_2O_3$	20.2
Hydrozincite	$3\ ZnCO_3\ 2H_2O$	35.1
Zincite	ZnO	35.2
Smithsonite	$ZnCO_3$	40.2

The most comum mineral, sphalerite (ZnS), dissolves according to the eq. (20) (Hedley and Tabachnick, 1968):

$$ZnS + 4\ CN^{-} = Zn(CN)_4^{2-} + S^{2-} \tag{20}$$

In addition to the zinc cyanide complex, the reaction produces sulfide ions, which are usually removed from the equilibrium reaction by oxidation to thiosulfate or sulfate.

The oxide zinc minerals such as Zincite, Willemite and Smithsonite also produce $Zn(CN)_4^{2-}$ as a final product in the presence of cyanide; these minerals also dissolve as zincate due to the amphoteric feature of zinc. The alkali content added to the cyanide solution as well as the alkali produced due to the reaction with cyanide produce zinc dissolution according to the eq. (21):

$$ZnO + 2\ OH^{-} = ZnO_2^{2-} + 2\ H_2O \tag{21}$$

It may also form zincate out of zinc cyanide complexes, in alkaline condition, liberating the bound cyanide ions, according to the eq. (23).

$$Zn(CN)_4^{2-} + 4\ OH^{-} = 4\ CN^{-} + ZnO_2^{2-} + H_2O \tag{23}$$

This is the main reason why zinc cyanide is not expected to interfere in gold leaching.

2.8. Behavior of Reduced Sulfur Species in Cyanide Solution

As mentioned above, the sulfide ion (S^{2-}) can be formed as a product of many mineral dissolution, such as pyrrhotite, spharelite, copper sulfide, among others. In the presence of oxygen, the sulfide ions may be oxidized, ultimately forming sulfate. However, the behavior of sulfide in cyanide solutions is somewhat complex (Coderre and Dixon, 1999). The reactions are illustrated in Figure 2.2.

Luthy and Bruce (1979) concluded that the dominant reaction pathways for the formation of thiocyanide from the oxidation products of sulfite were by reaction with either polysulfides or thiosulfate.

$$S_xS^{2-} + CN^- = S_{x-1}^{2-} + SCN^- \tag{24}$$

$$S_2O_3^{2-} + CN^- = SO_3^{2-} + SCN^- \tag{25}$$

Reaction (24) proceeds faster than reaction (25) by approximately three orders of magnitude. The undesirable consumption of cyanide ion through the formation of such stable complex as thiocyanate, which in turn increases the treatment costs associated with reagents, is a major variable in the financial feasibility of a gold mine (Dunn *et al.*, 2000). The presence of S^{2-} in cyanide leach solution may also result in the formation of a passive layer of sulfur on gold surface, which significantly reduces the gold leach rate. This layer can be removed under high cyanide concentration solution resulting in high reagent consumption (Breauer and Jeffrey, 2000).

To decrease these detrimental effects of reduced sulfur species on cyanidation, Breauer and Jeffrey (2000) have reviewed the use of lead salts to eliminate the sulfite ions from the solution. Hedley and Tabachnick (1968) affirmed that only the lead nitrate had a pronounced effect to accelerate the S^{2-} oxidation. This effect was not due to precipitation of PbS, as the amount of lead nitrate added in the experiment could not have precipitated more than 2.5 % of the S^{2-} in solution. Therefore, the main role of Pb ions may be to catalyze the oxidation of the hydrosulfite ions. This hypothesis was confirmed by Breauer and Jeffrey (2000).

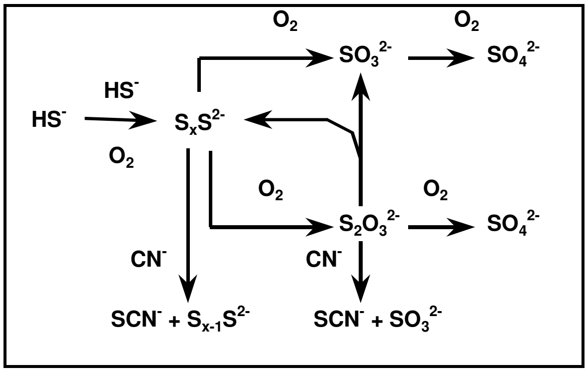


Figure 2.2 - Possible Reactions of Cyanide with Reduced Sufur Species in Aqueous Solution to Yield Thiocyanide (Luthy and Bruce, 1979)

2.9. Behavior of the Metal Complexes at the CIP (Carbon in Pulp) Circuit

At the Queiróz plant, the calcine, after filtration and repulping with barren solution from the zinc precipitation, is sent to the CIP circuit for recovering of gold lost in the filtration and zinc precipitation. Cyanide complexes are also adsorbed, to some extension, onto activated carbon.

Before 1980, gold adsorption mechanism was explained as the adsorption of AuCN or even metallic gold onto the carbon surface. However, MacDougall *et al.* (1980) and Adams (1989) showed that under typical leaching condition, gold is adsorbed as a linear gold cyanide complex, $Au(CN)_2^-$ predominantly forming an ion pair, e.g. with Ca^{2+}. This is supported by the findings that the nitrogen-nitrogen distance in the $Au(CN)_2^-$ of 0.66 nm is almost exactly equal to the c-axis spacing of 0.669 nm in graphite.

This mechanism also can explain the activated carbon selectivity for the cyanide complex of gold, silver, mercury and for $Cu(CN)_2^-$ ion. They all form linear complexes of similar dimensions.

The adsorption preference of these linear complexes has been found to be:

$$Au(CN)_2^- > Hg(CN)_2 > Ag(CN)_2^- > Cu(CN)_2^- \qquad (26)$$

The increase of the number of cyanide ligands changes the structure of the complex from linear to spatial. The consequence of that is a structural change increasing the distance between the nitrogen ions. Consequently the adsorption preference on activated carbon decreases with increasing number of cyanide groups:

$$Au(CN)_2^- >> Cu(CN)_3^{2-} > Zn(CN)_4^{2-} > Fe(CN)_6^{4-} \qquad (27)$$

Experimental results clearly indicate that when the CN:Cu ratio is 2:1 the copper loading is maximized, decreasing strongly when cyanide concentration increases, thus changing the complex to tri or tetracyanide species. The copper adsorption is most pronounced at low pH and low cyanide concentration, due to an HCN/CN equilibrium change (Lorosch, 2001).

To eluate gold from carbon the most usual processes used are the AARL and ZADRA. These two processes use an eluate with NaCN and NaOH (usually 2% and 1% respectively). Queiróz Plant uses only NaOH. Boshoff (1993) explains that gold can be eluated from activated carbon by free-cyanide solution when carbon contains low copper loading (less than 1200 g/t). In this case, the severity of acid wash (temperature and acid concentration) can also be reduced without any detrimental effect on gold recovery.

2.10. Behavior of Lead in Cyanide Solution

Since the sulfide minerals are, to some extent, soluble in cyanide solutions, there will always be some sulfur species in the leaching solution (Jeffrey, 2000). Lead, thallium, bismuth and mercury can increase the dissolution rate of gold in cyanidation (Tavani, 2000), lead being the most popular ions used in gold plants. They also comment that there are at least 4 proposed mechanisms for the lead action in gold leaching circuits:

- Precipitation of sulfides as highly insoluble lead sulfides preventing consumption of cyanide and oxygen. However, this mechanisms is not supported by Hedley and Tabachnick's work (1968). In their experiments it was shown that a lead concentration equivalent to only 2.5% of the sulfide ions present in solution, i.e., the effect in sulfide precipitation could not be caused by PbS precipitation because the amount of lead nitrate added could not precipitate stoichimetrically more than 2.5% of the sulfide in solution.
- Surface reaction on sulfide particles forming an insoluble lead sulfide layer preventing dissolution of these sulfides.
- Precipitation of lead sulfide, which undergoes further oxidation to soluble lead sulfate recycling the lead to solution.
- Contact reaction of lead onto the gold surface which then acts as a secondary oxidizing agent. Jeffrey and Breuer (2000) mentioned that a protective layer of the type $Au\backslash S_x$ formed on gold surface hinders the rate of gold leaching.

In fact there is not a firm explanation of the lead mechanisms. Hedley and Tabachnick (1968) suggest that lead accelerated sulfide oxidation. Similarly, Jeffrey and Breuer (2000) concluded that lead catalyzes the oxidation of the hydrosulfide ions.

Lead efficiency depends very much of the circuit where it is used. In experiments using rotating disc electrodes (Tavani, 2000), small amount of sulfide ions, such as 5 mg/L were shown sufficient to significantly decrease the gold leaching rate. Lead increased the rate at small concentration however it decrease the rate if a large amount was added to the solution. Ores systems look to control the amount of lead available by adsorption, since reduction of the leaching gold rate was not significant even at high lead concentration. It was also observed that the detrimental effect of the sulfide ore can happen if the lead addition is delayed, since there was a time lag between the lead addition and the increasing leaching rate when large concentration of sulfide ions is present. In other words, the sooner lead is added to the circuit the better (Tavani and Browner, 2000). Kondos *et al.* (1995) showed that the lead addition in a leaching system can be even more effective than the introduction of oxygen.

Lead addition is known to effectively improve leaching of some gold ores but lead can be also detrimental to the process when not properly adjusted. It can inhibit the kinetics of gold dissolution and increase reagent consumption. Having too large lead nitrate addition is also a waste of reagent. In 1994, CANMET carried out a study to improve the understanding of plant practice related to lead nitrate addition. The conclusions from this work were:

(i) There is an optimum lead addition rate for a given circuit and ore type,
(ii) Maximum benefits may occur if pre-aeration is employed,
(iii) Laboratory testwork has to be very carefully controlled for valid conclusion to be drawn,
(iv) Lead is, most of the time, in very low concentration in the cyanide solutions.

The most common method used to control lead addition is based on the relationship between the gold content in the leach residue, the dissolved oxygen concentration and the free cyanide concentration of the leach solution (Deschênes *et al.*, 1999). Because the lead concentration is very low, in general below 0.5 mg/L, it is not possible to use the analysis of the dissolved lead to control its addition, only Doyon Mine uses this method. At Kiena Mines they increase the lead nitrate addition rate when a decrease of the dissolved oxygen level is noticed.

Two methods are used to monitor the addition of lead nitrate in leaching circuit and these methods are not a direct measurement of lead in solution. The first method is called the “Prussian Blue Test”, used at Doyon Mine and at Morro Velho Operation in Brazil, until it’s shut down. A solution with ferric ions is added to a slightly acidified cyanide solution and lead nitrate requirement is related to the color obtained. A yellow color indicates no change of the addition rate; an orange color means a small increase; a blue color indicates a medium increase and a dark color indicates a large increase in the addition rate. At Morro Velho Operation the blue and dark solution was filtered; in the solids stay the ferrocyanide while in the filtrated solution contains the thiocyanide formed. If only ferrocyanide was in solution, this indicates that the pre-aeration (pre-lime) time should be increased, but if thiocyanide is also present the lead nitrate should be also increased. This approach can not be used by itself as a reliable control. The second method of monitoring the lead nitrate addition is called the “Reducing Power Method” (RPM). It was used at Lupin Gold Mines which is presently shut down. The Reducing Power is determined by titration of the leach solution with potassium permanganate. A relationship is determined between the reducing power and the optimum leaching conditions.

These two methods were studied by Deschênes *et al.* (1999) to provide to the gold industry a good control of the lead nitrate addition. The Prussian Blue Test was shown to be not robust enough and the degree of reliability is estimated to be around 40%; it is also difficult for automation. But the determination of the Reducing Power by potentiometric titration provides measurements that can be interpreted in terms of concentration of two constituents associated with cyanide leaching, namely thiosulfate and thiocyanide. The explanation of this method is not clear (Deschênes *et al.*, 2000). The titrations appear to follow complex oxidation pathways, and the stoichiometries are not in accordance with simple oxidation to sulfate. There is some evidence that the rates of oxidation and/or pathways of thiosulfate oxidation and thiocyanate are somewhat different in the mixed systems than when the species are measured independently.

2.11. How to Analyze Cyanide and Its Complexes?

The following definitions are provided by the International Cyanide Management Code published in 2003 to clarify the terminology related to cyanide analyses:

“Free Cyanide (CN$_{FREE}$): Only hydrogen cyanide and the cyanide ion in solution can be classified as “free” cyanide. The proportions of HCN and CN$^-$ in solution are established according to their equilibrium equation and by the solution pH.

(i) Methods used to detect free cyanide should not alter the stability of weaker cyanide complexes, as they may otherwise be included in the free cyanide results.

(ii) Methods used to detect free cyanide should be clear of interferences due to the presence of high concentration of more stable cyanide complexes or the cyanide forms. If not, the interference must be quantified and allowed for in the results.

WAD Cyanide (CN$_{WAD}$): Unlike the definition of “free cyanide”, which identifies the specific cyanide species being measured, WAD cyanide (**W**eak **A**cid **D**issociable) refers to these cyanide species measured by specific analytical techniques. WAD cyanide includes those cyanide species liberated at a moderate pH of 4.5, such as HCN (aq) and CN$^-$, the majority of Cu, Cd, Ni, Zn, Ag complexes and others with similar low dissociation constant.

(i) Methods used to measure WAD should be free from interferences due to the presence of high concentrations of more stable cyanide complexes or other cyanide forms. If not, the interference must be quantified and allowed for in the result.

Total Cyanide (CN$_{TOTAL}$): This measurement of cyanide includes all free cyanide, all dissociable cyanide complexes and all strong metal cyanide including ferro-cyanide ($Fe(CN)_6^{-4}$), ferri-cyanide ($Fe(CN)_6^{-3}$) and portions of hexacyano cobalt ($Co(CN)_6^{-3}$) and those of gold and platinum. Only the related or derived cyanate (CNO$^-$) and thiocyanate (SCN$^-$) compounds are excluded from the definition of total cyanide.

- Methods used to determine total cyanide must be shown to be capable of quantitatively determining all stable complexes of cyanide, including the cobalt cyanide complex. If the method determines other analytes as well (e.g. include SCN$^-$), those analytes need to be determined separately and allowed for in the total result.”

The plants usually use titration with $AgNO_3$ to control the process and express the results as concentration of NaCN in solution. Thus, it is necessary to make one more definition for the proposes of this work. Because Anglogold uses this protocol to express cyanide concentration, it will be necessary to make reference when the cyanide concentration is expressed as CN^-.

Titratable Cyanide (CN_{TITR}): Titratable cyanide is typically free cyanide plus a portion of metal-cyanide complexes such as zinc and copper cyanides, though the amount of metal-cyanide titrated is dependent upon the titrant and indicator used.

Figure 2.3 summarizes the cyanide terminology. Several methods can be used to determine cyanide in plant solutions (Adam, 2000; International Cyanide Code, 2003) as summarized in Table II.3. Each of these methodologies has its limitation. Any of these assay methods can be used, provided that their individual limitation is taken in account, particularly when base metals or high levels of thiocyanide are present. Copper provides the most significant problem for cyanide analysis due to the range of $Cu(CN)_x^{y-}$ complexes formed as equilibrium mixtures in copper-cyanide solutions, the precipitation of CuCN and the catalytic oxidation to cyanate (CNO^{--}).

Thiocyanate (SCN^-) and cyanate are other products that may be present in cyanide leach liquors. They can be determined using standard (APHA) methods. Other methods, using ion-chromatographic techniques can also be used.

2.12. Cyanide in Solid Waste

To determine if any metal complex was precipitated or adsorbed or is presence in solution with the ore, cyanide can be analyzed in the solid residue from a leaching circuit. According to the Standard Methods (ALPHA), determination of soluble cyanide in the solid waste requires sample leaching with distilled water until solubility equilibrium is established, one hour stirring being usual. Cyanide analysis is then performed on the leachate.

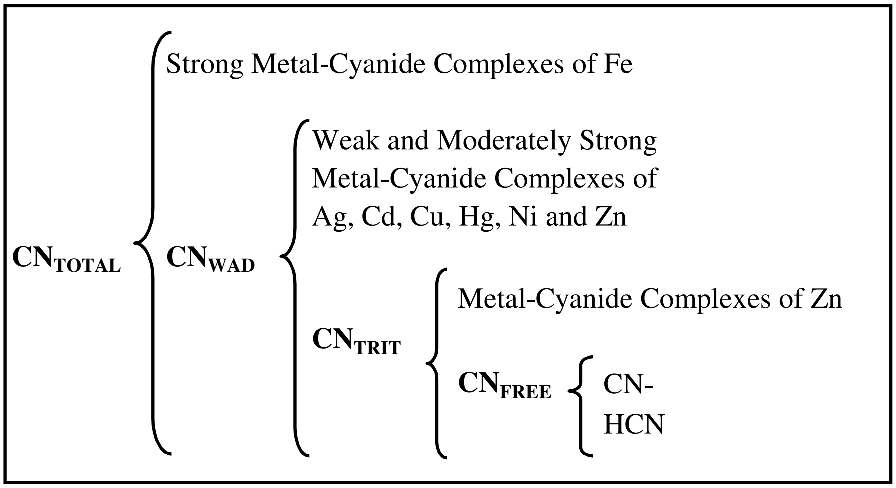


Figure 2.3 - Cyanide Terminology

Low cyanide concentration may indicate the presence of sparingly soluble metal cyanide. The cyanide content of the leachate is indicative of residual solubility of insoluble metal cyanide in waste. High levels indicate soluble cyanide in the solid waste. The insoluble cyanide of the solid waste can be determined using the total cyanide method by placing the solid in the distilled water and following the total cyanide (CN_{TOTAL}) analysis procedure.

2.13. Thiocyanide Analysis

Thiocyanide can be analyzed by a colorimetric method or by titration with silver nitrate. The main method to be used in this work will be titration. Because free cyanide competes for silver, each sample will be first acidified with 0.5 mol/L H_2SO_4 and sparged with nitrogen gas or air for 30 min to drive off free cyanide as HCN gas. The sample will be then re-acidified with 6 mol/L HNO_3, and 2 mL of ferric nitrate and then filtrated to remove any ferrocyanide precipitated. Then 2 mL of ferric nitrate (Fe^{3+} and SCN^- form an intense red color) is added as indicator just prior to titration.

The colorimetric method will be used sporadically to check the above procedure following the Standard (ALPHA) Methods (4500-CN^- M.).

2.14. Cyanate (CNO^-) Determination

Based at the Standard (ALPHA) Methods (4500-CN^- L) the CNO^- is converted to ammonium carbonate, $(NH_4)_2CO_3$, by acid hydrolysis at elevated temperature. Ammonia (NH_3) is determined before the conversion of the CNO^- and again afterwards. The CNO^- is estimated from the difference in NH_3 found in the two tests. The NH_3 measurement can be made by selective electrode or colorimetric method.

2.15. Interferences on Cyanide Analyses

The main interferences on the free cyanide determination by silver nitrate titration results from the presence of sulfides, oxidizing agents, base metals, mainly copper, thiocyanide, nitrate and nitrate, carbonates, thiosulfates, sulfites and other related sulfur compounds present in the cyanide solution (Lorosch, 2001; Mudder and Smith, 1991). To avoid any impact on the results of the analysis, it is important to understand the chemistry of each type of interference and to know how to avoid them.

The total cyanide methodology has been studied by Solujic and Milosavijevic (2000) and they found that it would be very difficult to have the same results from different laboratories. The method has low reproducibility and repeatability and serious interferences even from many ubiquitous species.

They mentioned that the problems associated with the approved methods for cyanide determination has prompted the EPA to look for solutions provided by different analytical methods. The EPA Methods 1677 (Weak Acid Dissociable Cyanide by Ligand Exchange/ Flow Injection/Amperometric Technique) has been tested since last year. The method tested is based on a recent study made by Milosavijevic *at al.* (1995) and has significant advantages over the EPA/ASTM approved procedures.

Table II.3 - Cyanide Analytical Methods (International Cyanide Management Code, 2003)

	Method	Comments
Free Cyanide and Titratable Cyanide[2]	$AgNO_3$ titration	▪ Preferred method ▪ For process solutions primarily above 1 mg/L ▪ LQL[1]: 1 mg/L ▪ $HCN(aq)$, CN^-, $Zn(CN)_x$, parts of $Cu(CN)_4^{3-}$
	$AgNO_3$ titration with potentiometric endpoint determination	▪ Alternative Method ▪ Precise method of endpoint determination ▪ Measures same species as primary method
	Micro diffusion of HCN from static sample into NaOH [ASTM D4282]	▪ Alternative Method ▪ Close to “Free Cyanide”
	Ion Selective Electrode	▪ Alternate Method ▪ Close to “Free Cyanide”
	Direct Colorimetry	▪ Alternative Method ▪ $HCN(aq)$, CN^-, $Zn(CN)_4^{2-}$, parts of $Cu(CN)_4^{3-}$
	Amperometric Determination	▪ Alternate Method ▪ Measures same species as primary method
WAD Cyanide	Manual distillation pH 4.5 + potentiometric or colorimetric finish [ISO/DIS 6703/2, DIN 38405 Part 13.2: 1981-02]	▪ Preferred method ▪ LQL[1]: 0.05 mg/L ▪ $HCN(aq)$, CN^-, $Zn/Cd/Cu/Ni/Ag(CN)_x$ ▪ Better results than ASTM method in the presence of high copper concentration
	Amenable to chlorination (CN Total – non-chlorinated part) [ASTM D2036-B, US-EPA 9010]	▪ Alternative Method ▪ Measures same species as primary method
	SFIA in-line ligant exchange + amperometric finish [US-EPA OIA-1677]	▪ Alternative Method ▪ Measures same species as primary method
	FIA In-line ligand exchange + amperometric finish [ASTM D4374]	▪ Alternative Method ▪ Measures same species as primary method
	Picric Acid, Colorimetric determination	▪ Alternative Method ▪ Measures same species as primary method
Total Cyanide	Manual batch distillation + titration/potentiometric or colorimetric finish [ISO/DIS 6703/1, DIN 38405 Part 13.1: 1981-02	▪ Preferred method ▪ LQL[1]: 0.10 mg/L ▪ $HCN(aq)$, CN^-, $Zn/Cd/Cu/Ni/Ag/Fe(CN)_x$, parts of $Au/Co/Pt/Pd(CN)_x$
	SFIA, in line UV irradiation, micro-distillation + colorimetric finish [ASTM D4374]	▪ Alternate Method ▪ Measures same species as primary method

[1] LQL, Lower Quantization Level, is defined as about 3 times Detection Level or 10 times the Deviation at near blank level.

[2] Titratable cyanide terminology is not included in International Cyanide Code.

3. Experimental

3.1. Equipment and Experimental Procedures

Laboratory Leaching Testwork

Lime, sodium cyanide, lead nitrate and air were all of commercial grade and taken from the plant. For the initial testwork four liters of pulp were collected directly from the feed of the first Pachuca leaching tank at the Calcine circuit, after pre-lime, and placed into an small Pachuca Tank with five liters capacity (12,5 cm D x 46 cm H with 60^o conical bottom) as showed in Figure 3.1. The % solids were measured to calculate the solids in the pulp and the amount of cyanide to be added. A weighted amount of cyanide, corresponding to 2.5 kg/t of calcine, was added to the pulp. Samples were taken at 0, 1, 2, 4, 8 and 12 hours. At the beginning and at the end of the tests the solution was analyzed for Au, Zn, Cu, Ni, Fe, Co, As, SO_4^{2-} .Cyanide consumption was measured by the CN_{TOTAL}, CN_{TRIT} and the SCN^- formation. The solids were also analyzed for Au, Zn, Cu, Ni, Fe, Co, As, SO_4^{2-}; in the last sample, it was also analyzed for CN_{TOTAL} to verify if there was any precipitation or adsorption on the solids. The balance was calculated using the cyanide consumed by each complex and compared with the cyanide added in the beginning of the test.

For the second series of laboratory tests, a large sample of calcine (120 kg) was collected in the calcine feeding pipe as pulp, before any lime addition. The sample was filtered, the solution being collected to be used later in the tests. The calcine was dried at low temperature with the sum light outside the building, homogenized and sampled in 3kg samples, using the JIS – Japonese International Standard methodology, to be used in the tests. The leaching tests were carried out in a tank of 5 liters capacity, with 4 baffles to avoid centrifugal effect of the pulp (Figure 3.2). The tank was not covered. An Ingold PH-206 pH monitor provided pH readings. The pHmeter was calibrated on a daily basis. Mixing was provided by an agitator with 10cm paddles powered by a variable-speed electric motor at a constant speed of 450 rpm. The dissolved oxygen was measured by a Digimed DMO-2 oxygen probe. The tests were performed on a pulp of 50% solids. The solution used was collected in the plant, during the initial sampling. The calcine was introduced into the reactor and repulped for a few minutes before starting the test. Lime (CaO) was added during the test, as powder, to maintain a constant pH. When added during pre-lime, lead nitrate was introduced immediately after the pH adjustment. No filtering was done after pre-lime and the leaching used the same pulp. Pre-lime, when applied, continued up to the planed time and no samples were taken during this period.

The leaching time was 8 hours to facilitate the test program and avoid night shift. All planned cyanide dosage was added in the beginning of the test, simulating the plant. Samples of pulp were taken by at times 0, 1, 2, 6 and 8 hours, filtered and the solution sent to be analyzed for Au, Zn, Cu, SO_4^{2-}. Cyanide consumption was measured by the CN_{TOTAL} (only in the end of the test), CN_{FREE} and the SCN^- formation. The solids were also analyzed for Au, Zn, Cu, SO_4^{2-}. The solids were not returned to the reactor. Air was introduced in the bottom of the reactor just below the impeller.

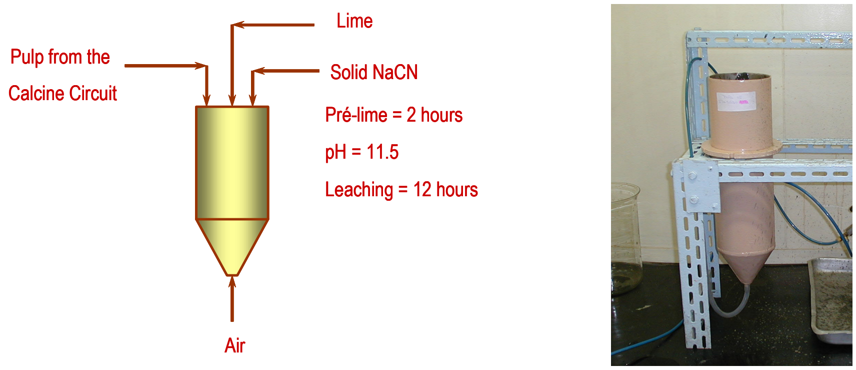


Figure 3.1 - Laboratory Pachuca Tank used to Carry out the Cyanidation Tests.

At the end of the test, the pulp was filtered. After, first solution was taken to be analyzed and the cake was washed with 5 liters of water. The calcine was than dried, sampled and sent to be analyzed by fire assay for gold. The cyanide concentration was not adjusted during the test. Table III.1 shows the phase II laboratory testwork planning. Replicated trials were run, using the standard condition (Table III.2), in order to estimate the magnitude of the experimental error. The tests were carried out in a rondomic order to counteract the effect of uncontrolled variables and systematic error that could affect the responses. The standard conditions were based in the plant operation.

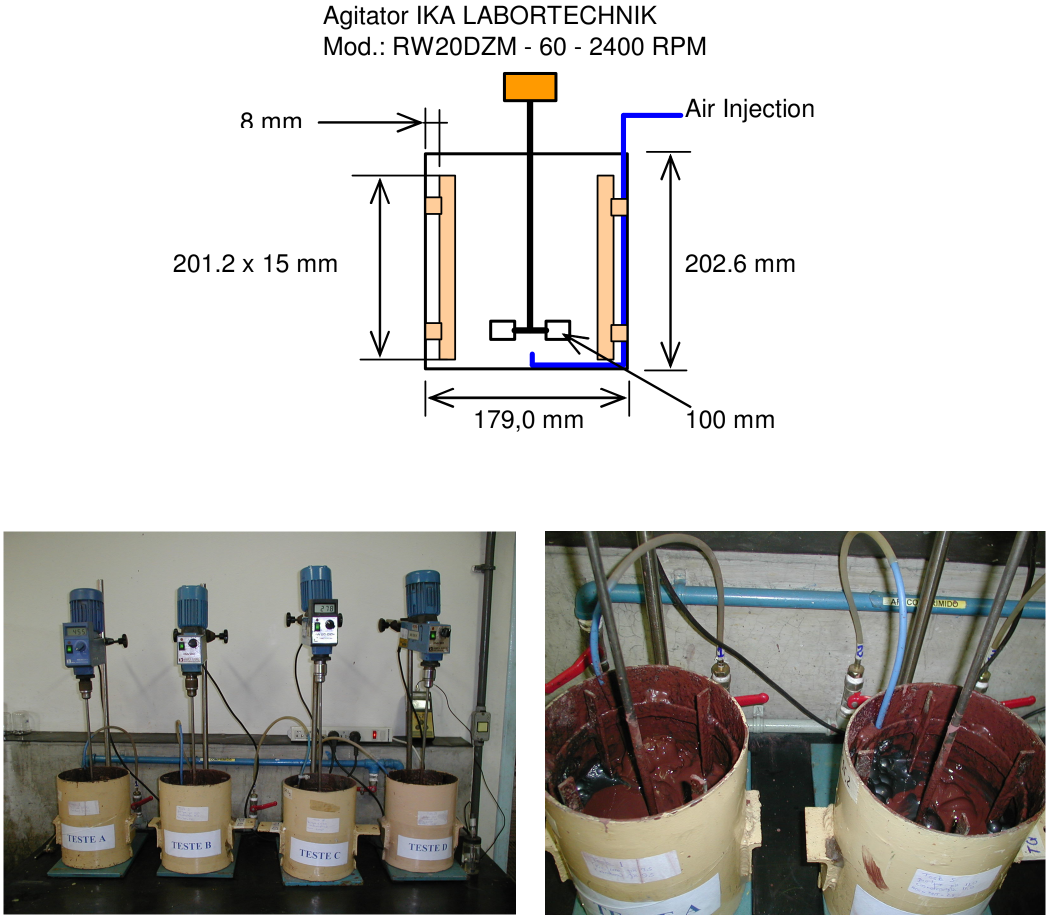


Figure 3.2 – Stirred Tanks used to Carry Out the Second Phase of the Laboratory Tests

Plant Evaluation

The industrial circuit evaluation was carried out with two sampling campaigns at the calcine leaching circuit to confirm some of the laboratory testwork conclusions. Table III.3 shows the sampling plan (points, material and analysis to be performed). Due to the variation of the calcine leaching feed, the two sampling programs were carried out at different days. In the second sampling it was included additional sampling points and analysis to better understand the cyanide balance. Enough amount of pulp was taken in each sampling point to allow for chemical analysis.

Table III.1 - Laboratory Testwork Planning

Test	%	Pre-Lime				Leaching			CIP	
nº	Solids	(h)	pH	Pb(NO$_3$)$_2$	CaO	(h)	pH	NaCN	Carbon	Observation
				(g/t)	(kg/t)			(kg/t)	(g/L)	
LBII- 001	50	2	9,5	-	Det	8	9,5	2,5		
LBII- 002	50	2	10,5	-	Det	8	10,5	2,5		
LBII- 003	50	2	11,0	-	Det	8	11,0	2,5		**Standard Condition**
LBII- 004	50	2	10,5	-	Det	8	11,0	2,5		Like Test LBII-002
LBII- 005	50	2	11,0	-	1.5 x Det	8	11,0	2,5		
LBII- 006	50	2	11,0	-	2 x Det	8	11,0	2,5		
LBII- 007	50	2	11,0	-	3 x Det	8	11,0	2,5		
LBII- 008	50	0	11,0	-	Det	8	11,0	2,5		
LBII- 009	50	2	11,0	-	Det	8	11,0	2,5		**Standard Condition**
LBII- 010	50	6	11,0	-	Det	8	11,0	2,5		
LBII- 011	50	12	11,0	-	Det	8	11,0	2,5		
LBII- 012	50	0	11,0	100	Det	8	11,0	2,5		
LBII- 013	50	2	11,0	100	Det	8	11,0	2,5		
LBII- 014	50	6	11,0	100	Det	8	11,0	2,5		
LBII- 015	50	12	11,0	100	Det	8	11,0	2,5		
LBII- 016	50	2	11,0	-	Det	8	11,0	1,5		
LBII- 017	50	2	11,0	-	Det	8	11,0	2,5		**Standard Condition**
LBII- 018	50	2	11,0	-	Det	8	11,0	5,0		
LBII- 019	50	2	11,0	100	Det	8	11,0	1,5		
LBII- 020	50	2	11,0	100	Det	8	11,0	2,5		Like Test LBII-013
LBII- 021	50	2	11,0	100	Det	8	11,0	5,0		
LBII- 022	50	2	11,0	5	Det	8	11,0	2,5		
LBII- 023	50	2	11,0	25	Det	8	11,0	2,5		
LBII- 024	50	2	11,0	50	Det	8	11,0	2,5		
LBII- 025	50	2	11,0	100	Det	8	11,0	2,5		Like Test LBII-013
LBII- 026	50	2	11,0	200	Det	8	11,0	2,5		
LBII- 027	50	2	11,0	-	Det	8	11,0	2,5	6	24 g of carbon
LBII- 028	50	2	11,0	-	Det	8	11,0	2,5	6	24 g of carbon
LBII- 029	50	2	11,0	-	Det	8	11,0	2,5		Deionozided water
LBII- 030	50	2	11,0	-	Det	8	11,0	2,5		Deionozided water

Det = Amount of lime necessary to increase the pH to 11 or to the test pH

Table III.2 - Plant Operation Conditions used in the Laboratory Tests.

Description	Conditions
Pre-lime contact time	2 hours
Pre-lime pH	11.5
NaCN addition	2.5 kg/t
% Solids	50%
$Pb(NO_3)_2$ addition	no addition

Table III.3 - Sampling Plan for Industrial Circuit Evaluation

Point	Name	Material	Sampling
1	Pre-lime Feed	Solid and Solution	▪ Au, Cu, S, Fe, Ni, Co, Ca, As, Zn and SO_4^- analysis ▪ Pulp flowrate and % solids
2	Leaching Tank Feed	Solid and Solution	▪ Au, Cu, S, Fe, Ni, Co, Ca, As, Zn, and SO_4^- analysis ▪ Pulp flowrate and % solids
3	Discharge of each Pachuca Tank	Solid and Solution	▪ Au, Cu, S, Fe, Ni, Co, Ca, As, Zn, and SO_4^- analysis ▪ % solids ▪ Ions and CN_{TOTAL} in solids only in the last Pachuca Tank ▪ pH, O_2 dissolved, CN_{TOTAL}, CN_{FREE}, SCN^- ,NH_3, CNO^- (test IT-02) ▪ NaCN addition points ▪ NaCN concentration in the solution added into the Pachuca Tanks

The samples at the calcine leaching circuit were taken at the outlet of each Pachuca tank. It is always difficult to take a representative sample in this type of circuit, especially solid samples. To avoid this problem, samples were composed during one hour period, with each sub-sample taken at each 15 min. CN_{FREE} was analyzed immediately after sampling. The samples were filtrated and the solution was preserved according to the chemical laboratory procedure for each type of analysis. Solids were dried, sampled and sent to gold analysis.

Table III.4 and III.5 shows the procedures used to characterize the calcine and the solution during the tests.

Table III.4 - Analytical Procedures for Solid Analyses

Analysis/Characterization	Methodology
Size Distribution	The material is screened, using water, at 65, 100, 150, 200, 270, 325, 400 mesh Tyler. The material below 400 mesh (38 μm) is collected, dried and sent to cyclosizer analysis.
Gold Analysis	Fire-assay
Chemical Analysis	Cu, Fe, Ni, Co, Ca, Zn are analyzed by acid total digestion and ICP-Plasma
Total Cyanide (CN_{TOTAL})	The ALPHA described in the item 3.12
Sulfur Content	Determined using a LECO equipment

Table III.5 - Analytical Procedures for Solution Analysis

Analysis/Characterization	Methodology
Gold Analysis	Atomic Absorption Spectrophotometer– A.A. (equipment)
Chemical Analysis	Cu, Fe, Ni, Co, Ca, Zn were analyzed by ICP-Plasma (equipment)
Total Cyanide (CN_{TOTAL})	The procedure RO-O-GECA-001-rev 0 (internal MMV procedure) was used. 500 mL of solution, containing no more than 10 mg/L CN^-, is added in a boiling flask. Cadmium nitrate and Cloridrato de hidroxilamina /HCl are added. The sample is distillated for 45 min, with one air bubble/s passing through the samples. The air carries the HCN gas from the flask to the absorber. After 45 min the heating is interrupted but continuous air flow is allowed for 15 min more. The NaOH solution (absorber) is analyzed by cyanide-selective electrode.
WAD cyanide (CN_{WAD})	It is the same procedure described above changing the acid for weak acid (pH 4.5 to 6.0).Cyanide is analyzed by a cyanide selective electrode (sample with pH >12,0)
Titratable Cyanide (CN_{TRIT})	It is analyzed using a conventional titration with silver nitrate ($AgNO_3$) with p-dimethylaminobenzal-rhodamine as indicator
Cyanate (CNO^-)	The procedure is the 4500-CN- L. from the standard methods (ALPHA)
Thiocyanide (SCN^-)	The method used in this work is titration. Because free cyanide competes for silver, each sample is first acidified with 0,5 mol/L H_2SO_4 and sparged with nitrogen gas or air for 30 min to drive off free cyanide as HCN gas. The sample is then re-acidified with 6 mol/L HNO_3, and 2 mL of ferric nitrate and then filtrated to remove any ferrocyanide precipitated. Then 2 mL of ferric nitrate (Fe^{3+} and SCN^- form an intense red color) is added as indicator just prior titration.

4. Results and Discussion

The data presented in Table IV.1 are intended to provide general information about the ore and the flotation (pyrite) concentrate composition. The calcine composition used in the tests varied depending on the mixing ratio of the graphite concentrate with the calcine.

Ore and Pyrite Concentrate

It is a very difficult task to take representative samples from the ore, the flotation concentrate and flotation tailings due to variation of the fed ore in the plant and the delay between feed and concentrate generation (Table IV.1).

Table IV.1 - Chemical Analysis of the Ore, Flotation Concentrate and Tailings

Sample	As (%)	Fe_2O_3 (%)	MgO (%)	Al_2O_3 (%)	SiO_2 (%)	CaO (%)	MnO (%)
Ore High grade	0.21	11.59	2.25	4.38	47.3	2.77	0.29
Ore medium grade	0.16	7.83	2.34	5.50	48.2	5.78	0.28
Ore low grade	0.16	12.64	2.34	7.27	47.2	6.02	0.31
Composed Concentrate	0.81	10.04	0.84	3.98	14.2	2.81	0.12
Flotation Tailings	0.26	9.51	2.33	5.67	57.16	4.32	0.32

Sample	Na_2O (%)	K_2O (%)	SO_4^{2-} (%)	CO_3^{2-} (%)	$C_{graphite}$ (%)	Ctotal (%)	Stotal (%)
Ore High grade	0.75	0.64	0.78	12.05	0.48	2.89	7.36
Ore medium grade	1.00	0.64	0.64	13.99	0.57	3.37	6.70
Ore low grade	1.00	0.90	0.80	13.34	0.62	3.29	6.16
Composed Concentrate	0.63	0.80	0.79	5.60	1.50	2.62	29.4
Flotation Tailings	1.10	0.72	0.36	16.05	0.32	7.33	0.91

Sample	Cr (g/t)	Co (g/t)	Cu (g/t)	Zn (g/t)	Cd (g/t)	Ni (g/t)	Pb (g/t)
Ore High grade	142	22.1	90	377	110	74	36
Ore medium grade	140	21.9	90	402	87	78	30
Ore low grade	151	22.4	98	437	73	83	36
Composed Concentrate	239	73.0	501	1802	317	255	123
Flotation Tailings	117	9.22	26.1	105	502	30.2	18

Sample	FeAsS (%)	FeS (%)	FeS_2 (%)
Ore High grade	0.27	2.21	11.63
Ore medium grade	0.19	2.47	10.31
Ore low grade	0.19	1.16	10.16
Concentrate	0.97	11.02	46.47
Tailings	0.24	1.43	0.42

Obs: Mineralogy based on chemical analysis

Table IV.1 shows three different analysis of the ore feeding the plant in different days, one typical flotation concentrate and a typical flotation tailings. The main purpose was to show the origin of the metallic ions that consume cyanide in the leaching circuit. Only copper ions are added to the flotation circuit as copper sulfate ($CuSO_4$), as activator of the pyrite. Considering the numbers above it is clear that the enrichment of the concentrate involves basically the sulfides, pyrite, pyrrhotite and arsenopyrite.

Calcine

Table IV.2 and Figure 4.1 show the calcine particle size distribution carried out by wet screening and cyclosizer analysis. One can see that the calcine is a very fine material, with 80% below 40 µm. Table IV.3 shows the chemical analysis of the calcine used in the laboratory tests. Zinc, cobalt, copper appear as the main cyanide consumers present in the sample. Residual pyrrhotite may also contribute to the formation of ferrous cyanide and thiocyanide.

Additional analyses were carried by Duarte (2004). The complete report is shown in Appendix 1. Table IV.4 to IV.6 summarize the main results. The chemical analysis shown in the Table IV.5 confirms that iron compounds and silica are the major constituents of the calcine.

4.1. Cyanide Balance and Speciation

The initial testwork was carried out with the following objectives:

(i) To validate the cyanide analysis methodology
(ii) To establish the cyanide complexes formed during leaching
(iii) To establish a trustable cyanide balance

For these tests the pulp was taken directly from the leaching circuit feed and placed in a small Pachuca Tank as showed in Figure 3.1. The procedure followed was described in the chapter 3. Table IV.7 summarizes the test conditions.

Table IV.2 - Calcine Size Distribution

Aperture (μm)	% Passing
38.0	78.8
29.7	76.4
20.5	69.2
14.8	58.6
10.1	47.7
8.1	40.7

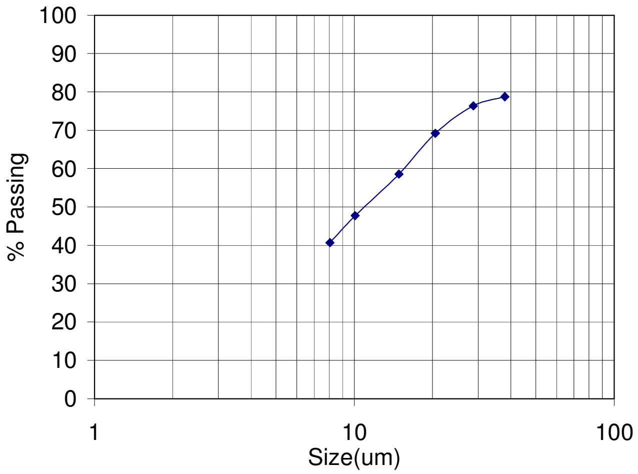


Figure 4.1 - Calcine Size Distribution curve

Table IV.3 – Calcine Chemical Analysis used in the Laboratory Tests

Sample	Au (g/t)	Fe (%)	FeO (%)	Fe_2O_3 (%)	MgO (%)	Al_2O_3 (%)	SiO_2 (%)	CaO (%)
Calcine	33.37	49	0.36	69.0	0.75	6.95	18.78	2.1

Sample	Na_2O (%)	K_2O (%)	SO_4^{2-} (%)	CO_3^{2-} (%)	$C_{graphite}$ (%)	Ctotal (%)	MnO (%)	Stotal (%)
Calcine	1.33	1.42	0.65	1.08	0.40	0.62	0.12	4.79

Sample	Cr (g/t)	Co (g/t)	Cu (g/t)	Zn (g/t)	Cd (g/t)	Ni (g/t)	As (%)	Pb (g/t)
Calcine	314	859	808	2084	443	422	0.85	104

Sample	FeAsS (%)	FeS (%)	FeS_2 (%)
Calcine(*)	1.43	2.65	6.24

(*) Presence of sulfides due to blending with fines from the flotation concentrate

Table IV.4 – Calcine Specific Gravity and Specific Superfice Area

Parameter	Results
Specific Gravity	3.85
BET Surface Area	4.4 m^2/g

Table IV.5 – Elemental Chemical Composition Determined by EDS (Duarte, 2004)

Element	% mass
Oxygen	35.2
Magnesium	2.1
Aluminum	6.4
Silica	13.7
Sulfur	5.6
Chlorine	0.4
Potassium	1.6
Calcium	1.6
Titanium	0.4
Iron	32.9

Table IV.6 – Crystalline Phases in the Calcine Determined by X-Ray Diffraction (Duarte, 2004)

Crystalline Phase	Chemical Formula
Hematite	Fe_2O_3
Quartz	SiO_2
Muscovite	$KAl_2(Si_3Al)O_{10}(OH)_2$
Clinochlore	$(Mg,Fe)_6(Si,Al)_4O_{10}(OH)_8$
Gypsum	$CaSO_4.2H_2O$
Titanium Oxide	TiO_2

Table IV.7 – Experimental Conditions used in the Initial Laboratory Test

Description	Condition
Calcine, kg	2.266
% de solids	40.3%
pH	11.0
Solution Volume, mL	3,354
Calcine S.G.	3.51
Volume of Pulp, L	4.0
Pulp Density, kg/L	1.405
NaCN added, g/t	2,500
NaCN added, g	5.7
Initial NaCN concentration, mg/L	1,689
Initial CN^- concentration, mg/L	896
Initial CN^- concentration, mg/L @ 50% Solids	1,326

Tables IV.8 and IV.9 show the results from samples taken directly from the pulp. To facilitate the results interpretation, these data were recalculated to a fixed pulp density using eq. (28), in a way that results obtained in different pulp densities could be compared. Table IV.10 shows the results transformed to 50% solid basis.

$$C_{METAL}^{50} = \left(\frac{\left[C_{METAL} \right]_S . V_S}{W_{SOLIDS}} \right) \tag{28}$$

C^{5O}_{METAL}	Metal concentration in solution converted to 50% solids (as g/t for Au or % w/w for the other metals)
$[C_{METAL}]_S$	Metal concentration in solution, at the test % solids (mg/L)
V_S	Solution volume (L)
W_{SOLIDS}	Weight of solids in the pulp (kg)

The methodology used to calculate cyanide speciation is based on the stability constants of the cyanide complexes (Adam, 2000) and Eh x pH diagrams (Xue *et al.*, 1984). It was assumed that:

(i) Sampling and sample preparation have been carried out to ensure stabilization
(ii) Accurate cyanide assays have been achieved, taking any limitation in account
(iii) Metals in solution have been assayed
(iv) Solution pH and Eh have been measured

Table IV.8 - Solution Composition in the Initial Laboratory Test

Time (h)	pH	Au (mg/L)	Zn (mg/L)	Cu (mg/L)	Ni (mg/L)	Fe (mg/L)	Co (mg/L)	As (mg/L)
0	11.7	0.0	0.01	0.08	0.04	0.03	0.01	1.47
1	11.6	10.1	94	83.5	0.4	0.3	0.1	0.1
2	11.4	12.8	108	94.6	0.7	0.7	0.2	0.6
4	11.3	15.6	125	107	1.0	1.5	0.2	0.5
8	11.3	16.9	159	130	2.0	3.6	0.4	0.8
12	11.3	17.0	168	140	2.8	4.6	0.4	1.7

Time (h)	SO_4^{-2} (mg/L)	CN_{TOTAL} (mg/L)	CN_{TRIT} (mg/L)	SCN (mg/L)
0	2058	0	0	0
1	2474	927	833	206
2	2641	923	829	245
4	2992	842	742	270
8	3512	680	629	339
12	3636	629	565	404

Table IV.9 - Calcine Composition in the Laboratory Test

Material	Au (g/t)	Zn (g/t)	Cu (g/t)	Ni (g/t)	Fe (%)	Co (g/t)	As (g/t)	S (%)	CN_{TOTAL} (g/t)
Feed	25.77	2568	674	377	35.79	94	8648	7.58	
Tailing	3.83	2469	600	393	36.45	100	8744	7.21	3.16

Table IV.10 - Recalculated Solution Composition to 50% Solids

Time (h)	pH	Au (mg/L)	Zn (mg/L)	Cu (mg/L)	Ni (mg/L)	Fe (mg/L)	Co (mg/L)	As (mg/L)
0	11.7	0.0	0.0	0.1	0.1	0.0	0.0	2.2
1	11.6	14.9	139	123	0.7	0.5	0.2	0.2
2	11.4	18.9	159	140	1.0	1.1	0.3	0.9
4	11.3	23.1	185	158	1.5	2.2	0.4	0.7
8	11.3	25.0	235	192	3.0	5.3	0.5	1.2
12	11.3	25.2	248	207	4.1	6.9	0.6	2.5

Time (hours)	SO_4^{-2} (mg/L)	CN_{TOTAL} (mg/L)	CN_{TRIT} (mg/L)	SCN (mg/L)
0	3046			0.0
1	3661	1372	1233	305
2	3909	1366	1227	362
4	4428	1246	1098	399
8	5198	1006	931	502
12	5382	931	836	598

Table IV.11 shows the stability constant of the cyanide complexes (after Busey, 1965; Hancock et al., 1972; Persson, 1971, 1974). Eh-pH diagrams are presented in the Figures 4.2 to 4.5. Though the diagrams do not show all metal complexes, the windows of the most stable complexes are very clearly indicated. Even with the diagrams showing different ions concentration from the tests, ([CN] = 10^{-3} mol/L (26 mg/L); [Cu] = 6.35 mg/L; [Fe] = 5.6 mg/L; [S] = 3.2 mg/L), the stability regions windows are clear. Based on the chemical analysis, on the Eh x pH diagrams and the stability constants the complexes present in solution were considered to be: $Fe(CN)_6^{4-}$, $Au(CN)_2^-$, $Cu(CN)_3^{2-}$, $Co(CN)_6^{3-}$, $Zn(CN)_4^{2-}$, $Ni(CN)_4^{2-}$, SCN^-. The HCN losses and CNO^- concentration was not measured at this stage.

Once chosen the complexes, the amount of CN^- complexed was calculated based on the metallic ions concentration and molar ratio. Table IV.12 shows the results.

Table IV.11 - Stability Constants of Cyanide Complexes (Adam, 2000)

Species	Lg β_n	Species	Lg β_n
$Co(CN)_6^{3-}$	64.0	$Zn(CN)_4^{2-}$	21.57
$Fe(CN)_6^{3-}$	43.9	$Zn(CN)_3^-$	16.88
$Fe(CN)_6^{4-}$	36.9	$Zn(CN)_2$	11.02
$Au(CN)_2^-$	36.6	$ZnCN^+$	5.34
$Cu(CN)_4^{3-}$	27.9	$Ni(CN)_4^{2-}$	31.1
$Cu(CN)_3^{2-}$	26.8	$Ni(CN)_3^-$	22.0
$Cu(CN)_2^-$	21.7	$Ni(CN)_2$	14.0
$Ag(CN)_3^{2-}$	21.8	$NiCN^+$	7.0
$Ag(CN)_2^-$	20.9		

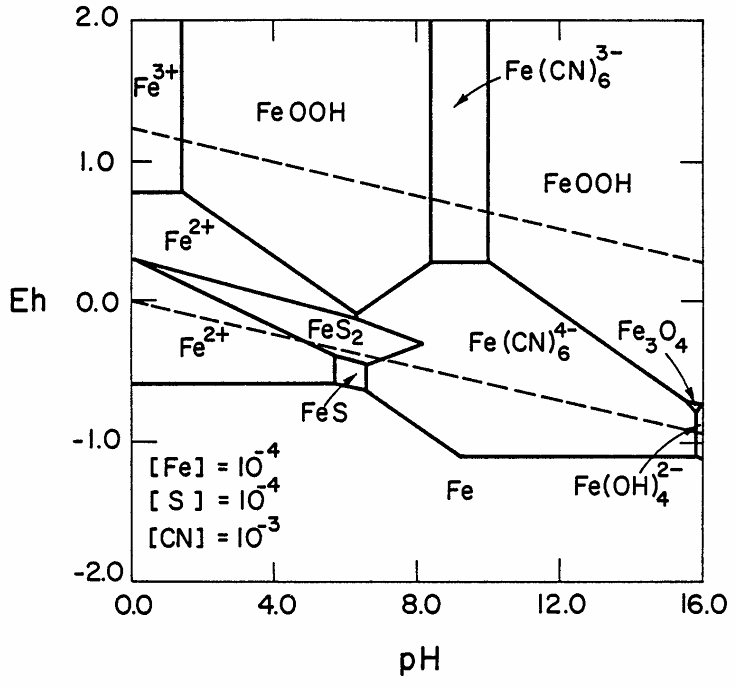


Figure 4.2 - Eh-pH Diagram for the System Fe-S-CN-H_2O at 25°C (Xue *et al.*, 1984).

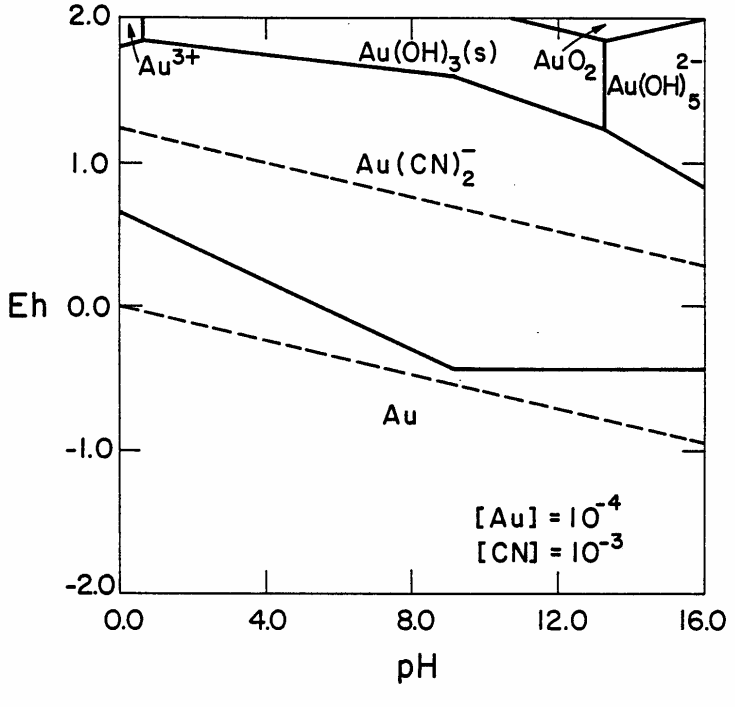


Figure 4.3 - Eh-pH Diagram for the System Au-CN-H_2O at 25°C (Xue *et al.*, 1984).

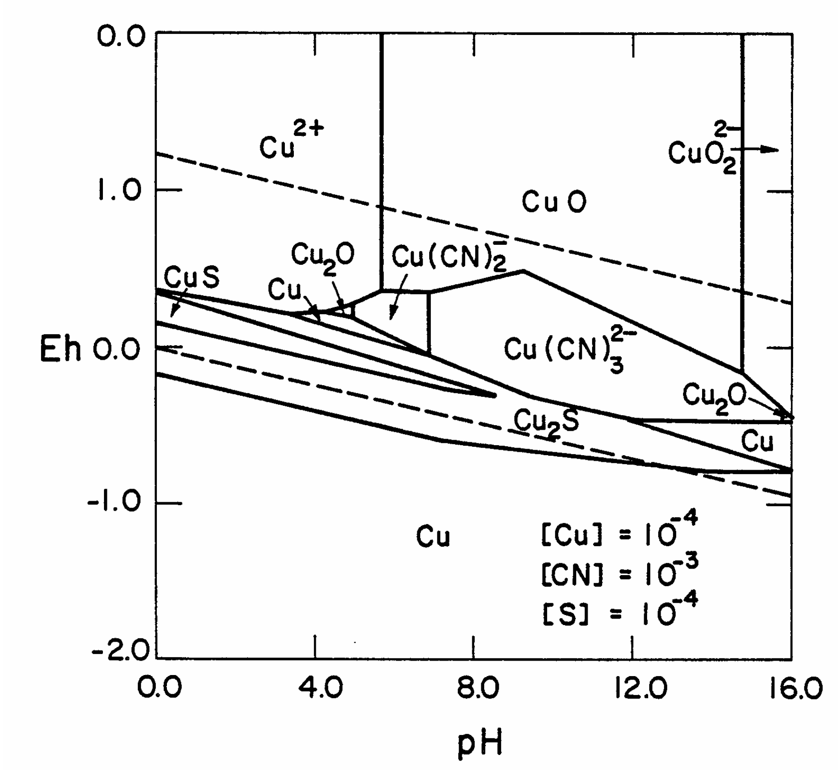


Figure 4.4 - Eh-pH Diagram for the System Cu-S-CN-H_2O at 25°C (Xue *et al.*, 1984)

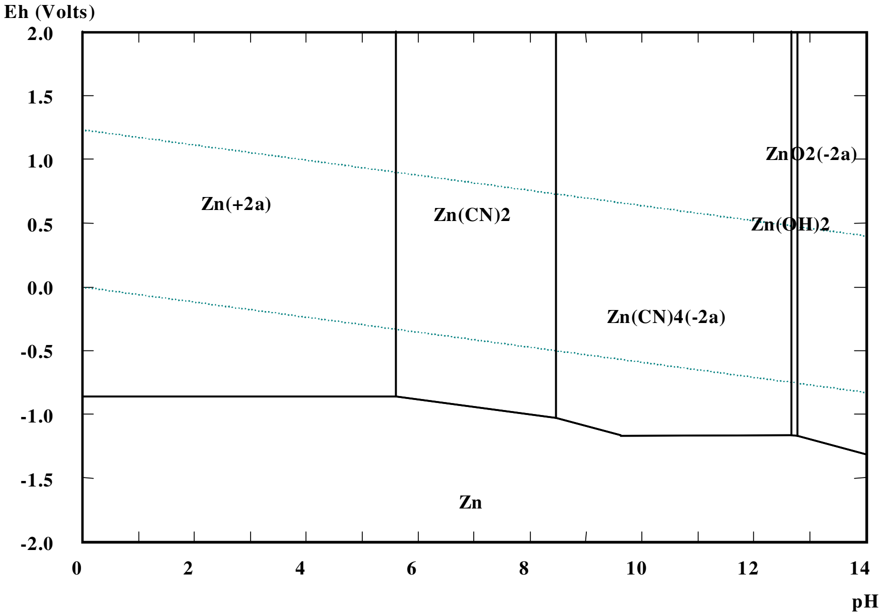


Figure 4.5 – Eh-pH Diagram for the System Zn-CN-H_2O, at 25°C. Concentration (mol/kg H_2O): Zn, 0,0006; CN^-, 0,005 (Ciminelli *et al.*, 2004).

Figure 4.6 and 4.7 illustrate the results shown in Table IV.12. From these results one can conclude:

- The biggest CN consumers are Cu, Zn and SCN with 31.9; 31 and 21% of the CN_{TOTAL} calculated.
- The average difference between the CN^- added and the calculated CN^- is -5.76% which can be considered good enough for the purpose of this work. If it is considered only the last determination (12 hours), which is the one normally used in the plant, the difference drops to -3.9%.
- The average difference between the CN^- added and the analyzed CN^- is -10.71% which is also adequate for the propose of this work. But if it is considered the last two measurements, the difference is superior to 24%.
- The calculated cyanide balance based on metal concentrations and metal cyano-complex stoichiometry was shown to be more reliable than CN_{TOTAL} determination which includes acidification and volatilization plus CN^- associated with SCN^-. The reason for this difference needs to be better investigated.

Gold Recovery Calculation

The feed gold grade was calculated based in a mass balance, which took into account gold concentration in solution (final sample), corrected to solids concentration, and the gold tailings grade as shown in eq. (29). Gold recovery was calculated based on the gold concentration in solution, corrected to solids concentration, divided by the recalculated feed grade as shown in eq. (30).

$$C_S^0 = \left(\frac{[Au]_S . V_S}{W_{SOLIDS}} \right) + C_S^T \qquad (29)$$

C_S^O	Gold concentration in the ore feed (g/t)
$[Au]_S$	Gold concentration in solution at the final sample (mg/L)
V_S	Solution volume (L)
W_{SOLIDS}	Weight of solids in the pulp (kg)
$C^T{}_S$	Gold concentration in the tailing (g/t)

Table IV.12 – Speciation Based on Metal Concentration

Time (h)		1	2	4	8	12
CN_{TRIT}	CN^- (mg/L)	833.0	829.0	742.0	629.0	565.0
CN_{FREE}[1]	CN^- (mg/L)	611.7	574.8	447.8	254.8	169.6
Cu	(mg/L)	123.6	140.0	158.4	192.4	207.2
$Cu(CN)_3^{2-}$	CN^- (mg/L)	202.4	229.3	259.4	315.1	339.4
Zn	(mg/L)	139.1	159.9	185.0	235.3	248.7
$Zn(CN)_4^{2-}$	CN^- (mg/L)	221.3	254.2	294.2	374.2	395.4
Fe	(mg/L)	0.5	1.1	2.2	5.3	6.9
$Fe(CN)_6^{4-}$	CN^- (mg/L)	1.4	3.1	6.1	14.8	19.1
Ni	(mg/L)	0.7	1.0	1.5	3.0	4.1
$Ni(CN)_4^{2-}$	CN^- (mg/L)	1.1	1.7	2.7	5.2	7.2
Co	(mg/L)	0.2	0.3	0.4	0.5	0.6
$Co(CN)_6^{3-}$	CN^- (mg/L)	0.5	0.8	0.9	1.4	1.5
Au	(mg/L)	14.9	18.9	23.1	25.0	25.2
$Au(CN)_2^-$	CN^- (mg/L)	3.9	5.0	6.1	6.6	6.6
Complexed CN	CN^- (mg/L)	1042.5	1069.0	1017.2	972.2	938.9
SCN-	(mg/L)	304.9	362.6	399.6	501.8	598.0
SCN-	CN^- (mg/L)	136.7	162.6	179.2	224.9	268.1
TOTAL CN calc	CN^- (mg/L)	1179.1	1231.5	1196.4	1197.1	1206.9
CN_{TOTAL} analyzed	CN^- (mg/L)	1372.1	1366.2	1246.3	1006.5	931.0
CN^- Added	CN^- (mg/L)	1326.5				
Difference between the CN added and CN_{TOTAL} Calculated and Analyzed						
from the Calculated CN_{TOTAL}		-11.11%	-7.16%	-9.81%	-9.76%	-9.02%
from the Analyzed CN_{TOTAL}[2]		3.44%	2.99%	-6.05%	-24.12%	-29.82%

[1] – $CN_{FREE} = CN_{TRIT}$ – CN associated with Zn

[2] – Analyzed $CN_{TOTAL} = CN_{TOTAL} + CN^-$ from SCN^-

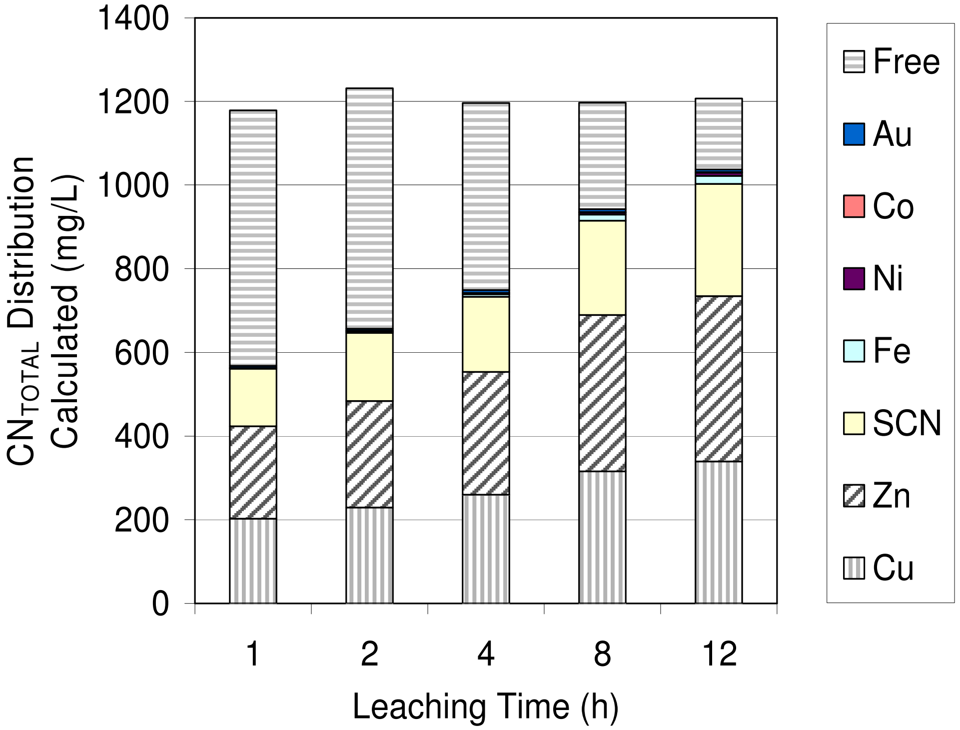


Figure 4.6 – Cyanide Balance per Sampling Time in the Laboratory Test – 50% solids, pH 11, NaCN added 2500 g/t Corresponding to 1326 mg/L in the Pulp.
OBS: CN^- associated with Au, Co, Fe is so small that is not evident in the graph

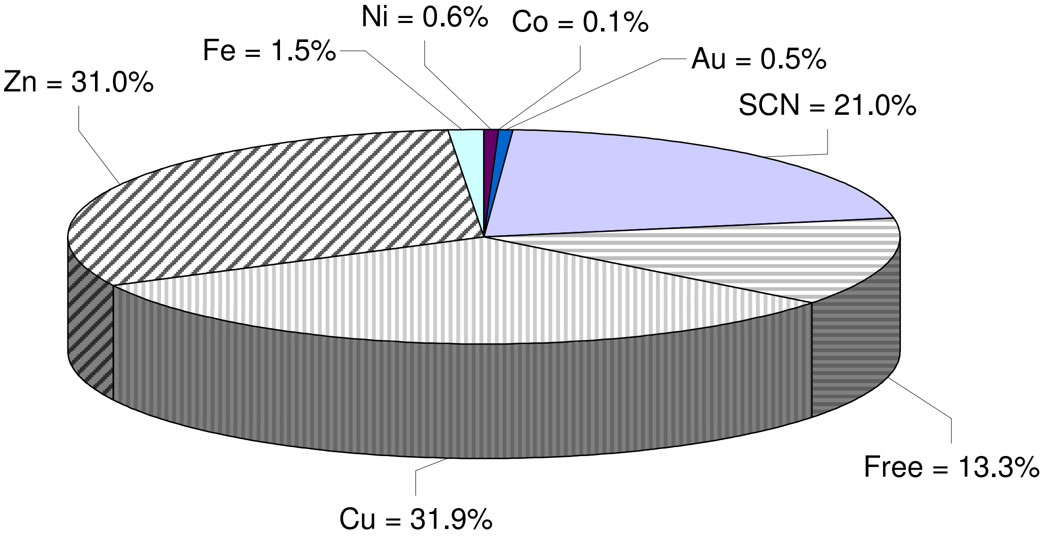


Figure 4.7 – Cyanide Speciation at the end of the Laboratory Testwork

$$R_{AU} = 100\frac{\left(\frac{[Au]_S.V_S}{W_{SOLIDS}}\right)}{C_S^0} \qquad (30)$$

R_{AU} Gold recovery (%)

The calculated feed grade was the compared with the analyzed feed grade in order to determine possible sampling errors. Table IV.13 shows the gold recovery results. The 11.1% difference between recalculated and analyzed feed was considered acceptable for this high gold grade sample, due to the so called "nugget effect" or the effect of a free gold particle in one of the samples.

Table IV.13 - Gold Balance - Analyzed and Fitted for the Lab Test

Leaching Time (h)	Au Solution (mg/L) @ 50% solids	Gold Recovery (%)	Gold Recovery Fitted Curve (%)
0	0.0	0.0	0.0
1	14.9	51.6%	56.9%
2	18.9	65.3%	70.2%
4	23.1	79.6%	79.5%
8	25.0	86.3%	85.1%
12	25.2	86.8%	87.2%
Recalculated Feed		28.99 g/t	
Analyzed Feed		25.77 g/t	
Difference		11.1 %	

As described in the item 3.4 the kinetics curve was also fitted to the analyzed numbers using eq. (9). Figure 4.8 shows the fitted curve to the experimental values.

The defined equation is showed at eq. (31).

$$\frac{1}{(C_S - 2.43)} = 0.0621t + 0.0378 \qquad (31)$$

Where:

C_S Gold concentration in the solids at time t (g/t)

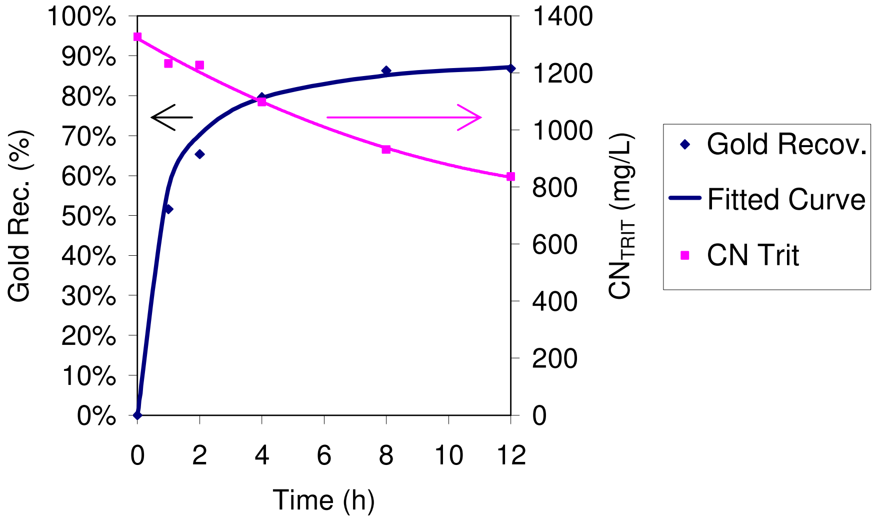


Figure 4.8 - Gold Recovery Curve and CN$_{TRIT}$ Concentration in the Laboratory Test – 50% solids, pH=11.5, 2.5 kg/t of NaCN

4.2. Plant Evaluation

Once the calculated cyanide balance and gold calculation was considered reliable, a plant evaluation sampling campaign was carried out. The procedure followed was described in a previous chapter. Figure 4.9 to 4.12 show the pre-lime Pachuca Tank, the leaching Pachuca Tanks, the first leaching Pachuca Tank and samples taken from the circuit, respectively.

4.2.1. Cyanide Losses

Cyanide can form complex with metals and thiosulfate but also could be lost, firstly as hydrogen cyanide or hydrocyanic acid (HCN) and secondly by oxidation to cyanate ($CNO_{(aq)}$). Figure 4.13 shows the Eh x pH diagram showing the CNO^- stability. It is very possible that part of the CN^- could be oxidized. However, this diagram is based on thermodynamic data and does not consider the oxidation kinetics. HCN(aq) is extremely soluble in aqueous solution and stay in equilibrium with HCN(gas) in solution (Adams, 1994). The HCN(gas) thus formed will volatilize to some extent.

Figure 4.9 - Pre-lime Pachuca Tank at the Calcine Leaching Circuit

Figure 4.10 - Leaching Pachuca Tanks

Figure 4.11 - First Leaching Pachuca Tank where Cyanide is Added

Figure 4.12 - Samples Taken from the Circuit

In order to calculate the possible losses of cyanide as HCN, eq. (32) was applied. This equation provides an estimation of HCN emission from the individual process tanks in CIP/CIL circuits. The calcine leaching circuit uses Pachuca Tank and not agitated tanks as in the CIP/CIL circuits, but it was considered an approximation of the Queiróz circuit (NPI – National Pollutant Inventory – Emission Estimation Technique Manual for Gold Ore Processing – Version 1.1 – Australia, 2001).

$$E = \left(\left[0.013 * \left[HCN_{(aq)} \right] + 0.46 \right] * A * \left(\frac{T}{10^6} \right) \right) * 1000 \tag{32}$$

where:

E	Emission of HCN (kg)
$[HCN_{(aq)}]$	$[NaCN]*\ 10^{(9.2-pH)}$
$[NaCN]$	Concentration (as mg/L) of NaCN in the leach/adsorption tank
pH	pH in the leach/adsorption tank
A	Surface area (m^2) of the leach/adsorption tank
T	Period of emission (hours)

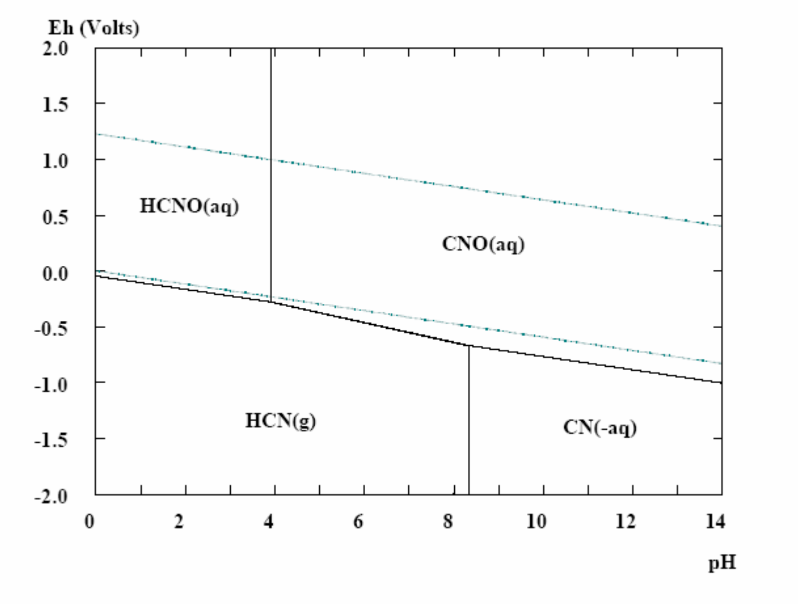


Figure 4.13 - Eh - pH Diagram for CN-H_2O System at 25°C

Table IV.14 shows results of one sampling campaign done in the calcine circuit to estimate the HCN losses. The top area of each Pachuca Tanks was 9.08 m^2.

Table IV.14 - HCN Losses Calculation from the Pachuca Leaching Tanks

Pachuca	Pulp	% Solids	Contact	NaCN	pH	$HCN_{(aq)}$	Emission		
	(m^3/h)		(h)	(mg/L)		(mg/L)	HCN (g/h)	CN (g/h)	NaCN (g/h)
Pachuca 1	28.1	43.9%	2.18	493	11.1	6.21	4.91	4.73	8.91
Pachuca 2	28.1	44.4%	2.18	437	10.9	8.72	5.21	5.01	9.45
Pachuca 3	28.1	43.6%	2.18	357	10.9	7.12	5.02	4.83	9.11
Pachuca 4	28.1	44.2%	2.18	327	11.5	1.64	4.37	4.21	7.93
Pachuca 5	28.1	44.1%	2.18	276	11.3	2.19	4.44	4.27	8.05
Pachuca 6	28.1	42.3%	2.18	250	11.2	2.50	4.47	4.31	8.11
							g/h	27.36	51.56
							kg/day	0.657	1.237
						% of the cyanide added			0.22%

A research carried out by CSIRO indicated that 1% of total cyanide could be lost through volatilization as HCN within the processing area of a typical CIP/CIL circuit (Heath *et al.*,1998). Based on this calculation, the losses in the calcine leaching circuit were estimated to be around 0.22%, as showed in the Table IV.14. It is much lower than the estimation made by Heath. However, in Australia the leaching conditions are quite different from the Queiróz circuit, as they use pH around 9 and high ionic strength due to water salinity, both conditions favoring volatilization. However, one can see that in the eq. (32) there are no factors or terms in the model taking into account the influence of the ionic strength due to the water salinity, and the influence of the air flowrate in the tanks.

Due to the results shown in the Table IV.14, the possibility of HCN losses was discarded and the CNO^- and NH_3 analysis were included in the test sampling program in order to improve the accuracy of the cyanide balance.

4.2.2. Plant Data

This test followed the experimental procedure discussed in chapter 3. Table IV.15 shows the plant operation conditions in the plant during the sampling campaign.

Table IV.15 - Plant Operation Condition

Description	Condition
Calcine Flowrate, t/h	17.8
% de solids	43.1%
pH	11.5
Solution Volume, m^3/h	23.5
Calcine S.G.	3.51
Volume each Leaching Pachuca Tank, m^3	80.0
Pulp Density, kg/L	1.446
NaCN added, g/t	1,403
NaCN solution flowrate, m^3/h	0.25
NaCN added, kg/h	25.0
Initial NaCN concentration, mg/L	913
Initial CN^- concentration, mg/L	485
Initial CN^- concentration, mg/L @ 50% Solids	744

It can be seen from the results, showed in the Table IV.16, that the cyanide complexed with metals had a good balance with the added cyanide. The difference between the added cyanide in the pulp and the calculated cyanide ranged from +5.9% to 4.7% with an average of 3.87%. Similar to the laboratory tests, the difference between the added cyanide and the CN_{TOTAL} analyzed was not good, increasing from -7.2% to -28.4% with a maximum of -33.8% and average of 25.0%. The reasons for this discrepancy are not fully understood at this point. Figure 4.14 shows the good mass balance of this test using the cyanide complexed calculation. Copper, zinc and thiocyanide were again the biggest cyanide consumers. The cyanide oxidation to CNO^- and NH_3 were very low, as expected. One can conclude from Figures 4.14 and 4.15 that:

(i) All metal complexes are formed in the first Pachuca Tanks

(ii) The SCN^- concentration slowly increases the from the first (402 mg/L) to the last Pachuca Tank (538 mg/L)

(iii) The SCN^- was the biggest CN consumer, 30.9%; followed by Zn and Cu with 25 and 27%, respectively, as the main consumers. Similar conclusion was reached from the results form the lab test.

Table IV.16 - Solution Analysis and Cyanide Balance of the Plant Evaluation

Pachuca		1	2	3	4	5
Contact Time (h)		2.9	5.8	8.7	11.6	.14.5
CN_{TRIT}	CN^- (mg/L)	364.1	294.1	246.5	235.2	249.0
CN_{FREE}	CN^- (mg/L)	149.2	103.0	57.5	40.2	50.0
CNO^-	(mg/L)	28.5	63.7	74.2	84.7	79.8
CNO^-	CN^- (mg/L)	17.6	39.5	45.9	52.4	49.4
Cu	(mg/L)	126.3	141.4	137.6	137.6	127.6
$Cu(CN)_3^{2-}$	CN^- (mg/L)	206.9	231.6	225.4	225.4	209.0
Zn	(mg/L)	135.1	120.1	118.9	122.6	125.1
$Zn(CN)_4^{2-}$	CN^- (mg/L)	214.9	191.0	189.0	195.0	199.0
Fe	(mg/L)	1.9	2.6	2.4	2.7	2.9
$Fe(CN)_6^{4-}$	CN^- (mg/L)	5.5	7.2	6.7	7.6	8.2
Ni	(mg/L)	3.6	5.7	5.2	6.8	7.1
$Ni(CN)_4^{2-}$	CN^- (mg/L)	6.3	10.0	9.1	11.9	12.5
Co	(mg/L)	0.4	0.5	0.4	0.5	0.6
$Co(CN)_6^{3-}$	CN^- (mg/L)	1.0	1.4	1.2	1.5	1.7
Au	(mg/L)	23.0	26.8	29.0	31.8	33.0
$Au(CN)_2^-$	CN^- (mg/L)	6.1	7.1	7.7	8.4	8.7
Complexed CN	CN^- (mg/L)	607.7	590.8	542.5	542.5	538.6
SCN-	(mg/L)	402.9	411.7	489.3	486.8	538.1
SCN-	CN^- (mg/L)	180.6	184.5	219.3	218.2	241.2
TOTAL CN calc	CN^- (mg/L)	788.3	775.4	761.8	760.7	779.8
CN_{TOTAL} analyzed	CN^- (mg/L)	690.7	579.3	493.0	493.0	533.1
CN^- Added	CN^- (mg/L)	744.4				
Difference between the CN added and CN_{TOTAL} Calculated and Analyzed						
from the Calculated CN_{TOTAL}		5.9%	4.2%	2.3%	2.2%	4.7%
from the Analyzed CN_{TOTAL}[1]		-7.2%	-22.2%	-33.8%	-33.8%	-28.4%

[1] – $CN_{FREE} = CN_{TRIT}$ – CN associated with Zn

[2] – Analyzed $CN_{TOTAL} = CN_{TOTAL} + CN^-$ from SCN^-

[3] – NH_3 concentration was very low and was not included in this balance

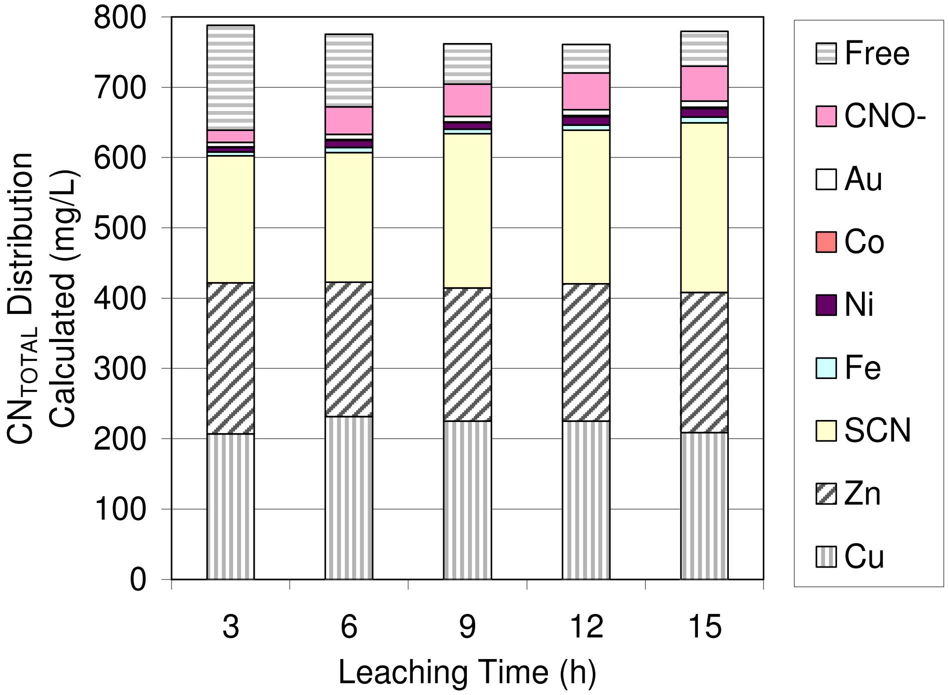


Figure 4.14 - Cyanide Balance per Sampling Time in the Calcine Leaching Circuit – 50% solids, pH 11.5, NaCN corresponding to 913 mg/L in the pulp.
OBS: CN^- associated with Au, Co, Fe is so small that is not visible in the graph.

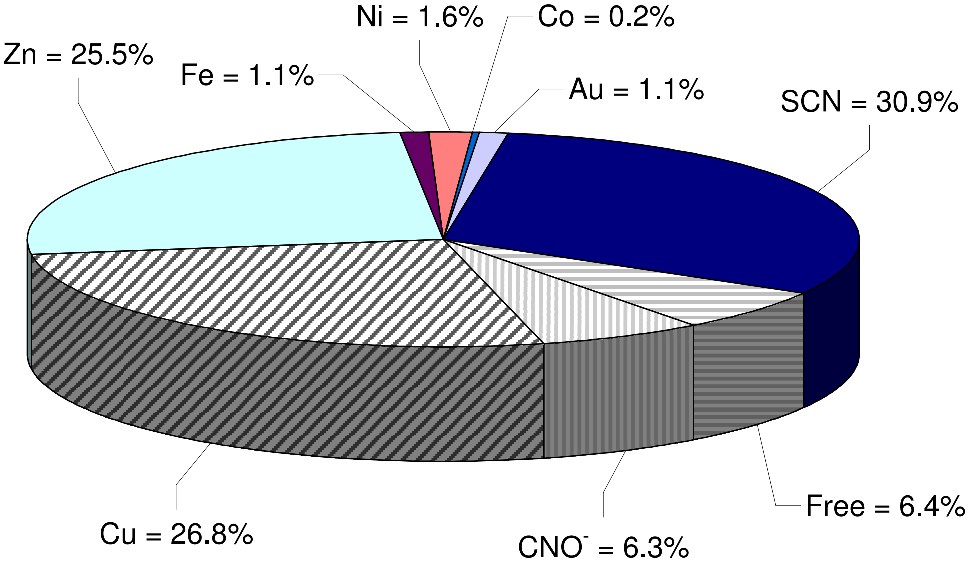


Figure 4.15 – Cyanide Speciation at the end of the Plant Evaluation Campaign.

- The CNO^- concentration increases from the first (28 mg/L) to the last Pachuca Tanks (80 mg/L)
- The CN_{FREE} is very low in all the Pachuca Tanks, i.e., decreasing from 150 to 50 mg/L
- Cyanide is complexed in different proportions compared to those obtained in laboratory (Table IV.17).

Table IV.17 – Cyanide distribution in the Laboratory and Plant Tests

CN- distribution	Laboratory	Plant
Free	14.0%	6.4%
CNO^-	--	6.3%
Cu	28.1%	26.8%
Zn	32.8%	25.5%
Fe	1.6%	1.1%
Ni	0.6%	1.6%
Co	0.1%	0.2%
Au	0.6%	1.1%
SCN	22.2%	30.9%

Confirming this trend, Figure 4.16 shows metal concentration in two different days of sampling in the calcine leaching circuit. It means that there are strong variations in the circuit regarding the metal cyanocomplex concentrations in solution on a day-by-day basis.

Gold Recovery

The head grade was calculated based in the gold concentration in solution at the last tank, corrected to solids concentration, plus the gold tailing grade as showed in the eq. (33). Gold recovery was calculated based on the gold concentration in solution, corrected to solids concentration, divided by the recalculated feed grade as showed in the eq. (34).

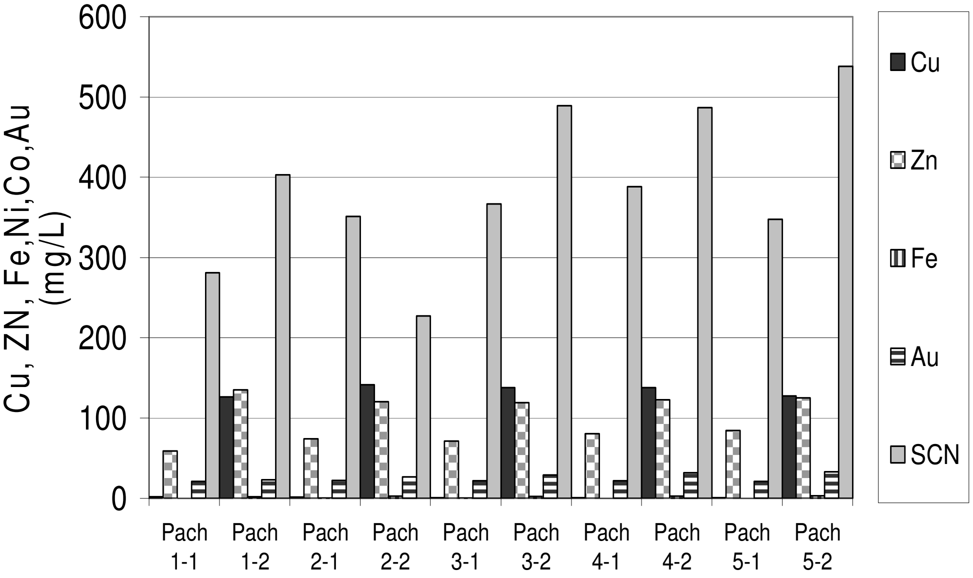


Figure 4.16 - Metals Concentration in Solution in the Calcine Leaching Pachuca Tanks (Pach 1-1 = Pachuca 1 day 1 and Pach 1-2 = Pachuca 1 day 2 and the same for all)

$$C_S^0 = \left(\frac{[Au]_S . V_{SF}}{W_{SOLF}} \right) + C_S^T \qquad (33)$$

C_S^O	Gold concentration in the ore in feed (g/t)
$[Au]_S$	Gold concentration in solution in the final sample (mg/L)
V_{SF}	Solution flowrate (m^3/h)
W_{SOLF}	Solids flowrate (t/h)
C^T_S	Gold concentration in the tailing (g/t)

$$R_{AU} = 100 \frac{\left(\frac{[Au]_S . V_{SF}}{W_{SOLF}} \right)}{C_S^0} \qquad (34)$$

R_{AU}	Gold recovery (%)

Table IV.18 shows gold recovery obtained in the plant evaluation. The 3.9% difference between recalculated and analyzed feed was considered acceptable for this high gold grade due to the presence of free gold which could cause the called "nugget effect". Figure 4.15 shows the experimental results and the fitted curve defined by eq. (35).

Table IV.18 - Gold Balance - Analyzed and Fitted for the Plant Evaluation

Leaching Time (h)	Au Solution (mg/L) @ 50% solids	Gold Recovery (%)	Gold Recovery Fitted Curve (%)
0	0.0	0.0	0.0
2.9	23.0	59.0%	60.4%
5.8	26.8	68.6%	74.3%
8.7	29.0	74.4%	80.5%
11.6	31.8	81.4%	84.0%
14.5	33.0	84.6%	86.3%
Recalculated Feed		30.03 g/t	
Analyzed Feed		31.20 g/t	
Difference		3.9%	

The defined equation, based on the eq. (9), is shown by eq. (35).

$$\frac{1}{(C_S - 1.15)} = 0.013t + 0.0228 \tag{35}$$

Where:

C_S Gold concentration in the solids at time *t* (g/t)

Comparing these results with the laboratory leaching curve one can observe that (Figure 4.16):

- Residual cyanide concentration in solution is much smaller and decreases much faster in the industrial circuit.
- Gold leaching is slower than in the laboratory test.

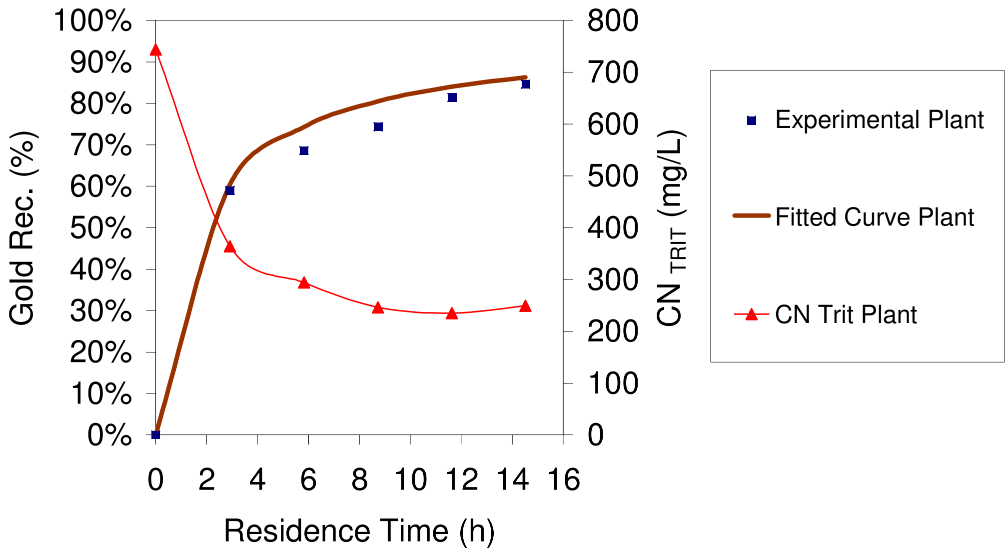


Figure 4.15 – Gold Recovery and CN_{TRIT} Concentration for Plant Evaluation – calculated to 50% solids, pH=11.5, 1.4 kg/t of NaCN

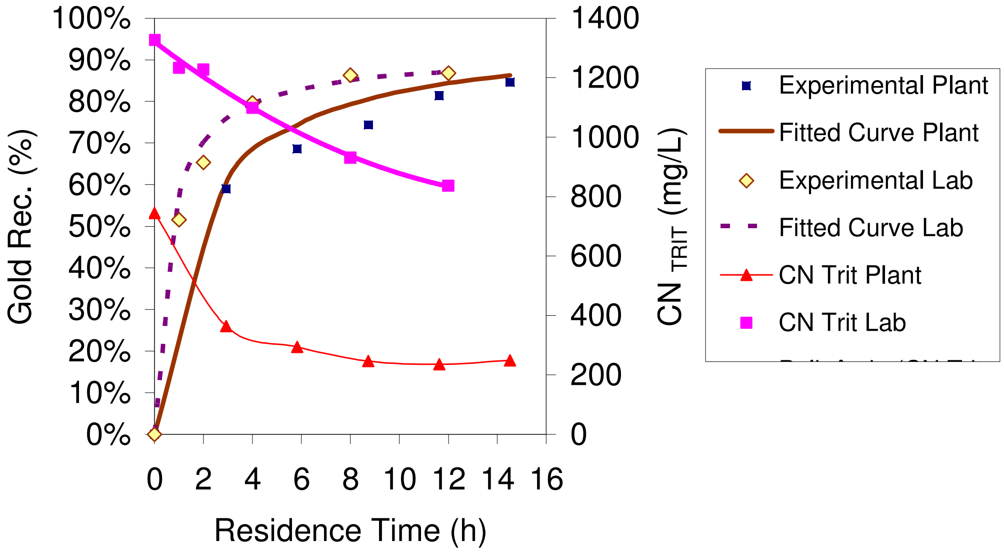


Figure 4.16 – Laboratory and Plant Leaching Curve

The slower gold leaching kinetics in the plant, compared to the laboratory results, can be explained by the lower cyanide concentration in the beginning of leaching and the very quick cyanide consumption in the first Pachuca tanks. Even though part of this cyanide is complexed by zinc, thus not affecting the final gold recovery, leaching rate would be affected. In the plant, cyanide concentration is controlled only by CN_{TRIT}, which include zinc cyanocomplex. Considering the concentrations of zinc cyanide in solution, an investigation was carried out in order to evaluate the effect of increasing the $[Zn(CN)_4^{2-}/CN_{FREE}]$ ratio on gold leaching.

From Figure 4.17 one can see that in concentrations of 1 and 2.5 kg/t of NaCN and the corresponding $NaZn(CN)_4$ (it means that the same amount of CN was added in the pulp in both cases) the kinetics of gold dissolution and final recovery are smaller when zinc cyanide is used. It goes against the literature saying zinc cyanide has no influence on gold leaching (Lorosch, 2001) because enough cyanide ions should be liberated in solution to allow gold leaching. In fact this has happened, however the initial kinetics was slower and perhaps longer residence times would be necessary. It is a good indication that zinc in solution should be also controlled in the industrial circuit to avoid analyzing zinc cyanide as free cyanide . Both, the leaching rate and gold recovery (at 8 h), increase as cyanide concentration increases, regardless of its origin as free or complexed cyanide.

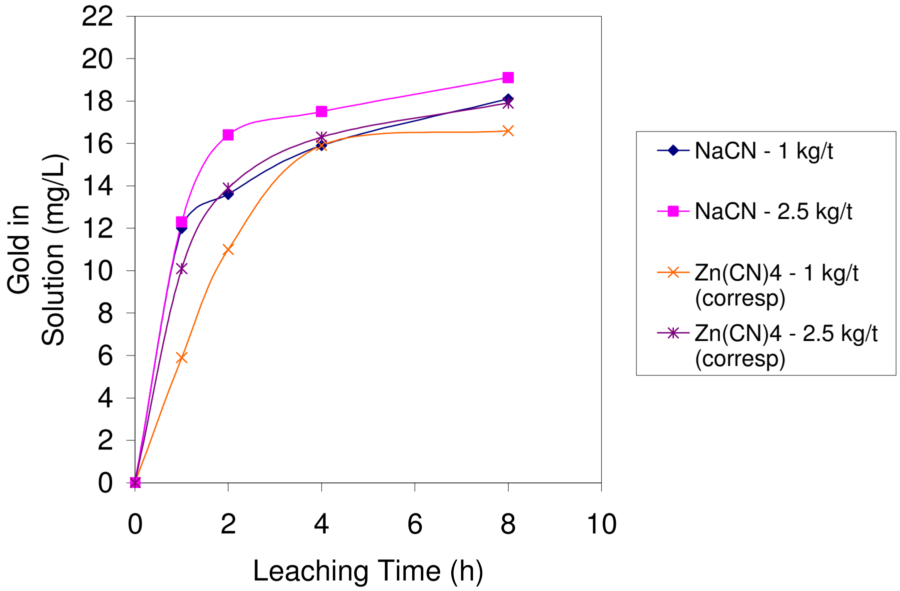


Figure 4.17 – Comparison of the Calcine Gold Leaching in Laboratory Condition Using $NaCN$ and $NaZn(CN)_4$ – 50% solids, pH=11.5, 2 hours pre-lime, 8 hours leaching

Based on all the above work, it was clear that it would be very difficult to get a definitive and complete information about cyanide speciation in the Industrial Calcine Leaching circuit due to the variation in the leaching feed and conditions. It would be also difficult to carry out any test in the plant before getting enough information about the optimized leaching parameters. Thus, it was decided to go back to the laboratory and make the necessary testwork before carrying out any additional industrial test.

4.3. The Influence of Process Variables on Cyanide Consumption and Gold Recovery

Similar investigations on the effect of process variables such as lime, pre-lime time, lead nitrate and cyanide concentration on gold leaching circuits have been carried out in many plants. Recent works have been published with testwork carried out with ore from New Britannia Mine – Canada, Fort Knox Mine – Alaska, Musselwhite Mine – USA, Geant Dormant Mine and Stawell Mines – Canada by Deschênes and Fulton (1999). Additional laboratory studies with ore and concentrates have also been reported in the literature with samples with different mineralogy, grade and gold recoveries (Deschênes *et al.*,1998; Fang *et al.*,1998; Tavani *et al.*,2000).

Gold recovery from the Calcine produced in the Queiróz plant is not expected to be a difficult task, however, as shown above, this material has peculiarities, which makes the leaching a slight more complex. Thus, a large sample was collected in the circuit and studied in laboratory using the procedure discussed in chapter 3. This may avoid strong variations in the ions concentration in solution.

The first part of this work comprised a number of tests to verify the possible error and gold analysis variation that would be expected in the testwork, in the calcine, in order to establish a confidence interval.

Feed Gold Grade

As explained in chapter 3, 120 kg of calcine was taken from the calcine leaching feed. This large sample was dried, homogenized and divided in sub-samples of 3kg each using the JIS – Japonese International Standard methodology. Nine 3kg samples were taken randomly from these sub-samples to obtain the feed gold grade variation. Table

IV.19 shows the results. The average was shown to be 29.57 ± 0.878 g/t with 95% of confidence level.

Table IV.19 - Analyzed Feed Gold Grades, by Fire Assay, in the Calcine Samples

Sample Number	Gold Grade (g/t)
1	29.73
2	30.27
3	29.65
4	29.93
5	29.43
6	28.90
7	29.13
8	29.20
9	29.87
Average	**29.57**
St. Deviation	0.439
Avr. + 2SD	30.45
Avr. - 2SD	28.69

Table IV.20 summarizes the recalculated feed grade from the tests, to be discussed in the following section. The average of the recalculated feed grade was virtually the same of the analyzed feed grade, 29.47 ± 1.98 g/t (95% confidence level) against 29.57 ± 0.88 g/t but the standard variation was higher (1.98 against 0.88 g/t). Figure 4.18 illustrate the distribution of the recalculated gold grades with respect to the confidence interval. The results indicate that gold balance was good enough for a good interpretation of the testwork. Only two tests (LBII 007 and 018) were out of the interval defined by the average ± 2 SD, and were discarded.

Table IV.20 - Average of all Recalculated Feed Gold Grade from the Laboratory Tests

Test	Recalculated Gold Grade (g/t)
LBII001	28.59
LBII002	27.92
LBII003	30.95
LBII004	29.20
LBII005	29.90
LBII006	30.19
LBII007	33.62
LBII008	29.60
LBII009	27.43
LBII010	27.73
LBII011	29.03
LBII012	30.01
LBII013	28.88
LBII014	29.36
LBII015	29.44
LBII016	30.13
LBII017	28.71
LBII018	36.76
LBII019	28.65
LBII020	28.51
LBII021	30.41
LBII022	31.43
LBII023	28.92
LBII024	28.36
LBII025	28.65
LBII026	28.38
LBII027	(1)
LBII028	(1)
LBII029	26.99
LBII030	27.32
Average	**29.47**
Standard Dev.	1.98
Avr. + 2SD	33.43
Avr. - 2SD	25.50

(1) Carbon in leaching tests head grade were not recalculated

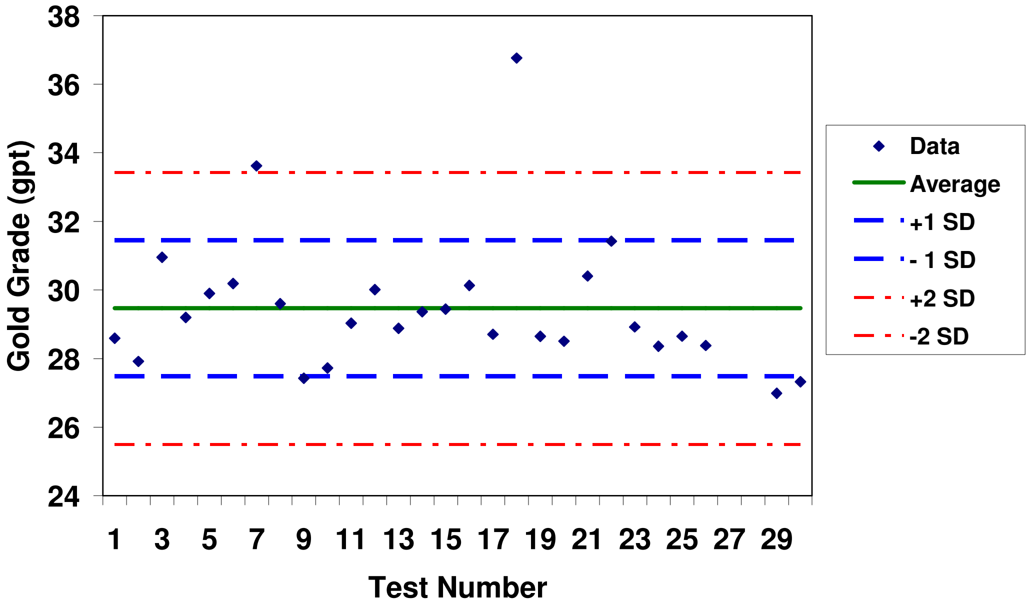


Figure 4.18 - Variation of the Recalculated Feed Gold Grade in the Laboratory Tests

Gold in Solution, Gold Recovery and Cyanide Consumption

In addition to the evaluation of the feed grade variation, five replicates of the *standard leaching* tests were carried out to estimate the magnitude of the experimental error on gold recovery and cyanide consumption. Figure 4.19 shows gold in solution and Figure 4.20 shows gold recoveries. The experimental conditions of these *standard leaching* tests are shown in Table IV.21. Two tests were carried out with deionized water to observe if the industrial water had any influence on the results.

Table IV.21 – Standard Leaching Test Laboratory Conditions

Description	Condition
Calcine, kg	3.0
% de solids	50%
pH	11.0
Pré-lime, h	2
Leaching time, h	8
Solution Volume, L	3
Calcine S.G.	3.51
Volume of Pulp, L	3.85
Pulp Density, kg/L	1.557
NaCN added, g/t	2,500

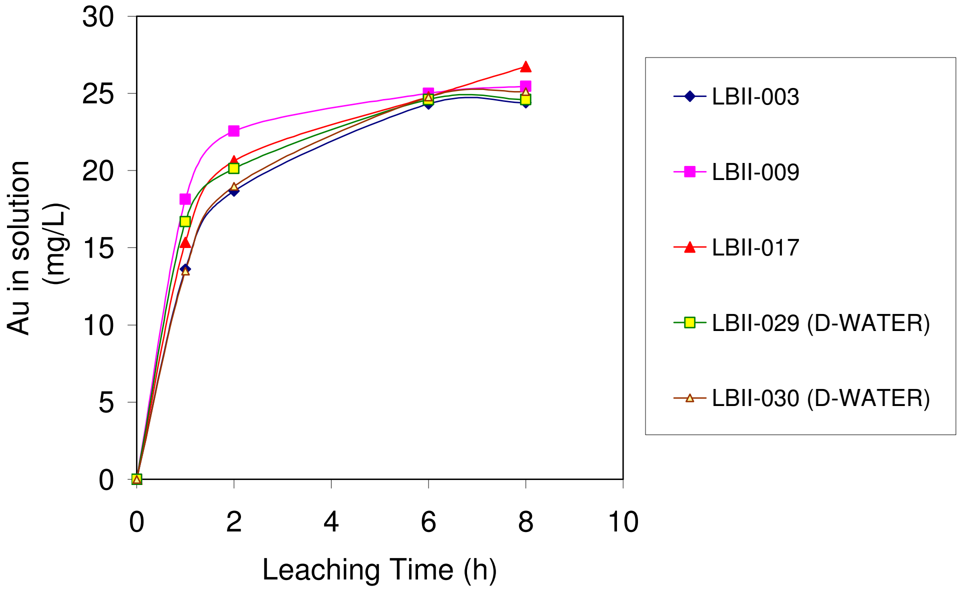


Figure 4.19 –Gold Leaching in Standard Tests using Industrial Water and Deionized Water (D-WATER) – 50% solids, pre-lime 2 hours, pH=11.5, 2.5 kg/t NaCN, leaching time 8 hours

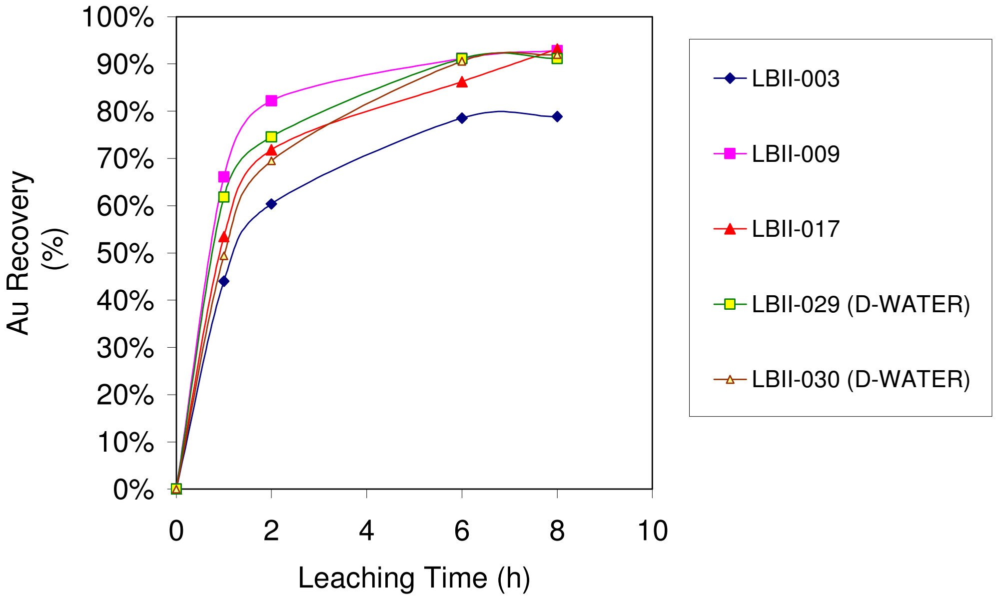


Figure 4.20 - Gold Recovery in Standard Tests using Industrial Water and Deionized Water (D-WATER) – 50% solids, pre-lime 2 hours, pH=11.5, 2.5 kg/t NaCN, leaching time 8 hours

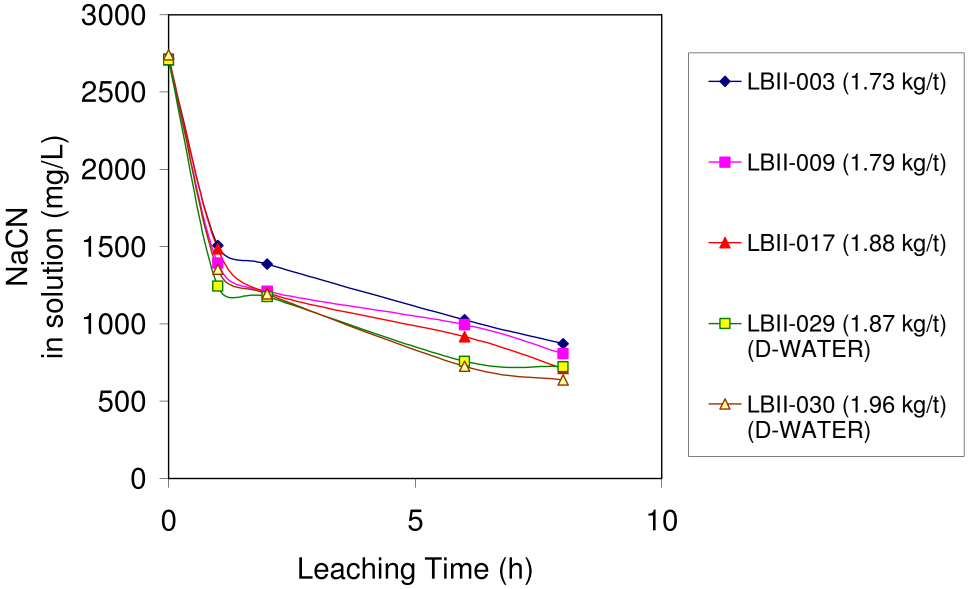


Figure 4.21 - Cyanide Consumption in Standard Tests Using Industrial Water and Deionized Water (D-WATER) – 50% solids, pre-lime 2 hours, pH=11.5, 2.5 kg/t NaCN, leaching time 8 hours

From these figures one can observe that though there are some differences in the leaching curves, except for the test LBII-003, gold recoveries were quite similar in the end, varying from 91.1 to 93.1%, with an average of 92.23%. Cyanide consumptions were also very similar; deionized water did not influence the results. The error in the final gold recovery was found to be 2.0% and in cyanide consumption it was 7.4%. Considering these results, these parameters were used with confidence to analyze the effect of lime, pre-lime time, lead nitrate and cyanide concentration on the leaching tests.

4.3.1. Lime Addition in the Pre-Lime

At Nova Lima Plant, now shut down, the lime addition in the pre-lime used to be an usual procedure. Sometimes however it was used in excess to the normal dosage; this procedure is called “alkaline protection”. The excess of lime, which is not totally soluble, was used to avoid pyrrhotite and pyrite oxidation in the leaching circuit, which would result in an increase of cyanide consumption.

The influence of lime addition in the pre-lime on gold recovery and cyanide consumption was studied. The lime was added in the beginning of the 2 hours pre-lime and gold recovery, SCN, SO_4^{-2}, Cu, Zn and CN_{TRIT} were analyzed at the end of this period. In the present work, thiosulfate could not be analyzed due to the lack of facilities in the Queiróz plant chemical laboratory. Figures 4.22 shows the influence of lime addition on gold recovery, SCN and SO_4^{-2} formation in solution. Increasing lime addition in the pre-lime stage up to 15 kg/t did not affect SCN^- formation as well gold recovery and sulfate formation. Nevertheless, cyanide consumption decreased substantially as well as soluble copper concentration.

If lime addition increases beyond 15 kg/t, gold recovery decreases substantially, from 93 to 71%, due probably to the viscosity increase in the pulp, which in turn decreases the oxygen available for gold leaching. The viscosity was not measured but the change was evident during the experiments. Deschênes *et al.*(1998) also showed that cyanide consumption is inversely proportional to the amount of lime added. He also found that lime addition in large dosages decreases gold recovery, however, in a lesser proportion then found in the present work.

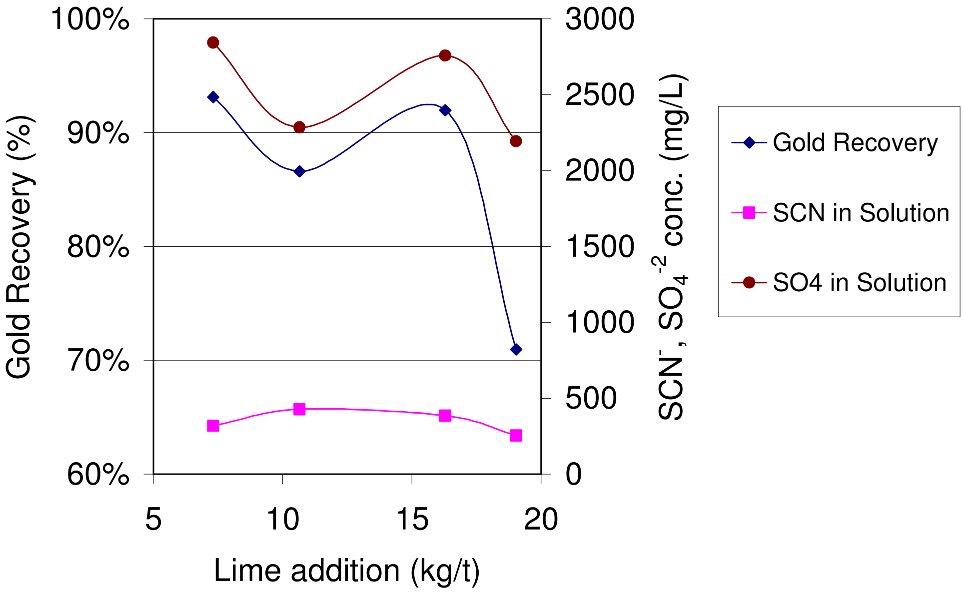


Figure 4.22 - Effect of Lime Addition in the Pre-lime on Gold Recovery, SCN and SO_4^{-2} Formation in the Calcine Leaching – Pre-leaching: 2 hours, Gold leaching: 8 hours, NaCN: 2.5 kg/t, 50% solids.

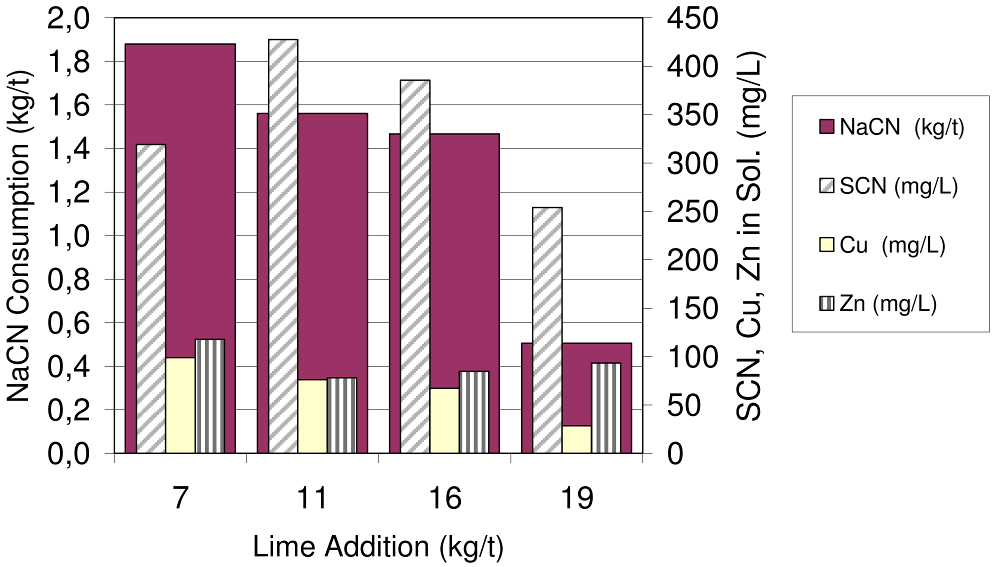


Figure 4.23 - Influence of the Lime Addition on Calcine Leaching – Lime was added in the beginning of the pre-lime – pre-lime: 2 hours, leaching with 2.5kg/t NaCN, 8 hours, 50% solids

A new continuous test campaign in the plant should be carried out carefully controlling lime addition between 7 and 10 kg/t of ore to confirm the above results. With this lime dosage cyanide consumption should decrease from 1.88 to 1.56 kg/t (22%) and gold recovery should not be affected. The benefit is shown below. Lime cost is US$0.05/kg while the cyanide cost is US$1.05/kg thus, even increasing lime addition from 7 to 10 kg/t this modification could give a revenue of :

Lime increasing	(10 -7 kg/t)*US$0.05/kg =	+US$0.150/t
Cyanide savings	(1.88-1.56 kg/t)*US$1.05/kg =	- US$0.336/t
Savings		- US$0.186/t

Considering savings of US$ 0.186/t and 17t/h of calcine treated in the circuit the total saving could be of US$26,591/year (0.186*17t/h*365*24*96%) in the Queiróz Plant; after the plant expansion in 2006, this savings would be the double.

4.3.2. Pre-Lime Duration

Due to the presence of pyrite and pyrrhotite in the calcine, pre-lime could become an important procedure to avoid excessive cyanide consumption. Lead nitrate is also reported to have a benefit on gold recovery and cyanide consumption when added in the beginning of the pre-lime due to the prevention of thiocyanide formation during leaching.

Different pre-lime duration (0, 2, 8, 12 h) was investigated and 100 g/t of lead nitrate was added in some tests to verify its effect on gold recovery and cyanide consumption. Copper, zinc and SCN^- concentrations were also measured in the end of the tests to define how much cyanide was consumed. Figure 4.24 shows gold recovery and SCN^- formation following the planned pre-lime time and 8 hours leaching time. By increasing the pre-lime duration gold recovery increased from 91.5 to 93.9%, a variation in the range of the experimental error. However, by increasing pre-lime duration, cyanide consumption decreased and oxygen dissolved in the pulp increased.

Lead nitrate seems to have a detrimental effect on gold recovery. With the addition of 100 g/t of lead nitrate gold recovery decreased from 89.3% (no pre-lime) to 83.8% (2 hours of pre-lime). By increasing the pre-lime duration to 6 and 12 hours gold recovery goes back to the level of 93%. The increase of pre-lime duration did not affect thiocyanide formation even in the presence of lead nitrate.

The above results do not corroborate many of the findings in the literature, which shows the benefit of lead nitrate in systems containing soluble sulfides. Deschênes *et al.*(1998, 1999 and 2000) showed that there is an optimum dosage of lead nitrate depending on mineralogy. Lead nitrate usually increases gold leaching rate up to a maximum. In excess, lead nitrate will show the opposite effect.

Figure 4.25 shows oxygen concentration after pre-lime (or in beginning of the leaching) and Figure 4.26 shows metal concentration in solution. One can see in Figure 4.25 that lead nitrate had a detrimental effect on the dissolved oxygen concentration in the pulp, which could explain why gold recovery was affected when it was added in the pulp. Increasing pre-lime duration, copper and zinc are precipitated from the solution and as consequence, cyanide consumption decreases. Table IV.22 shows the results..

These results confirm the previous ones. By increasing pre-lime duration up to 12 hours, which in turn increases lime consumption from 6.3 to 9.6 kg/t (52.3%), cyanide consumption would decrease from 1.95 to 1.70 kg/t (12.8%) while gold recovery would increase from 91.52 to 93.87% (2.35%). The net savings, considering lime costs of US$0.05/kg and cyanide costs of US$1.05/kg, is calculated below.

Table IV.22 – Effect of the Pre-lime in Cyanide and Lime Consumption and Gold Recovery

Pre-lime	NaCN consumption (kg/t)		Lime Consumption (kg/t)		Gold Recovery (%)	
(hours)	0 g/t Pb	100 g/t Pb	0 g/t Pb	100 g/t Pb	0 g/t Pb	100 g/t Pb
0	1.95	2.01	6.3	6.6	91.52	89.34
2	1.79	1.90	7.3	7.1	92.85	83.87
6	1.70	1.65	8.4	8.1	93.72	93.67
12	1.70	1.75	9.6	8.4	93.87	92.12

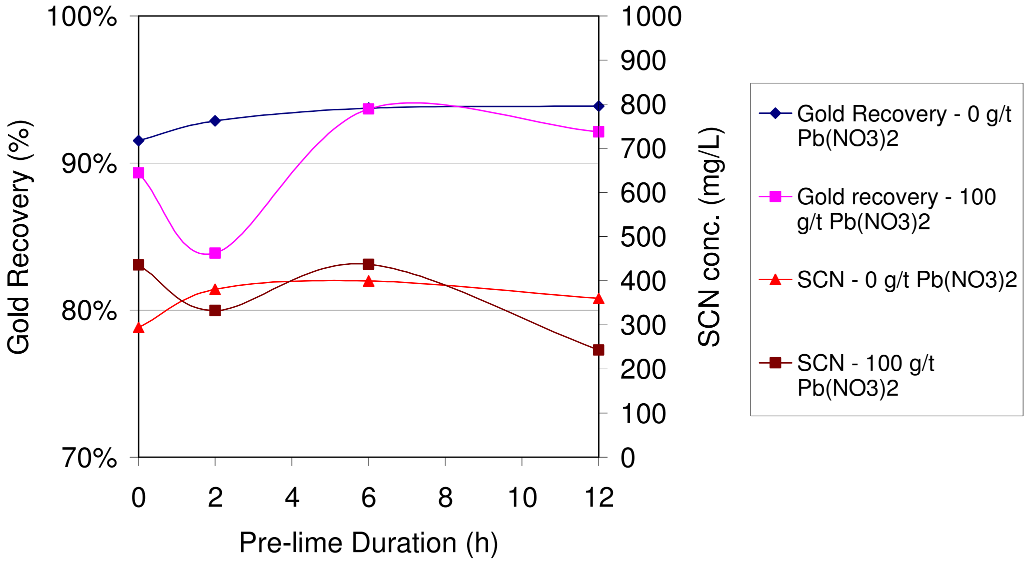


Figure 4.24 – Effect of Pre-lime and Lead Nitrate Addition on Calcine Leaching – Leaching: 2.5 kg/t NaCN, pH=11.5, 8 hours, 50% solids

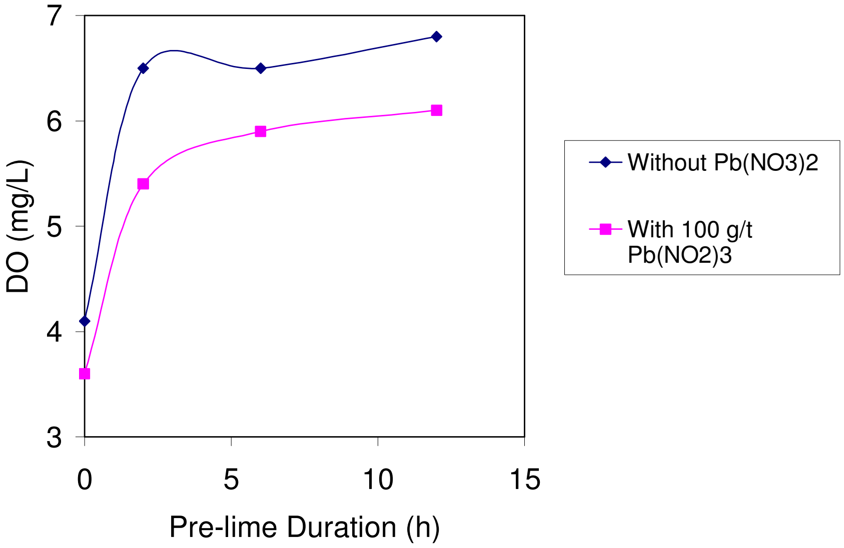


Figure 4.25 - Effect of Pre-lime and Lead Nitrate Addition on Dissolved Oxygen in the Beginning of the Calcine Leaching – Leaching: 2.5 kg/t NaCN, pH=11.5, 8 hours, 50% solids

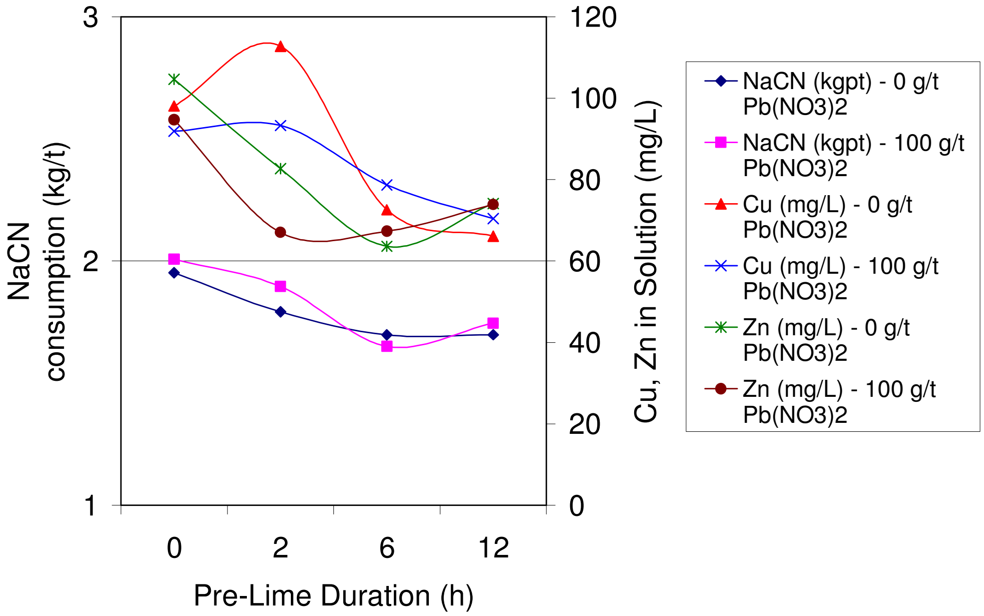


Figure 4.26 - Effect of pre-lime and lead nitrate addition on metal dissolution and NaCN consumption on Calcine Leaching – 2.5 kg/t NaCN, pH=11.5, 8 hours, 50% solids

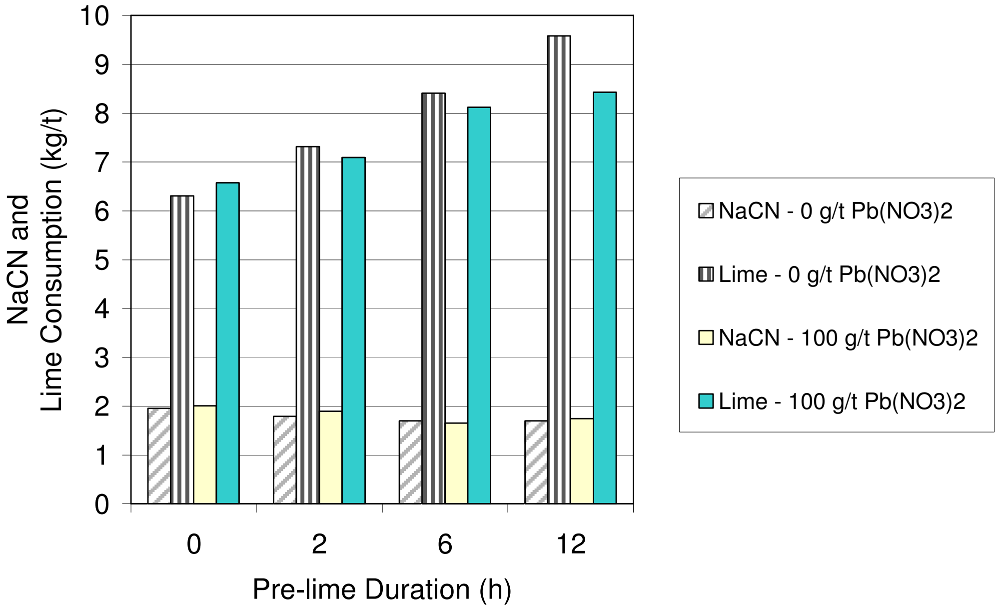


Figure 4.27 **-** Effect of pre-lime and lead nitrate addition on metal dissolution and NaCN consumption on Calcine Leaching –2.5 kg/t NaCN, pH=11.5, 8 hours, 50% solids

Considering a net savings of US$0.098/t, as shown below, and 17t/h of calcine treated, the annual net revenue could be of US$ 13,939/year (0.098*17t/h*365*24*96%). For a increase in gold recovery of 2.35% the total revenue could reach US$ 1.28M/year [(29.47g/t*2.35%*17t/h*365*24*96%*US$400/oz)/31.104]. A new campaign in the plant should be carried out to confirm these results.

Lime increasing	(9.6-6.3 kg/t)*US$0.05/kg =	+US$0.165/t
Cyanide savings	(1.95-1.70 kg/t)*US$1.05/kg =	- US$0.263/t
Savings		- US$0.098/t

4.3.3. NaCN influence

The effect of cyanide dosage was determined under the following conditions: pre-lime: 2 hours, with and without 100 g/t $Pb(NO_3)_2$, pH=11.5, Leaching: 8 hours and pH=11.5. Figure 4.28 shows the results. Gold recovery increased from 70% to 93% when cyanide dosage increased from 1.5 to 2.0 kg/t of cyanide with and without lead nitrate. By increasing the cyanide up to 5 kg/t gold recovery decreased to 82% and 88% without and with 100 g/t of lead nitrate respectively. Figure 4.28 also shows that SCN^- concentration increases with the increase of cyanide concentration. Figure 4.29 and Figure 4.30 show the metal concentration and the cyanide consumption in the end of 8 hours leaching time. The use of an excess of cyanide results in unnecessary cyanide consumption and has a detrimental effect on gold recovery, probably due to the high concentration of thiocyanide and metal complexes in solution.

This phenomenon of high cyanide consumption, in the calcine, is attributed to the presence, of reactive pyrite and pyrrhotite (Luthy and Bruce, 1979) and also leachable copper due to the addition of copper sulfate in flotation as activator. Deschênes *et al.* (1997) also showed that pyrrhotite has lower solubility than pyrite; the cyanide consumption by pyrite was 7 times higher than pyrrhotite. It was not carried out any test increasing pre-lime time and cyanide concentration, as the increase of cyanide to 2 kg/t of cyanide led to similar recovery as those obtained in other tests. Lead nitrate had little or no effect on gold recovery, thiocyanide formation, and metal dissolution.

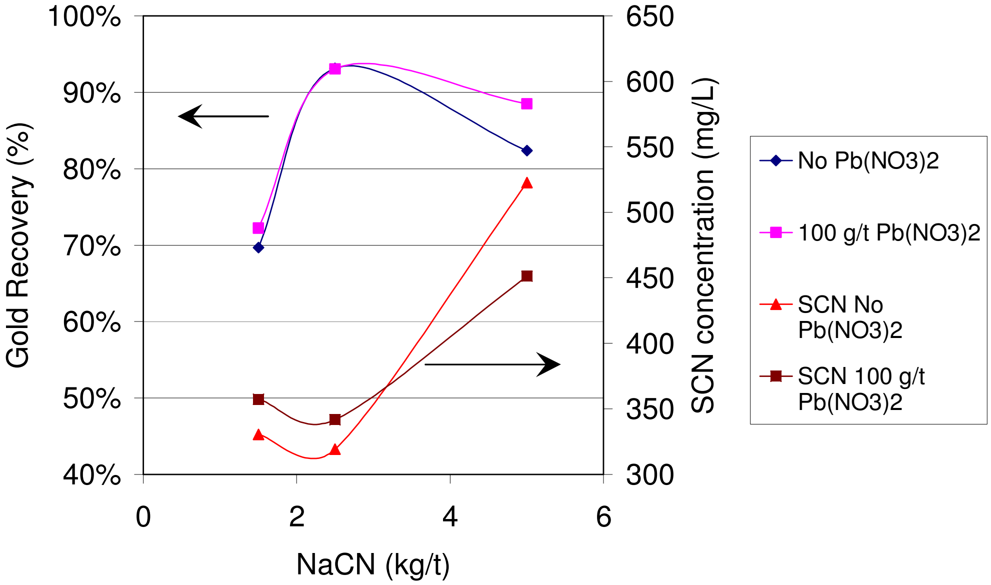


Figure 4.28 - Effect of Cyanide Concentration and Lead Nitrate Addition Gold Recovery and SCN$^-$ Formation on Calcine Leaching – pre-lime: 2 hours, pH = 11.5, Leaching: pH=11.5, 8 hours, 50% solids

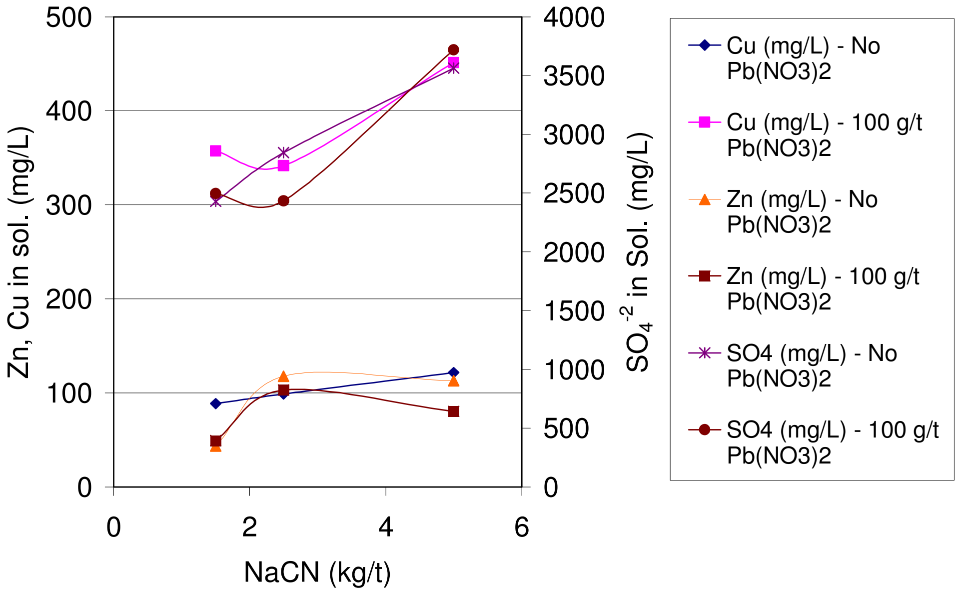


Figure 4.29 - Effect of Cyanide Concentration and Lead Nitrate Addition on Metal Dissolution on Calcine Leaching – pre-lime: 2 hours, pH = 11.5, Leaching: pH=11.5, 8 hours, 50% solids

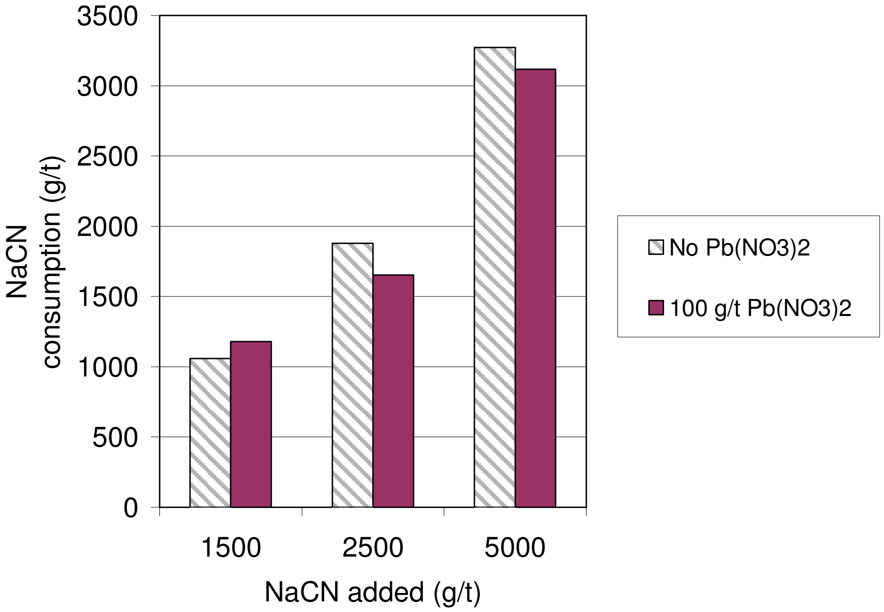


Figure 4.30 - Cyanide Consumption on Calcine Leaching – pre-lime: 2 hours, pH = 11.5, Leaching: pH=11.5, 8 hours, 50% solids

4.3.4. pH Influence

The pH is one of the important gold leaching parameters. The influence of the pH variation in the pre-lime and in the leaching stages was investigated. Figure 4.31 shows the results. The pH had very little or no influence on gold recovery in studied range; there is no explanation for the lower recovery at pH 10.5.

4.3.5. Lead Nitrate

Lead nitrate has been abandoned along of the last few years due to environmental problems caused in the smelting house and due it's the high cost. Part of the lead goes together with the zinc precipitate to the refinery and once it is smelted, lead is liberated causing health problems in the refinery even with good exhausting system. However, recent studies (Deschênes *et al.*,1998; Fang *et al.*,1998; Tavani *et al.*, 2000) concluded that lead nitrate can improve significantly gold leaching kinetics of some ores. In some cases it may work better than the increase of oxygen injection in the pulp.

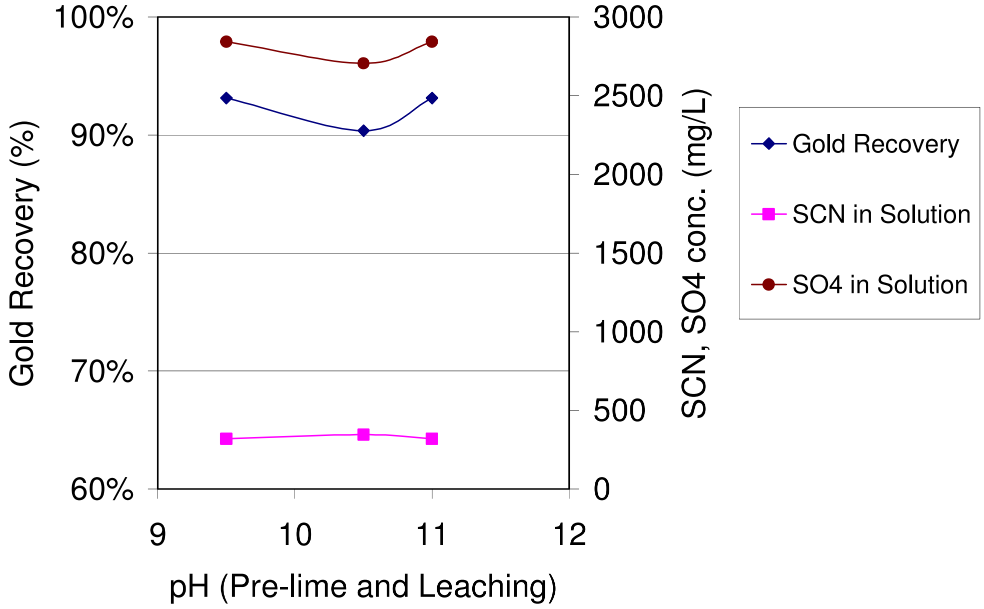


Figure 4.31 - Effect of the pH on gold dissolution – pre-lime: 2 hours, Leaching: pH=11.5, 8 hours, 50% solids

Deschênes *et al.* (1999) have studied various industrial uses of lead nitrate. They found that there should be an upper limit of the lead nitrate addition to the pulp. Even in the same plant, ore characteristics can change daily and therefore lead can improve or hinder gold leaching kinetics. It was also established that lead shows the maximum benefit when pre-lime is employed.

As discussed in chapter 2, lead nitrate was expected to have a good influence on the calcine leaching due to the presence of pyrite and pyrrhotite mixed with the pure calcine. The influence of lead nitrate addition was studied in a range varying from zero to 200g/t. Figure 4.32 shows gold recovery, sulfate and thiocyanide formation in these tests. Figure 4.33 shows the cyanide consumption. From the Figure 4.32 one can see that in low dosage, 5 to 50 g/t, lead had a very detrimental effect on gold recovery and no effect on thiocyanide formation.

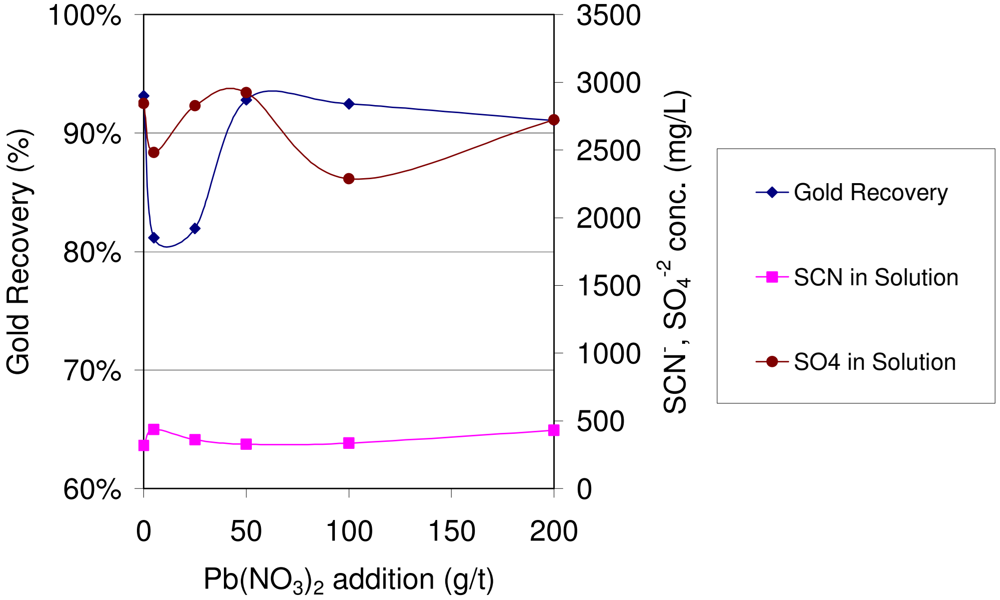


Figure 4.32 - Effect of the Lead Nitrate on Gold Dissolution on Calcine Leaching – pre-lime: 2 hours, Leaching: pH=11.5, 8 hours, 50% solids

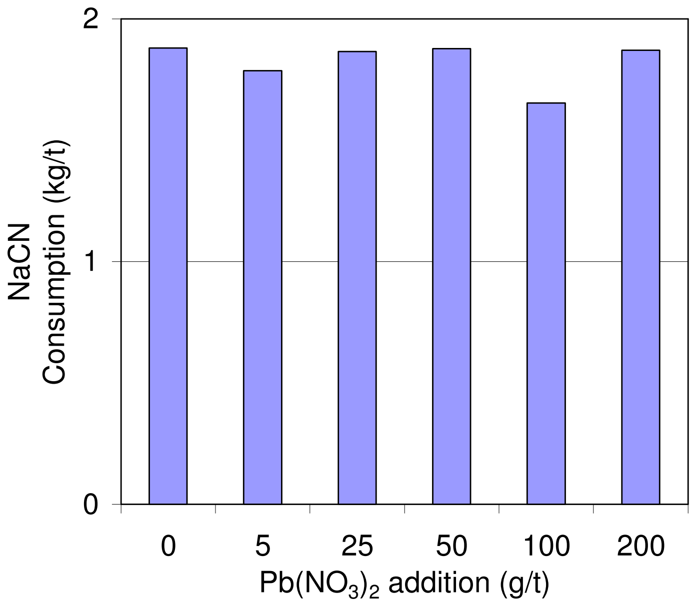


Figure 4.33 - Effect of the lead nitrate on the cyanide consumption on Calcine Leaching – pre-lime: 2 hours, Leaching: pH=11.5, 8 hours, 50% solids

Jeffrey and Breuer (2000) mentioned that a protective layer of the type $Au\backslash S_x$ formed on gold surface hinders the rate of gold leaching. It could explain the detrimental effect on gold recovery at low lead nitrate concentration. Lead had also no influence on cyanide consumption, confirming the lack of effect on thiocyanide formation.

4.3.6. Activated Carbon

The influence of activated carbon on cyanide consumption was also investigated. The carbon, 24 g/L, was added to the pulp just after cyanide addition, following 2 hours of pre-lime. Table IV.23 shows the results. Carbon had no influence on gold recovery and cyanide consumption. There was not observed copper and zinc adsorption on the carbon.

Table IV-23 – Results of the leaching tests in presence of activated carbon

Test no	Cyanide Consumption (kg/t)	Gold Tails. (g/t)	Recovery (%)
27	2.0	2.01	93.2%
28	2.0	2.01	93.2%

4.4. Laboratory Confirmation Test

Three laboratory tests were carried out to confirm the beneficial effects of pre-lime on cyanide consumption. Three different materials were used: pure calcine (calcine directly from the roaster), calcine (pure calcine plus the fines from the “circuit of fines”) and composed concentrate (pyrite concentrate + graphite concentrate). The latter material is usually treated in the leaching circuit when the bi-annual overall maintenance in the roaster is carried out. Tables IV.24 to IV.26 show the results.

One can see that for pure Calcine the pre-lime had no influence on gold recovery but cyanide consumption can be 45% less (from 1.15 to 0.63 kg/t) than without pre-lime. Lime consumption increased 33% (from 4.8 to 6.4 kg/t). For the composed concentrate gold recovery increased 2.9% (from 75.2 to 78.1 %) when pre-lime duration increased from 0 to 16 hours, however, cyanide consumption decreased only 7% (from 6 to 5.5 kg/t). Lime consumption increased 59% (from 5.6 to 8.9 kg/t).

Table IV.24 – Confirmation Laboratory Tests - Pure Calcine

3.0 kg/t NaCN, 24 hours Leaching, pH=11.5

Pre-Lime (h)	Gold Recovery (%)	Lime Consumption (kg/t)	Lime Consumption Increase (%)	NaCN Consumption (g/t)	NaCN Consumption Decrease (%)
0	93.9	4.8	0%	1148	0%
2	93.9	4.7	0%	895	-22%
6	93.4	5.3	10%	767	-33%
16	93.5	6.4	33%	635	-45%

Table IV.25 – Confirmation Laboratory Tests – Composed Concentrate

6.0 kg/t NaCN, 24 hours Leaching, pH=11.5

Pre-Lime (h)	Gold Recovery (%)	Lime Consumption (kg/t)	Lime Consumption Increase (%)	NaCN Consumption (g/t)	NaCN Consumption Decrease (%)
0	75.2	5.6	0%	5979	0%
2	75.2	6.3	13%	5959	0%
6	77.7	7.4	32%	5855	-2%
16	78.1	8.9	59%	5571	-7%

Table IV.26 – Confirmation Laboratory Tests – Calcine (Pure Calcine + Fines)

4.0 kg/t NaCN, 24 hours Leaching, pH=11.5

Pre-Lime (h)	Gold Recovery (%)	Lime Consumption (kg/t)	Lime Consumption Increase (%)	NaCN Consumption (g/t)	NaCN Consumption Decrease (%)
0	92.8	7.4	0%	2083	0%
2	92.7	8.0	8%	1990	-4%
6	92.1	9.7	31%	1951	-6%
16	92.8	13.7	85%	1622	-22%

Previous results on the reduction of cyanide consumption with the calcine sample (pure calcine + fines) have been confirmed. Pre-lime had no influence on gold recovery but cyanide consumption decreased around 22% (from 2.0 to 1.6 kg/t) when compared with the cyanide consumption without pre-lime. Lime consumption increased 85% (from 7.4 to 13.7 kg/t). The previous results have shown a savings of 12.8 % for cyanide and a lime consumption increase of 52.3%. Making again the savings calculation:

Lime increasing	(13.7-7.4 kg/t)*US$0.05/kg =	+US$0.315/t
Cyanide saving	(2.0-1.6 kg/t)*US$1.05/kg =	- US$0.420/t
Savings		- US$0.105/t

Considering a net savings of US$0.105/t and 17t/h of calcine treated, the annual net revenue could be of US$ 15,011/year (0.105*17t/h*365*24*96%). A campaign in the plant should be carried out to confirm these results.

Making the same calculation for pure calcine:

Lime increasing	(6.4-4.8 kg/t)*US$0.05/kg =	+US$0.080/t
Cyanide saving	(1.1-0.6 kg/t)*US$1.05/kg =	- US$0.525/t
Savings		- US$0.445/t

The annual savings could be even better, US$ 63,618/year, considering the same basis. An industrial test will be carried out to confirm the laboratory results and to establish the optimum operational conditions, which may vary depending on the calcine composition.

5. Conclusions

The main conclusion is that it is possible to reduce cyanide consumption in the plant by increasing the pre-lime time to 10 - 16 hours and increasing the lime addition to 10 – 13 kg/t. This modification would imply in relevant decreases of 22 to 45% in the cyanide consumption and an operational savings between US$ 15,000 to US$ 63,000/year. Some additional conclusions are:

(i) Cyanide speciation is a good tool to understand cyanide consumption.

(ii) The main cyanide consumers are zinc, copper and thiocyanide. Zinc came from the ore, most of the copper came from the copper sulfate added in the flotation circuit as activator.

(iii) Thiocyanide is formed in the leaching circuit as a result of the presence of unroasted sulfides in the leaching feed. In order to avoid an unnecessary cyanide consumption, the fines should not be fed directly to the calcine leaching circuit.

(iv) There is an optimum lime dosage. If superior to 15kg/t, gold recovery decreases, reaching 71% at 19kg lime/t ore. The decrease is due to the viscosity increase in the pulp, which in turn decreases the oxygen available for gold leaching.

(v) The increase of the pre-lime time decreases the metal cyanocomplex formation, thus decreasing cyanide consumption. The increase of the pre-lime time also increased oxygen concentration in the first tanks of the calcine leaching circuit, thus increasing the initial gold leaching rate.

(vi) Zinc cyanide is an effective lixiviant for gold; however, the leaching rate is slightly slower than with free cyanide.

(vii) Lead nitrate addition is not effective in reducing thiocyanide formation. Conversely, lead nitrate had a detrimental effect on both oxygen concentration and gold recovery. This could be not fully understood in the present work.

(viii) The increase of cyanide addition caused a detrimental effect on gold recovery and on cyanide consumption. Considering the actual operational conditions, it looks that the plant is working at the right cyanide dosing point.

(ix) The analytical methodology for CN_{TOTAL} determination based on cyanide distillation underestimates the cyanide in the leaching tanks.

6. Suggestions for Future Works

(i) Modification of the actual operation in order to avoid the presence of unroasted sulfide in the industrial leaching circuit.

(ii) The methodology of CN_{TOTAL} should revised and studied in detail to understand why there was a significant variation between the analyzed cyanide and the added cyanide.

(iii) Chromatography should be used to confirm cyanide speciation.

(iv) An industrial test should be carried out to confirm the laboratory results.

(v) An investigation of the zinc cyanide influence in the circuit should be carried out.

(vi) The substitution of the copper sulfate addition in the flotation circuit should studied to avoid cyanide consumption in the leaching circuit.

7. References

[1] LOROSCH, J. *Process and Environmental Chemistry of Cyanidation.* Degussa AG, 2002. 491p.

[2] SMITH, A and MUDDER, T. *The Chemistry and Treatment of Cyanidation Waste.* Mining Journal Books, 1991. 307p.

[3] LU, J., DREISINGER, D.B.,COOPER, W.C. Thermodynamics of the Aqueous Copper-Cyanide System. *Hydrometallurgy,* v.66 p23 – 36, 2002.

[4] LEHMANN, M.N., KAUR, P., PENNIFOLD, R.M., DUNN, J.G. A Comparative Study of the Dissolution of Hexagonal and Monoclinic Pyrrhotites in Cyanide Solution. *Hydrometallurgy*. v.55 p255 – 273, 2000.

[5] BREUER, P.L., JEFFREY, M.I. The Reduction of Copper(II) and the Oxidation of Thiosulfate and Oxysulfur Anions in Gold Leaching Solutions, *Hydrometallurgy*. v. 70 p163 – 173, 2003.

[6] DYMOV, I., FERRON, C.J., FLEMING. Development of a Novel Cyanide Recovery Process for an Oxidized Gold/Silver Ore. RANDOL GOLD FORUM, 1997, p241 – 246.

[7] MACPHAIL, P.K., FLEMING, C.A., SARBUTT, K.W. Cyanide Recovery by the SART Process for the Lobo-Marte Project, Chile, 1998.

[8] REES, K.L., VAN DEVENTER, J.S.J., Preg-robbing Phenomena in the Cyanidation of Sulfide Gold Ores, *Hydrometallurgy*, v. 58, p61 – 80, 2000.

[9] BOSHOFF, P.T.E., Cyanide-Free AARL Elutions are Feasible. *Minerals Engineering*, v.7, n° 2/3, p251 – 264, 1994.

[10] HERDLEY, N., TABACHNICK, H. Chemistry of Cyanidation. Cyanamid. Quebec, 1968, 54p.

[11] CODERRE, F., DIXON D.G. Modeling the Cyanide Heap Leaching of Cupriferous Gold Ores. Part 1: Introduction and Interpretation of Laboratory Column Leaching Data. *Hydrometallurgy*, v.55, p151 – 175, 1999.

[12] NICOL, M.J. Carbon-In-Leach – Theory. CARBON SCHOOL, MINTEK, 1985, p13.1 – 13.2.

[13] ADAMS, M.D. A Methodology for Determination the Deportment of Cyanide Losses in Gold Plants, *Minerals Engineering*, v.14, n° 4, p383 – 390, 2001.

[14] JEFFREY, M.I., BREUER, P.L. The Cyanide Leaching of Gold in Solutions Containing Sulfide. *Minerals Engineering*, v.13, n° 10-11, p1097 – 1106, 2000.

[15] NGUYEN, A.V., TRAN, T. Technical Note: A Simple Method for Predicting Equilibrium Composition of Leaching Systems. *Minerals Engineering*, v.14, n^o 3, p359 – 364, 2001.

[16] REES, K.L., VAN DEVENTER, J.S.J. The Mechanism of Enhanced Gold Extraction from Ores in the Presence of Activated Carbon. *Hydrometallurgy*, v. 58, p151 – 167, 2000.

[17] GUZMAN, L., SEGARRA, M., CHIMENOS, J.M., FERNANDEZ, M.A., ESPIELL, F. Gold Cyanidation Using Hydrogen Peroxide. *Hydrometallurgy*, v.52, p21 – 35, 1999.

[18] ADAMS, M.D., No.C2108 – Cyanide Removal, MINTEK Communication, p.2, 1994.

[19] VUUREN, VAN C.P.J., SNYMAN, C.P., BOSHOFF, A.J. Gold Losses from Cyanide Solutions. Part I: The Influence of the Silicate Minerals. *Minerals Engineering*, v. 13, n^o 8-9, pp. 823-830, 2000.

[20] VUUREN, VAN C.P.J., SNYMAN, C.P., BOSHOFF, A.J. Technical Note: Gold Losses from Cyanide Solutions. Part II: The Influence of the Carbonaceous Materials Present in the Shale Material. *Minerals Engineering*, v. 13, n^o 10-11, pp. 1177-1181, 2000.

[21] VUUREN, VAN C.P.J. Technical Note: Gold Losses from Cyanide Solutions. Part III: The Recovery of Absorbed Gold From Shale Material. *Minerals Engineering*, v. 14, n^o 4, pp. 439-443, 2001.

[22] FANG Z., MELOUM M., MUHAMMEND M., Modeling and Optimization of Cyanidation Process. A Chemometric Approach by Regression Analysis. *Analytica Chimica Acta* 367, pp. 123-133, 1998.

[23] KONDOS, P.D., DESCHÊNES, G., MORRISON, R.M., Process Optimization Studies in Gold Cyanidation. *Hydrometallurgy* v. 39, pp. 235-250, 1995.

[24] REES, K.L., DEVENTER, VAN J.S.J., DUNNE, R.C., Gold Process Modeling II: The Effect of Ore Type on Leaching and Adsorption Dynamics at Telfer Gold Mine. *Mineral Engineering*, v. 14, n^o. 8, pp. 887-900, 2001.

[25] LORENZEN, L., DEVENTER, VAN J.S.J., Electrochemical Interactions Between Gold and Its Associated Minerals During Cyanidation. *Hydrometallurgy*, v. 30, pp. 177-194, 1992.

[26] LUNA, R.M., LAPIDUS, G.T., Cyanidation Kinetics of Silver Sulfide. *Hydrometallurgy*, v. 56, pp. 171-188, 2000.

[27] HEALEY, S., JEAN, P., DESCHÊNES, G., FULTON, M., Improvement of Cyanidation at New Britannia Mine. Randol Gold and Silver Forum, pp. 149-154, 1999.

[28] HEALEY, S., JEAN, P., DESCHÊNES, G., FULTON, M., Latest Improvement to the Leaching Circuit at New Britannia Mine. Randol Gold and Silver Forum, pp. 301-307, 2000.

[29] TAVANI, S., BROWNER, R.E., Gold Cyanidation Rates in the Presence of Sulphide and Lead Ions. Randol Gold and Silver Forum, pp. 309-314, 2000.

[30] DESCHÊNES, G., HALL, D., FULTON, M., Cyanidation of a Pyrrhotite-Bearing Gold Ore. CANMET, internal report, 2003.

[31] HOLLOW, J., DESCHÊNES, G., GUO, H., FULTON, M., HILL, E., Optimizing Cyanidation Parameters for Processing of Blended Fort Knox and True North Ores at the Fort Knox Mine. CANMET, internal report, 2003.

[32] GOMES, A., CHAVES, A.P., CIMINELLI, V.S.T., The Evaluation of the Cyanidation and Adsorption Processes at Fazenda Brasileiro Gold Mine. *Latin American Perspectives*, SME, pp. 193-202, 1998.

APPENDIX 1

LABORATORY TESTS
CONTROL SHEETS

PRIMEIROS TESTES
DE LABORATÓRIO

Trabalho de Tese de Mestrado

Teste: Laboratório 1
Data: 10/07/2003

Objetivo:

Simular em laboratório o comportamento da polpa na planta e fazer o primeiro balanço de cianeto para checar as análises de cianeto feitas no laboratório

SOLUÇÃO ANALISADA

Tempo (horas)	pH	Au (mg/L)	Zn (mg/L)	Cu (mg/L)	Ni (mg/L)	Fe (mg/L)	Co (mg/L)	As (mg/L)	SO_4^{-2} (mg/L)	S^{-2} (mg/L)	CN_{TOTAL} (mg/L)	CN_{LIVRE} (mg/L)	SCN (mg/L)
0	11,7	0,0	0,01	0,08	0,04	0,03	0,01	1,47	2058	<0.002	0	0	0
1	11,6	10,1	94	83,5	0,44	0,34	0,14	0,11	2474		927	833	206
2	11,4	12,8	108	94,6	0,67	0,75	0,20	0,63	2641		923	829	245
4	11,3	15,6	125	107	1,02	1,49	0,24	0,50	2992		842	742	270
8	11,3	16,9	159	130	2,00	3,59	0,36	0,82	3512		680	629	339
12	11,3	17,0	168	140	2,77	4,63	0,39	1,70	3636		629	565	404

MINERAL ANALISADO

	Au (gpt)	Zn (gpt)	Cu (gpt)	Ni (gpt)	Fe (%)	Co (gpt)	As (gpt)	S (%)	S^{-2} (%)	CN_{TOTAL} (gpt)	CN_{LIVRE} (gpt)	SCN (gpt)
ALIMENTAÇÃO	25,77	2568	674	377	35,79	94	8648	7,58				
REJEITO	3,83	2469	600	393	36,45	100	8744	7,21	<0.002	3,16		

DADOS DO TESTE

Peso de minério, kg	2,266	
% de sólidos	40,32%	
Volume de solução, L	3,354	
S.G. do Calcinado	3,51	
Volume de Polpa, L	4	
Densidade da Polpa, kg/L	1,405	
NaCN adicionado, gpt	2500	
NaCN adicionado, g	5,666	
Conc. de NaCN INICiAL,mg/L	1689,0	
Conc. de CN^- INICiAL,mg/L	896,2	
Conc. de CN^- INICiAL,mg/L	1326,5	Concentração de CN ajustada para 50% de sólidos

NaCN	49
CN	26
CN/NaCN	0,531

AJUSTE DOS TEORES PARA 50% DE SÓLIDOS

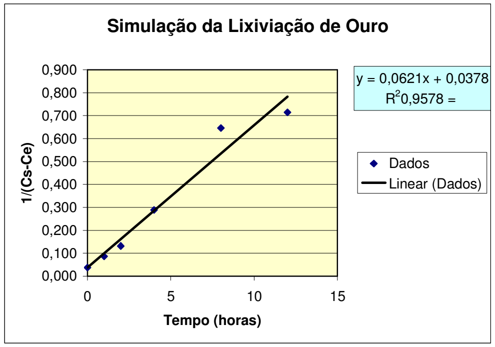
Tempo (horas)	pH	Au (mg/L)	Zn (mg/L)	Cu (mg/L)	Ni (mg/L)	Fe (mg/L)	Co (mg/L)	As (mg/L)	SO_4^{-2} (mg/L)	S^{-2} (mg/L)	CN_{TOTAL} (mg/L)	CN_{LIVRE} (mg/L)	SCN (mg/L)
0	11,7	0,0	0,0	0,1	0,1	0,0	0,0	2,2	3046,2	<0.002			0,0
1	11,6	14,9	139,1	123,6	0,7	0,5	0,2	0,2	3661,9		1372,1	1233,0	304,9
2	11,4	18,9	159,9	140,0	1,0	1,1	0,3	0,9	3909,1		1366,2	1227,1	362,6
4	11,3	23,1	185,0	158,4	1,5	2,2	0,4	0,7	4428,6		1246,3	1098,3	399,6
8	11,3	25,0	235,3	192,4	3,0	5,3	0,5	1,2	5198,3		1006,5	931,0	501,8
12	11,3	25,2	248,7	207,2	4,1	6,9	0,6	2,5	5381,9		931,0	836,3	598,0

RECUPERAÇÃO DE OURO				SIMULA
0	11,7	0,0	1326,5	0
1	11,6	51,6%	1233,0	56,9%
2	11,4	65,3%	1227,1	70,2%
4	11,3	79,6%	1098,3	79,5%
8	11,3	86,3%	931,0	85,1%
12	11,3	86,8%	836,3	87,2%
Alimentação RECAL		**28,99**		
Alimentação ANAL		**25,77**		
Diferença		**11,1%**		

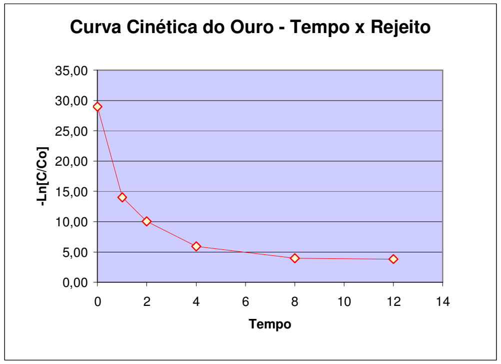
Minério	1/(Cs-Ce)
28,99	0,03765
14,04	0,08611
10,05	0,13129
5,90	0,28800
3,98	0,64599
3,83	0,71429
Ce(ajuste)	2,43

Do Gráfico	
A =	0,0621
B =	0,0378

Minério Simulaçã	1/(Cs-Ce)
28,89	0,0378
12,44	0,0999
8,60	0,1620
5,92	0,2862
4,30	0,5346
3,71	0,7830



t	Alim	C/Co	-ln[C/Co]
0,001	28,99	100%	0,01
1	14,04	48%	0,72491
2	10,05	35%	1,0598
4	5,90	20%	1,59172
8	3,98	14%	1,98626
12	3,83	13%	2,02418



CONSUMO DE CIANETO																												
											Fe se	Fe se													Consum			
Tempo	Cu	$Cu(CN)3^{-2}$	CN	Zn	$Zn(CN)_4^{-2}$	CN	S	SCN	CN	Fe	FeS_2	FeS	$Fe(CN)_6^{-4}$	CN	Ni	$Ni(CN)_4^{-2}$	CN	Co	$Co(CN)_6^{-3}$	CN	Au	$Au(CN)_2^{-}$	CN	TOTAL	CN_{LIVRE}	CN_{LIVRE}-Zn	+Livre	CN_{LIVRE}/Cu
0	0,0	0	0	0,0	0	0	0	0,0	0	0,0			0	0	0,0	0	0	0,0	0	0	0,0	0	0	0	0	0	0	
1	123,6	275,4	202,4	139,1	360,4	221,3	168,2	304,9	136,69	0,5	147,2	294,4	1,91	1,40	0,7	1,80	1,15	0,2	0,76	0,55	14,9	18,90	3,95	567,4	833	611,7	1179,1	4,95
2	140,0	312,0	229,3	159,9	414,1	254,2	200,1	362,6	162,56	1,1	175,1	350,1	4,20	3,09	1,0	2,74	1,75	0,3	1,08	0,78	18,9	23,95	5,00	656,7	829	574,8	1231,5	4,10
4	158,4	352,9	259,4	185,0	479,2	294,2	220,5	399,6	179,15	2,2	192,9	385,9	8,35	6,14	1,5	4,17	2,66	0,4	1,29	0,94	23,1	29,19	6,09	748,6	742	447,8	1196,4	2,83
8	192,4	428,8	315,1	235,3	609,6	374,2	276,8	501,8	224,93	5,3	242,2	484,5	20,12	14,80	3,0	8,18	5,22	0,5	1,94	1,41	25,0	31,62	6,60	942,4	629	254,8	1197,1	1,32
12	207,2	461,8	339,4	248,7	644,1	395,4	329,9	598,0	268,06	6,9	288,7	577,4	25,94	19,09	4,1	11,33	7,23	0,6	2,10	1,53	25,2	31,80	6,64	1037,4	565	169,6	1206,9	0,82
CONSUMO DE CIANETO			**28,2%**			**32,9%**			**22,3%**					**1,59%**			**0,60%**			**0,13%**			**0,55%**			**MEDIA**	**1202,2**	
																										Diferença	**9,4%**	

$Cu(CN)_3^{-2}$	141,5	$Zn(CN)_4^{-2}$	169,4	SCN	58	$Fe(CN)_6^{-4}$	212	$Ni(CN)_4^{-2}$	163	$Co(CN)_6^{-3}$	215	$Au(CN)_2^{-}$	249
CN	104	CN	104	CN	26	CN	156	CN	104	CN	156	CN	52
Cu	63,5	Zn	65,4	S	32	Fe	56	Ni	59	Co	59	Au	197
						FeS_2	120						
						FeS	88						

AMOSTRAGEM
NA PLANTA

AMOSTRAGEM INDUSTRIAL
LIXIVIAÇÃO DE CALCINADO
TESTE nº TI-02

ENTRADA DE DADOS SÓLIDOS **SG Calcinado 3,51**

Ponto de Amostragem	Forma	Polpa (m^3/h)	Sólido (t/h)	% Sólidos	Densidade Polpa (t/m^3)	O_2 dissolvido	pH	Conc. de Carvão (gpL)	Eh	Au (gpt)	Zn (gpt)	Cu (gpt)	Ni (gpt)	Fe (%)	Co (gpt)	As (gpt)	SO_4^{-2} (%)	S^{-2} (%)	CN_{TOTAL} (gpt)	CN_{TRIT} (gpt)	SCN (gpt)	Ca (gpt)
Alimentação do Pré-lime	Minério	25,8	17,8	46,23%	1,494	0,0	0,0	0,0	0,0	0,00	0	0	0	0,0	0	0	0,0	0,0	0,0	0,0	0,0	0,00
Alimentação da Cianetação	Minério	26,7	17,8	45,13%	1,477	0,0	0,0	0,0	0,0	0,00	0	0	0	0,0	0	0	0,0	0,0	0,0	0,0	0,0	0,00
Descarga Pachuca 1	Minério	27,4	17,8	44,42%	1,466	0,0	0,0	0,0	0,0	0,00	0	0	0	0,0	0	0	0,0	0,0	0,0	0,0	0,0	0,00
Descarga Pachuca 2	Minério	27,6	17,8	44,20%	1,462	0,0	0,0	0,0	0,0	0,00	0	0	0	0,0	0	0	0,0	0,0	0,0	0,0	0,0	0,00
Descarga Pachuca 3	Minério	27,5	17,8	44,25%	1,463	0,0	0,0	0,0	0,0	0,00	0	0	0	0,0	0	0	0,0	0,0	0,0	0,0	0,0	0,00
Descarga Pachuca 4	Minério	27,6	17,8	44,17%	1,462	0,0	0,0	0,0	0,0	0,00	0	0	0	0,0	0	0	0,0	0,0	0,0	0,0	0,0	0,00
Descarga Pachuca 5	Minério	27,6	17,8	44,13%	1,461	0,0	0,0	0,0	0,0	6,00	0	0	0	0,0	0	0	0,0	0,0	0,0	0,0	0,0	0,00
Descarga Pachuca 6	Minério																					
Descarga Pachuca 7	Minério																					
Alimentação CIP	Minério																					
Descarga Tanque 1	Solução																					
Descarga Tanque 2	Solução																					
Descarga Tanque 3	Solução																					
Descarga Tanque 4	Solução																					
Descarga Tanque 5	Solução																					
Rejeito CIP	Minério																					

ADIÇÃO DE NaCN - Solução Concentrada

Concentração	kg/m^3	100,0	10%			
Densidade da Solução	t/m^3	1,10				
Vazão de Solução	m^3/h	0,25				
Adição de NaCN	kg/h	25,00		NaCN	49	
Adição de Cianeto	kg/t	1,403		CN	26	
Concentração NaCN em solução	g/m^3	913	44,4% Solidos	CN/NaCN	0,531	13,3
Concentração CN em solução	g/m^3	485				
Concentração CN em solução	g/m^3	744	50% Solidos	(ou seja se a polpa estive a 50% de sólidos e a mesma quantidade de cianeto fosse adicionada esta seria a concentração de CN em solução)		

CÁLCULOS COM SOLUÇÃO DE CIANETAÇÃO

Ponto de Amostragem	Forma	Solução (m^3/h)	Solução (t/h)	% Sólidos	Densidade Polpa (t/m^3)	O_2 dissolvido	pH	Conc. de Carvão (gpL)	Eh	Au (mg/L)	Zn (mg/L)	Cu (mg/L)	Ni (mg/L)	Fe (mg/L)	Co (mg/L)	As (mg/L)	SO_4^{-2} (mg/L)	S_{TOTAL} (mg/L)	CN_{TOTAL} (mg/L)	CN_{TRIT} (mg/L)	SCN (mg/L)	R.Power	NH_3 (mg/L)	CNO- (mg/L)	TSD (mg/L)	Ag (mg/L)
Alimentação do Pré-lime	Solução	20,73	0			0	6,3	0	0	0	2,65	0,05	0,73	0,28	0,25	0	2260	754	0,2	0	29	40	2,97	1,71	2746	0,00
Alimentação da Cianetação	Solução	21,66	0			0	11,3	0	0	0	0	0,16	0,23	0	0,03	0	2080	753	0,32	0	170	229	3,21	14,7	5462	0,00
Descarga Pachuca 1	Solução	22,30	0			0	11,5	0	0	18,4	108	101	2,87	1,57	0,31	0	2400	840	552	291	322	488	3,21	24,5	5420	0,75
Descarga Pachuca 2	Solução	22,50	0			0	11,3	0	0	21,4	96	113	4,54	2,08	0,42	0	2400	800	463	235	329	537	2,68	54,8	5588	1,20
Descarga Pachuca 3	Solução	22,45	0			0	11,2	0	0	23,2	95	110	4,14	1,92	0,35	0	1820	607	394	197	391	541	3,43	63,8	5510	1,32
Descarga Pachuca 4	Solução	22,52	0			0	11,4	0	0	25,4	98	110	5,42	2,19	0,44	0	1940	647	394	188	389	541	2,68	72,8	5470	1,40
Descarga Pachuca 5	Solução	22,56	0			0	11,4	0	0	26,4	100	102	5,68	2,35	0,51	0	2140	714	426	199	430	541	1,82	68,6	4326	1,42
Descarga Pachuca 6	Solução	0,00	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Descarga Pachuca 7	Solução	0,00	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Alimentação Precip. Zinco	Solução		0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Descarga Precip. Zinco	Solução		0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Alimentação CIP	Solução	0,00	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Descarga Tanque 1	Solução	0,00	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Descarga Tanque 2	Solução	0,00	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Descarga Tanque 3	Solução	0,00	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Descarga Tanque 4	Solução	0,00	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Descarga Tanque 5	Solução	0,00	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Eluate	Solução	0	0			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

TRANSFORMANDO TUDO PARA 50% DE SÓLIDOS PARA FACILITAR OS CÁLCULOS - RELAÇÃO DIRETA COM O MINÉRIO

Ponto de Amostragem	Forma	Solução (m^3/h)	Solução (t/h)	% Sólidos	Densidade Polpa (t/m^3)	O_2 dissolvido	pH	Conc. de Carvão (gpL)	Eh	Au (mg/L)	Zn (mg/L)	Cu (mg/L)	Ni (mg/L)	Fe (mg/L)	Co (mg/L)	As (mg/L)	SO_4^{-2} (mg/L)	S_{TOTAL} (mg/L)	CN_{TOTAL} (mg/L)	CN_{TRIT} (mg/L)	SCN (mg/L)	R.Power	NH_3 (mg/L)	CNO- (mg/L)	TSD (mg/L)	Ag (mg/L)
Alimentação do Pré-lime	Solução			50,0%			6,3			0,00	3,08	0,06	0,85	0,33	0,29	0,00	2628,53	876,95	0,2	0,0	33,7	40	3,5	2,0	0	0,00
Alimentação da Cianetação	Solução			50,0%			11,3			0,00	0,00	0,19	0,27	0,00	0,03	0,00	2419,18	875,79	0,4	0,0	197,7	229	3,7	17,1	0	0,00
Descarga Pachuca 1	Solução			50,0%			11,5			23,0	135,1	126,4	3,6	2,0	0,4	0,0	3003,1	1051,1	690,7	364,1	402,9	488	3,7	28,5	0	0,87
Descarga Pachuca 2	Solução			50,0%			11,3			26,8	120,1	141,4	5,7	2,6	0,5	0,0	3003,1	1001,0	579,3	294,1	411,7	537	3,1	63,7	0	1,40
Descarga Pachuca 3	Solução			50,0%			11,2			29,0	118,9	137,6	5,2	2,4	0,4	0,0	2277,4	759,5	493,0	246,5	489,3	541	4,0	74,2	0	1,54
Descarga Pachuca 4	Solução			50,0%			11,4			31,8	122,6	137,6	6,8	2,7	0,6	0,0	2427,5	809,6	493,0	235,2	486,8	541	3,1	84,7	0	1,63
Descarga Pachuca 5	Solução			50,0%			11,4			33,0	125,1	127,6	7,1	2,9	0,6	0,0	2677,8	893,4	533,1	249,0	538,1	541	2,1	79,8	0	1,65
Descarga Pachuca 6	Solução																					0	0	0	0	0
Descarga Pachuca 7	Solução																					0	0	0	0	0
Alimentação Precip. Zinco	Solução																					0	0	0	0	0
Descarga Precip. Zinco	Solução																					0	0	0	0	0
Alimentação CIP	Solução																					0	0	0	0	0
Descarga Tanque 1	Solução																					0	0	0	0	0
Descarga Tanque 2	Solução																					0	0	0	0	0
Descarga Tanque 3	Solução																					0	0	0	0	0
Descarga Tanque 4	Solução																					0	0	0	0	0
Descarga Tanque 5	Solução																					0	0	0	0	0
Eluate	Solução																					0	0	0	0	0

LIXIVIAÇÃO

Tempo de Contato	pH			SIMULA
0		0	744	0,0%
2,92	11,5	59,0%	364,1	60,4%
5,82	11,3	68,6%	294,1	74,3%
8,73	11,2	74,4%	246,5	80,5%
11,63	11,4	81,4%	235,2	84,0%
14,52	11,4	84,6%	249,0	86,3%

Alimentação RECALCULADA	**39,03**
Alimentação ANALISADA	**0,00**
Diferença	**100,0%**

Não é possivel ajustar

Minério	1/(Cs-Ce)
39,03	0,02664
16,01	0,06892
12,26	0,09297
10,00	0,11759
7,25	0,17387
6,00	0,22222

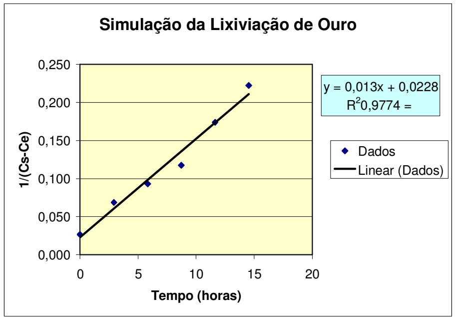
Ce(ajuste 1,5

Do Gráfico

A = 0,013

B = 0,0228

			Minério Simulaçã	1/(Cs-Ce)
			45,36	0,0228
16,0%	23,0	27,41	17,95	0,06079
20,6%	26,8	33,71	11,65	0,09851
20,5%	29,0	36,52	8,84	0,13629
16,6%	31,8	38,11	7,25	0,17397
15,6%	33,0	39,13	6,23	0,21161



CONSUMO DE CIANETO

Tempo	Cu	$Cu(CN)3^{-2}$	CN	Zn	$Zn(CN)_4^{-2}$	CN	S	SCN	CN	Fe	$Fe(CN)_6^{-4}$	CN	Ni	$Ni(CN)_4^{-2}$	CN	Co	$Co(CN)_6^{-3}$	CN	Au	$Au(CN)_2^{-}$	CN	TOTAL	CN_{TRIT}	CN_{TRIT}-Zn	Consum +Livre	CN_{LIVRE}/Cu	CNO	CN-
0,0	0,19	0,00	0	0,00	0,00	0,0	109,1	197,7	88,63	0,00	0,00	0,00	0,27	0	0	0,03	0	0	0,00	0	0	88,6	0,0	0	0			
2,9	126,38	281,62	207,0	135,14	350,04	214,9	222,3	402,9	180,62	1,96	7,44	5,47	3,59	9,92	6,33	0,39	1,41	1,03	23,02	29,10	6,08	639,1	364,1	149,2	788,3	1,18	28,5	17,6
5,8	141,40	315,08	231,6	120,12	311,15	191,0	227,1	411,7	184,54	2,60	9,85	7,25	5,68	15,69	10,01	0,53	1,92	1,39	26,78	33,85	7,07	672,3	294,1	103,0	775,4	0,73	63,7	39,5
8,7	137,64	306,71	225,4	118,87	307,91	189,0	269,9	489,3	219,32	2,40	9,10	6,69	5,18	14,31	9,13	0,44	1,60	1,16	29,03	36,69	7,66	704,4	246,5	57,5	761,8	0,42	74,2	45,9
11,6	137,64	306,71	225,4	122,63	317,63	195,0	268,6	486,8	218,20	2,74	10,37	7,63	6,78	18,74	11,95	0,55	2,01	1,46	31,78	40,17	8,39	720,5	235,2	40,2	760,7	0,29	84,7	52,4
14,5	127,63	284,41	209,0	125,13	324,11	199,0	296,9	538,1	241,20	2,94	11,13	8,19	7,11	19,64	12,53	0,64	2,33	1,69	33,03	41,75	8,72	729,7	249,0	50,0	779,8	0,39	79,8	49,4
0,0	0,00	0,00	0,0	0,00	0,00	0,0	0,0	0,0	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,0	0,0	0,0	0,0			
CONSUMO DE CIANETO			**27,04%**			**25,74%**			**31,20%**			**1,06%**			**1,62%**			**0,22%**			**1,13%**	**693,2**		**6,47%**	**773,2**	**MEDIA**	CNO	42
	$Cu(CN)_3^{-2}$	141,5		$Zn(CN)_4^{-2}$	169,4		SCN	58		$Fe(CN)_6^{-4}$	212		$Ni(CN)_4^{-2}$	163		$Co(CN)_6^{-3}$	215		$Au(CN)_2^{-}$	249		**37,3**			**-3,9%**	**Diferença**	CN	26
	CN	104		CN	104		CN	26		CN	156		CN	104		CN	156		CN	52				94,46%	**11,8**	**DESVPAD**		
	Cu	63,5		Zn	65,4		S	32		Fe	56		Ni	59		Co	59		Au	197				Conferencia				
										FeS_2	120																	
										FeS	88																	

SÉRIE FINAL
DE TESTES DE LABORATÓRIO

LABORATORY TESTWORK - Summary Tables

Gold in Solution

Time	Test number LB																													
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
1	10,9	16,5	13,6	20,5	21,7	16,7	16,8	15,7	18,1	18,4	21,0	19,1	14,6	20,2	21,0	9,7	15,3	15,8	10,8	23,7	20,6	17,6	15,6	14,5	17,4	22,1	0,8	0,9	16,7	13,5
2	16,8	21,8	18,7	24,1	24,8	21,9	17,7	21,6	22,6	22,7	23,7	22,3	15,2	23,5	23,9	12,3	20,6	29,5	12,7	24,4	19,9	22,5	18,9	21,7	22,3	22,3	0,2	0,3	20,1	19,0
6	20,3	24,8	24,3	16,8	24,9	27,3	21,9	26,2	25,0	25,5	25,9	23,9	24,1	26,1	25,8	19,2	24,8	29,8	18,4	26,4	24,7	25,1	21,8	26,0	25,3	24,0	0,0	0,0	24,6	24,8
8	20,8	25,2	24,4	24,9	25,9	27,8	23,9	27,1	25,5	26,0	27,2	26,8	24,2	27,5	27,1	21,0	26,7	30,3	20,7	26,5	26,9	25,5	23,7	26,3	26,5	25,8	0,0	0,0	24,6	25,1

Cooper in Solution

Time	Test number LB																													
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
0	2,5	3,5	0,0	0,1	2,3	3,8	0,3	0,0	0,3	0,0	0,6	4,5	0,2	0,0	0,1	0,1	0,2	2,3	0,1	0,2	2,4	0,2	0,0	0,2	0,0	1,0	1,7	0,1	0,1	1,7
1	55,1	64,0	64,8	54,1	44,1	53,1	62,2	54,2	47,0	46,6	42,9	48,7	46,2	59,7	46,1	58,4	51,8	60,3	69,0	74,6	73,8	48,9	70,7	45,3	85,8	48,7	51,7	53,9	65,0	59,6
2	81,8	74,0	90,1	89,9	60,9	55,9	64,0	55,1	64,8	55,6	49,1	77,8	28,1	61,6	47,8	86,6	63,1	62,3	56,7	79,3	74,7	75,2	73,3	56,6	119,1	63,4	61,1	58,8	84,7	77,2
6	90,9	85,9	94,8	82,6	62,8	65,2	32,0	64,5	65,8	67,2	64,3	91,9	85,1	75,9	61,7	73,4	65,0	112,9	79,4	99,1	80,3	84,6	68,0	69,8	124,8	83,7	67,7	71,5	72,5	81,9
8	91,9	85,9	91,0	124,8	76,2	67,1	28,3	98,1	112,8	72,6	66,1	91,9	93,3	78,7	70,4	88,5	98,9	121,7	90,8	129,3	93,4	96,8	134,2	84,9	85,8	84,6	71,4	76,4	84,7	85,6

Zinc in Solution

Time	Test number LB																													
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
0	0,7	0,1	0,7	0,0	0,4	0,3	0,7	0,0	0,0	0,0	0,1	0,3	0,0	0,0	0,0	0,0	0,0	0,4	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	1,1	0,0	0,0	2,1
1	11,8	66,7	32,9	41,3	47,5	76,4	26,5	104,6	39,5	52,9	63,4	47,8	46,2	59,7	72,1	34,8	61,2	61,3	62,4	38,7	66,3	55,5	70,7	61,3	73,4	78,2	37,6	44,1	63,1	67,9
2	31,8	71,3	36,6	86,2	60,9	75,5	29,3	78,5	63,9	54,7	66,1	72,2	16,0	60,7	68,7	71,6	81,9	64,2	60,5	71,7	64,5	83,7	65,3	75,5	100,1	60,7	37,6	53,9	53,7	109,8
6	32,2	77,7	39,4	80,7	61,9	83,9	85,9	79,4	19,7	56,5	65,2	94,7	77,9	61,6	58,2	48,0	69,7	103,2	47,1	83,1	72,9	75,2	43,8	90,6	102,0	65,3	49,8	53,9	54,6	89,3
8	33,1	87,7	86,4	94,5	78,1	84,8	93,2	104,6	82,7	63,6	74,2	94,7	67,0	67,3	73,9	43,3	117,7	112,9	49,2	102,9	80,3	86,5	115,4	110,4	66,7	88,3	50,8	55,9	55,5	95,8

SCN in Solution

Time	Test number LB																													
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
1	251,7	232,2	285,3	214,7	243,8	128,6	115,2	234,5	205,8	252,7	86,7	210,9	171,2	335,7	108,7	237,2	186,4	258,9	180,6	209,6	231,7	240,7	247,8	187,7	212,5	249,2	139,1	187,2	256,1	199,1
2	353,7	275,1	287,2	217,4	390,4	153,7	144,4	295,2	256,6	314,5	208,2	271,9	177,5	409,7	382,5	258,9	194,0	203,4	191,0	292,6	268,1	319,7	302,3	249,1	301,1	266,7	160,7	201,9	283,4	201,0
6	220,5	315,3	355,7	291,7	348,5	331,7	219,4	339,1	313,9	375,5	352,9	400,3	388,5	417,3	391,2	313,5	276,8	294,9	311,1	319,1	458,7	321,5	343,5	255,7	342,1	311,7	218,1	212,6	390,7	252,2
8	499,7	345,5	505,9	354,1	427,6	385,7	254,1	294,2	380,6	399,7	360,0	435,9	332,3	437,2	243,4	330,5	319,2	522,7	357,4	341,7	451,3	436,2	361,4	328,3	335,5	430,4	173,9	229,3	438,7	240,1

Sulfate in Solution

Time	Test number LB																													
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
0	2315	1536	1577	1505	1847	1398	896	2354	1654	1738	1465	1950	1811	1840	1530	1657	1958	1830	1891	1964	1756	1824	1503	1774	1982	1674	1410	1646	1657	1991
1	2425	2249	2422	2257	2343	1938	1719	2896	1823	2312	1733	2719	2173	2143	2086	2259	2316	2511	2193	2133	2410	2369	2200	2264	2287	2372	1579	1842	2184	2103
2	2425	2249	2478	2073	2343	2162	1664	2466	2312	2312	2144	2531	1594	2030	1964	1958	2429	3387	2080	2322	2709	2313	2415	2547	2154	2483	1654	1901	2260	2457
6	3013	2706	2478	2202	2343	2814	1773	2410	1955	2491	2019	2531	2626	2750	2138	2128	2316	3251	2269	2436	2952	2482	2934	2264	2154	2483	1579	2038	2617	2401
8	3013	2706	2666	2202	2285	2758	2194	2597	2425	2312	2198	2419	2391	2940	2243	2429	2844	3563	2496	2436	3718	2482	2826	2924	2287	2722	1654	1646	2542	2457

Gold Recovery (%)

Time	Test number LB																													
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
0	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%	0,0%
1	38,2%	59,3%	44,0%	70,1%	72,6%	55,3%	50,0%	53,0%	66,1%	66,3%	72,3%	63,7%	50,6%	68,8%	71,5%	32,2%	53,5%	42,9%	37,6%	83,1%	67,9%	56,1%	53,8%	51,2%	60,9%	77,8%	2,9%	3,0%	61,8%	49,4%
2	58,8%	77,9%	60,3%	82,6%	82,8%	72,5%	52,8%	72,9%	82,2%	81,8%	81,6%	74,3%	52,7%	80,1%	81,2%	40,9%	71,8%	80,2%	44,2%	85,4%	65,4%	71,4%	65,3%	76,5%	77,8%	78,7%	0,7%	1,0%	74,6%	69,5%
6	71,0%	88,7%	78,5%	57,5%	83,4%	90,4%	65,3%	88,7%	91,1%	92,1%	89,3%	79,7%	83,5%	88,8%	87,7%	63,7%	86,3%	81,0%	64,4%	92,7%	81,1%	80,0%	75,5%	91,8%	88,5%	84,6%	0,1%	0,1%	91,1%	90,6%
8	72,6%	90,4%	78,8%	85,1%	86,6%	92,0%	71,0%	91,5%	92,9%	93,7%	93,9%	89,3%	83,9%	93,7%	92,1%	69,7%	93,1%	82,3%	72,2%	93,1%	88,5%	81,2%	82,0%	92,8%	92,5%	91,0%	0,1%	0,1%	91,1%	91,9%

NaCN$_{TRIT}$ in Solution

Time	Test number LB																													
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
0	2722	2733	2717	2782	2683	2737	2789	2729	2714	2845	2854	2720	2815	2688	2933	1623	2709	5137	1638	2702	5352	2711	2850	2701	2677	2772	2713	2601	2709	2739
1	994	1333	1507	1476	1706	1979	2230	1290	1394	1644	1829	1173	1145	1606	2000	898	1486	4213	593	1601	3431	1354	1599	1505	1249	1569	1556	1476	1245	1354
2	629	1105	1386	1271	1525	1819	2177	1141	1211	1606	1663	1089	685	1522	1729	714	1202	3132	467	1282	2648	1169	1424	1392	1563	1382	1266	1122	1175	1194
6	322	653	1026	1036	1188	1331	2040	919	994	1139	1121	817	761	1062	1207	488	917	1940	331	1073	2256	742	942	917	1055	953	638	708	759	725
8	196	362	872	957	1043	1162	2235	638	808	951	953	578	721	947	925	497	712	1774	390	951	2013	812	765	712	942	738	557	548	723	637

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-001**
Sample:	**Calcine**
Description:	**pH INFLUENCE**
Date:	**17/03/2004**

Teste nº		**%**	**Pré-Lime**				**Cianetação**			**CIP**
		Sólidos	**(h)**	**pH**	**$Pb(NO_3)_2$ (gpt)**	**CaO (kg/t)**	**(h)**	**pH**	**NaCN (kg/t)**	**Carvão (g/L)**
LBII	**1**	50	2	9,5	0	0,0	8	11,0	2,5	0

Real Conditions										
WT% solids	**Pre-lime (h)**	**NaCN (g)**	**NaCN (kg/t)**	**NaCN (mg/L)**	**CN (mg/L)**	**CaO (g)**	**CaO (kg/t)**	**pH**	**$Pb(NO_3)_2$ (mg)**	**$Pb(NO_3)_2$ (g/t)**
52,1%	2	7,53	2,50	2721,55	1444,09		0,00	11	0,00	0,00

Samples	**Solids**	**Volume**	**Pulp**	**Assays Solution**					**Assays Solids**		
Name	**Sólidos (g)**	**Solution (mL)**	**(g)**	**Au (mg/L)**	**Cu (mg/L)**	**Zn (mg/L)**	**SCN (mg/L)**	**$SO_4^=$ (mg/L)**	**Au (gpt)**	**Cu (g/t)**	**Zn (gpt)**
Feed Anal.	3010,00	2764,97	5774,97						29,57		
0 hours			766,12	0,0	2,7	0,8	0,0	2520,0			
1 hours			780,66	11,9	60,0	12,8	274,0	2640,0			
2 hours			808,89	18,3	89,0	34,6	385,0	2640,0			
6 hours			795,20	22,1	99,0	35,0	240,0	3280,0			
8 hours			2624,10	22,6	100,0	36,0	544,0	3280,0			
Tailing									7,83	643	1715

Calculation to 50% Solids

Samples	**Solids**	**Volume**	**Recovery**	**Assays Solution**					**Assays Solids**		
Name	**Sólidos (g)**	**Solution (mL)**	**Au (%)**	**Au (mg/L)**	**Cu (mg/L)**	**Zn (mg/L)**	**SCN (mg/L)**	**$SO_4^=$ (mg/L)**	**Au (gpt)**	**Cu (g/t)**	**Zn (gpt)**
Feed Calc.	3010,00	3010,00							28,59	734,9	1748,1
0 hours			0,0%	0,0	2,5	0,7	0,0	2314,9			
1 hours			38,2%	10,9	55,1	11,8	251,7	2425,1			
2 hours			58,8%	16,8	81,8	31,8	353,7	2425,1			
6 hours			71,0%	20,3	90,9	32,2	220,5	3013,0			
8 hours			72,6%	20,8	91,9	33,1	499,7	3013,0			
Tailing									7,83	643	1715

Samples	**DO**	**pH**	**$NaCN_{TRIT}$**	**CN^-_{TRIT}**	**Associated**					Calculated	Analyzed
Name	**(mg/L)**		**(mg/L)**	**(mg/L)**	**$Cu(CN)_3^{-4}$**	**$Zn(CN)_4^{-2}$**	**SCN^-**	**CN_{ASSOC}**	**% added**	**CN_{TOTAL}**	**CN_{TOTAL}**
Pre-Lime		9,5									
0 hours	5,3	10,15	2721,55	1444,09	0,00	0,00	0,00	0,00			
1 hours		9,85	994,4	528,0	90,27	18,70	112,83	221,79	15,4%	636,97	
2 hours	5,8	9,86	629,0	334,0	133,90	50,54	158,54	342,98	23,8%	518,44	
6 hours	9,6	9,73	322,0	171,0	148,94	51,13	98,83	298,90	20,7%	371,07	
8 hours	8,6	9,53	195,9	104,0	150,45	52,59	224,01	427,04	29,6%	307,03	382,00
	NaCN consumption (kg/t)		2,3								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-002**
Sample: **Calcine**
Description: **pH INFLUENCE**
Date: **18/03/2004**

Teste nº		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	002	50	2	10,5	0	6,4	8	10,5	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
52,2%	2	7,70	2,50	2732,63	1449,97	19,82	6,43	11	0,00	0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3082,80	2817,80	5900,60						29,57		
0 hours			795,90	0,0	3,9	0,1	0,0	1680,0			
1 hours			803,20	18,1	70,0	73,0	254,0	2460,0			
2 hours			813,20	23,8	81,0	78,0	301,0	2460,0			
6 hours			780,80	27,1	94,0	85,0	345,0	2960,0			
8 hours			2707,50	27,6	94,0	96,0	378,0	2960,0			
Tailing									2,69	657	1700

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3082,80	3082,80							27,92	742,9	1787,7
0 hours			0,0%	0,0	3,5	0,1	0,0	1535,6			
1 hours			59,3%	16,5	64,0	66,7	232,2	2248,5			
2 hours			77,9%	21,8	74,0	71,3	275,1	2248,5			
6 hours			88,7%	24,8	85,9	77,7	315,3	2705,6			
8 hours			90,4%	25,2	85,9	87,7	345,5	2705,6			
Tailing									2,69	657	1700

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime		10,51									
0 hours	7,0	10,6	2732,63	1449,97	0,00	0,00	0,00	0,00			
1 hours	6,5	10,5	1333,3	708,0	104,79	106,11	104,07	314,97	21,7%	918,90	
2 hours	8,3	10,4	1105,5	587,0	121,26	113,37	123,33	357,96	24,7%	821,63	
6 hours	8,2	10,5	653,5	347,0	140,72	123,55	141,36	405,63	28,0%	611,27	
8 hours	5,7	10,5	361,6	192,0	140,72	139,54	154,88	435,14	30,0%	472,26	590,00
	NaCN consumption (kg/t)		2,2								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-003**
Sample: **Calcine**
Description: **pH INFLUENCE - STANDARD TEST**
Date: **23/03/2004**

Teste nº		% Sólidos	Pré-Lime (h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	Cianetação (h)	pH	NaCN (kg/t)	CIP Carvão (g/L)
LBII	**003**	50	2	11,0	0	7,3	8	11,0	2,5	0

Real Conditions

WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,6%	2	7,65	2,55	2716,72	1441,52	21,97	7,32	11	0,00	0,00

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3000,00	2815,90	5815,90						29,57		
0 hours			793,60	0,0	0,1	0,7	0,0	1680,0			
1 hours			808,90	14,5	69,0	35,0	304,0	2580,0			
2 hours			789,60	19,9	96,0	39,0	306,0	2640,0			
6 hours			815,70	25,9	101,0	42,0	379,0	2640,0			
8 hours			2608,10	26,0	97,0	92,0	539,0	2840,0			
Tailing									6,55	600	1524

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3000,00	3000,00							30,95	691,0	1610,4
0 hours			0,0%	0,0	0,0	0,7	0,0	1576,9			
1 hours			44,0%	13,6	64,8	32,9	285,3	2421,7			
2 hours			60,3%	18,7	90,1	36,6	287,2	2478,0			
6 hours			78,5%	24,3	94,8	39,4	355,7	2478,0			
8 hours			78,8%	24,4	91,0	86,4	505,9	2665,7			
Tailing									6,55	600	1524

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated $Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
Pre-Lime	5,11	11,1									
0 hours	5,4	10,6	2716,72	1441,52	0,00	0,00	0,00	0,00			
1 hours	5,5	10,7	1506,6	800,0	106,07	52,24	127,91	286,23	19,9%	958,31	
2 hours	6,28	10,8	1386,1	736,0	147,58	58,21	128,75	334,55	23,2%	941,79	
6 hours	6,8	10,7	1026,4	545,0	155,27	62,69	159,47	377,43	26,2%	762,96	
8 hours	7,6	11,0	871,9	463,0	149,12	137,32	226,79	513,23	35,6%	749,44	1009,00
	NaCN consumption (kg/t)		1,7								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-004**
Sample: **Calcine**
Description: **LIME ADDITION INFLUENCE**
Date: **23/03/2004**

Teste nº		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	004	50	2	11,0	0	7,0	8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
52,2%	2	7,71	2,55	2782,19	1476,26	21,26	7,04	11	0,00	0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^{=}$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3020,70	2771,20	5791,90						29,57		
0 hours			800,40	0,0	0,1	0,0	0,0	1640,0			
1 hours			800,50	22,3	59,0	45,0	234,0	2460,0			
2 hours			765,00	26,3	98,0	94,0	237,0	2260,0			
6 hours			814,00	18,3	90,0	88,0	318,0	2400,0			
8 hours			2612,00	27,1	136,0	103,0	386,0	2400,0			
Tailing									4,34	539	1453

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^{=}$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3020,70	3020,70							29,20	663,8	1547,5
0 hours			0,0%	0,0	0,1	0,0	0,0	1504,5			
1 hours			70,1%	20,5	54,1	41,3	214,7	2256,8			
2 hours			82,6%	24,1	89,9	86,2	217,4	2073,3			
6 hours			57,5%	16,8	82,6	80,7	291,7	2201,8			
8 hours			85,1%	24,9	124,8	94,5	354,1	2201,8			
Tailing									4,34	539	1453

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	4,4	10,52									
0 hours	5,6	10,0	2782,19	1476,26	0,00	0,00	0,00	0,00			
1 hours	5,7	10,5	1476,5	784,0	88,65	65,65	96,23	250,53	17,0%	938,30	
2 hours	5,7	10,7	1271,2	675,0	147,25	137,13	97,47	381,85	25,9%	959,38	
6 hours	6,2	10,8	1035,8	550,0	135,23	128,38	130,78	394,38	26,7%	813,61	
8 hours	6,5	10,7	956,7	508,0	204,34	150,26	158,74	513,35	34,8%	862,61	730,00
	NaCN consumption (kg/t)		1,7								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-005**
Sample:	**Calcine**
Description:	**LIME ADDITION INFLUENCE - 1.5 times the normal addition**
Date:	**23/03/2004**

Teste nº		% Sólidos	Pré-Lime (h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	Cianetação (h)	pH	NaCN (kg/t)	CIP Carvão (g/L)
LBII	**005**	50	2		0	1.5 Det	8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,2%	2	7,71	2,55	2682,76	1423,50	32,16	10,66	11	0,00	0,00

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3018,00	2873,91	5891,91						29,57		
0 hours			780,95	0,0	2,4	0,4	0,0	1940,0			
1 hours			748,89	22,8	46,3	49,9	256,0	2460,0			
2 hours			809,99	26,0	64,0	64,0	410,0	2460,0			
6 hours			811,08	26,2	66,0	65,0	366,0	2460,0			
8 hours			2741,00	27,2	80,0	82,0	449,0	2400,0			
Tailing									4,00	631	1530

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3018,00	3018,00							29,90	707,2	1608,1
0 hours			0,0%	0,0	2,3	0,4	0,0	1847,4			
1 hours			72,6%	21,7	44,1	47,5	243,8	2342,6			
2 hours			82,8%	24,8	60,9	60,9	390,4	2342,6			
6 hours			83,4%	24,9	62,8	61,9	348,5	2342,6			
8 hours			86,6%	25,9	76,2	78,1	427,6	2285,4			
Tailing									4,00	631	1530

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated $Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
Pre-Lime	4,6	12,71									
0 hours		11,1	2682,76	1423,50	0,00	0,00	0,00	0,00			
1 hours		10,8	1706,2	906,0	72,21	75,56	109,28	257,05	18,1%	1053,77	
2 hours	9,4	10,9	1525,4	810,0	99,81	96,91	175,02	371,75	26,1%	1006,73	
6 hours	8,9	10,7	1188,3	631,0	102,93	98,43	156,24	357,60	25,1%	832,36	
8 hours	8	10,8	1043,3	554,0	124,77	124,17	191,67	440,61	31,0%	802,94	853,00
	NaCN consumption (kg/t)		1,6								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-006**
Sample: **Calcine**
Description: **LIME ADDITION INFLUENCE - 2.0 times the normal addition**
Date: **23/03/2004**

Teste nº		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	**006**	50	2		0	2.0 Det	8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,8%	2	7,66	2,55	2737,18	1452,38	48,94	16,29	11	0,00	0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3003,60	2798,50	5802,10						29,57		
0 hours			822,00	0,0	4,1	0,3	0,0	1500,0			
1 hours			786,20	17,9	57,0	82,0	138,0	2080,0			
2 hours			774,50	23,5	60,0	81,0	165,0	2320,0			
6 hours			805,60	29,3	70,0	90,0	356,0	3020,0			
8 hours			2613,80	29,8	72,0	91,0	414,0	2960,0			
Tailing									2,42	665	1677

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3003,60	3003,60							30,19	732,1	1761,8
0 hours			0,0%	0,0	3,8	0,3	0,0	1397,6			
1 hours			55,3%	16,7	53,1	76,4	128,6	1938,0			
2 hours			72,5%	21,9	55,9	75,5	153,7	2161,6			
6 hours			90,4%	27,3	65,2	83,9	331,7	2813,8			
8 hours			92,0%	27,8	67,1	84,8	385,7	2757,9			
Tailing									2,42	665	1677

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	4,5	12,8									
0 hours	5,3	11,5	2737,18	1452,38	0,00	0,00	0,00	0,00			
1 hours	6,5	11,4	1979,3	1051,0	86,98	121,49	57,64	266,11	18,3%	1259,47	
2 hours	6,3	11,2	1819,2	966,0	91,56	120,01	68,91	280,48	19,3%	1177,57	
6 hours	5,97	11,0	1331,5	707,0	106,82	133,35	148,69	388,85	26,8%	947,16	
8 hours	5,6	11,0	1162,0	617,0	109,87	134,83	172,91	417,61	28,8%	861,70	1004,00
	NaCN consumption (kg/t)		1,5								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-007**
Sample: **Calcine**
Description: **LIME ADDITION INFLUENCE - 3.0 times the normal addition**
Date: **23/03/2004**

Teste nº		% Sólidos	Pré-Lime				Cianetação			CIP
			(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	**007**	50	2		0	3.0 Det	8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
52,2%	2	7,65	2,55	2789,26	1480,02	57,10	19,03	11	0,00	0,00

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution					Assays Solids		
				Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3000,20	2742,66	5742,86						29,57		
0 hours			793,10	0,0	0,4	0,8	0,0	980,0			
1 hours			848,90	18,4	68,0	29,0	126,0	1880,0			
2 hours			821,76	19,4	70,0	32,0	158,0	1820,0			
6 hours			854,40	24,0	35,0	94,0	240,0	1940,0			
8 hours			2424,70	26,1	31,0	102,0	278,0	2400,0			
Tailing									9,76	665	1677

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution					Assays Solids		
				Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3000,20	3000,20							33,62	693,3	1770,2
0 hours			0,0%	0,0	0,3	0,7	0,0	895,9			
1 hours			50,0%	16,8	62,2	26,5	115,2	1718,6			
2 hours			52,8%	17,7	64,0	29,3	144,4	1663,8			
6 hours			65,3%	21,9	32,0	85,9	219,4	1773,5			
8 hours			71,0%	23,9	28,3	93,2	254,1	2194,0			
Tailing									9,76	665	1677

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated					Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
					$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added		
Pre-Lime	4,8	13,05									
0 hours	5,6	13,0	2789,26	1480,02	0,00	0,00	0,00	0,00			
1 hours	7,2	12,7	2229,8	1184,0	101,81	42,16	51,63	195,60	13,2%	1327,97	
2 hours	7,15	12,5	2177,0	1156,0	104,80	46,52	64,75	216,07	14,6%	1307,32	
6 hours	6,8	12,7	2039,5	1083,0	52,40	136,65	98,35	287,40	19,4%	1272,05	
8 hours	7,3	12,6	2235,4	1187,0	46,41	148,28	113,92	308,61	20,9%	1381,69	1705,00
	NaCN consumption (kg/t)		0,5								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-008**
Sample:	**Calcine**
Description:	**PRE-LIME INFLUENCE - 0 hours**
Date:	**23/03/2004**

Teste nº		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	Pb(NO$_3$)$_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	008	50	0	11,0	0		8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	Pb(NO$_3$)$_2$ (mg)	Pb(NO$_3$)$_2$ (g/t)
51,7%	0	7,73	2,55	2728,85	1447,96	19,12	6,30	11	0,00	0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	SO$_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3032,60	2832,70	5865,30						29,57		
0 hours			796,20	0,0	0,1	0,0	0,0	2520,0			
1 hours			792,40	16,8	58,0	112,0	251,0	3100,0			
2 hours			811,30	23,1	59,0	84,0	316,0	2640,0			
6 hours			1226,70	28,1	69,0	85,0	363,0	2580,0			
8 hours			2238,70	29,0	105,0	112,0	315,0	2780,0			
Tailing									2,51	581	1491

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	SO$_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3032,60	3032,60							29,60	679,1	1595,6
0 hours			0,0%	0,0	0,0	0,0	0,0	2353,9			
1 hours			53,0%	15,7	54,2	104,6	234,5	2895,7			
2 hours			72,9%	21,6	55,1	78,5	295,2	2466,0			
6 hours			88,7%	26,2	64,5	79,4	339,1	2409,9			
8 hours			91,5%	27,1	98,1	104,6	294,2	2596,8			
Tailing									2,51	581	1491

Samples	DO	pH	NaCN$_{TRIT}$	CN$^-_{TRIT}$	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	Cu(CN)$_3^{-4}$	Zn(CN)$_4^{-2}$	SCN$^-$	CN$_{ASSOC}$	% added	CN$_{TOTAL}$	CN$_{TOTAL}$
Pre-Lime											
0 hours	4,1	11,0	2728,85	1447,96	0,00	0,00	0,00	0,00			
1 hours	6,4	11,0	1290,0	685,0	88,73	166,36	105,10	360,19	24,9%	940,09	
2 hours	6,3	11,0	1141,2	606,0	90,26	124,77	132,32	347,35	24,0%	821,03	
6 hours	6,6	11,0	919,0	488,0	105,56	126,26	152,00	383,81	26,5%	719,82	
8 hours	6,4	11,0	638,4	339,0	160,63	166,36	131,90	458,90	31,7%	666,00	590,00
	NaCN consumption (kg/t)		2,0								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-009
Sample: Calcine
Description: PRE-LIME INFLUENCE - 2 hours
Date: 25/03/2004

Teste nº		% Sólidos	Pré-Lime (h)	Pré-Lime pH	Pré-Lime Pb(NO$_3$)$_2$ (gpt)	Pré-Lime CaO (kg/t)	Cianetação (h)	Cianetação pH	Cianetação NaCN (kg/t)	CIP Carvão (g/L)
LBII	009	50	2	11,0	0		8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	Pb(NO$_3$)$_2$ (mg)	Pb(NO$_3$)$_2$ (g/t)
51,6%	2	7,66	2,55	2714,29	1440,24	21,97	7,32	11	0,00	0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	SO$_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3003,00	2822,10	5825,10						29,57		
0 hours			820,00	0,0	0,4	0,0	0,0	1760,0			
1 hours			794,40	19,3	50,0	42,0	219,0	1940,0			
2 hours			826,10	24,0	69,0	68,0	273,0	2460,0			
6 hours			830,60	24,0	70,0	21,0	334,0	2080,0			
8 hours			2554,00	27,1	120,0	88,0	405,0	2580,0			
Tailing									1,96	537	1491

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	SO$_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3003,00	3003,00							27,43	649,8	1573,7
0 hours			0,0%	0,0	0,3	0,0	0,0	1654,0			
1 hours			66,1%	18,1	47,0	39,5	205,8	1823,1			
2 hours			82,2%	22,6	64,8	63,9	256,6	2311,8			
6 hours			91,1%	25,0	65,8	19,7	313,9	1954,7			
8 hours			92,9%	25,5	112,8	82,7	380,6	2424,6			
Tailing									1,96	537	1491

Samples	DO	pH	NaCN$_{TRIT}$	CN$^-_{TRIT}$	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	Cu(CN)$_3^{-4}$	Zn(CN)$_4^{-2}$	SCN$^-$	CN$_{ASSOC}$	% added	CN$_{TOTAL}$	CN$_{TOTAL}$
Pre-Lime	6,19	11,0									
0 hours	6,5	11,0	2714,29	1440,24	0,00	0,00	0,00	0,00			
1 hours	6,5	11,0	1393,6	740,0	76,96	62,77	92,26	231,98	16,1%	879,72	
2 hours	6,4	11,0	1210,9	643,0	106,20	101,62	115,01	322,83	22,4%	850,82	
6 hours	6,8	10,7	994,4	528,0	107,74	31,38	140,70	279,83	19,4%	667,12	
8 hours	6,5	10,6	807,9	429,0	184,70	131,51	170,62	486,82	33,8%	745,21	679,00
	NaCN consumption (kg/t)		1,8								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-010
Sample: Calcine
Description: PRE-LIME INFLUENCE - 6 hours
Date: 25/03/2004

Teste nº		% Sólidos	Pré-Lime (h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	Cianetação (h)	pH	NaCN (kg/t)	CIP Carvão (g/L)
LBII	**010**	50	6	11,0	0		8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
52,7%	6	7,69	2,55	2845,43	1509,82	25,36	8,41	11	0,00	0,00

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3015,90	2702,58	5718,48						29,57		
0 hours			815,78	0,0	0,0	0,0	0,0	1940,0			
1 hours			834,60	20,5	52,0	59,0	282,0	2580,0			
2 hours			807,20	25,3	62,0	61,0	351,0	2580,0			
6 hours			810,90	28,5	75,0	63,0	419,0	2780,0			
8 hours			2450,00	29,0	81,0	71,0	446,0	2580,0			
Tailing									1,74	558	1546

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3015,90	3015,90							27,73	630,6	1609,6
0 hours			0,0%	0,0	0,0	0,0	0,0	1738,5			
1 hours			66,3%	18,4	46,6	52,9	252,7	2312,0			
2 hours			81,8%	22,7	55,6	54,7	314,5	2312,0			
6 hours			92,1%	25,5	67,2	56,5	375,5	2491,2			
8 hours			93,7%	26,0	72,6	63,6	399,7	2312,0			
Tailing									1,74	558	1546

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated $Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
Pre-Lime	6,19	11,0									
0 hours	6,5	11,0	2845,43	1509,82	0,00	0,00	0,00	0,00			
1 hours	6,5	11,0	1644,1	873,0	76,32	84,08	113,28	273,67	18,1%	1033,39	
2 hours	6,4	11,0	1606,4	853,0	90,99	86,93	141,00	318,92	21,1%	1030,92	
6 hours	6,8	10,7	1139,4	605,0	110,07	89,78	168,31	368,16	24,4%	804,85	
8 hours	6,5	10,6	951,0	505,0	118,88	101,18	179,16	399,22	26,4%	725,05	853,00
	NaCN consumption (kg/t)		1,7								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-011
Sample: Calcine
Description: PRE-LIME INFLUENCE - 12 hours
Date: 25/03/2004

Teste nº		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	**011**	50	12	11,0	0		8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
52,8%	12	7,68	2,55	2854,17	1514,46	28,85	9,58	11	0,00	0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3011,80	2690,80	5702,60						29,57		
0 hours			829,70	0,0	0,7	0,1	0,0	1640,0			
1 hours			813,50	23,5	48,0	71,0	97,0	1940,0			
2 hours			821,30	26,5	55,0	74,0	233,0	2400,0			
6 hours			786,80	29,0	72,0	73,0	395,0	2260,0			
8 hours			2451,30	30,5	74,0	83,0	403,0	2460,0			
Tailing									1,78	536	1436

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3011,80	3011,80							29,03	602,1	1510,2
0 hours			0,0%	0,0	0,6	0,1	0,0	1465,2			
1 hours			72,3%	21,0	42,9	63,4	86,7	1733,2			
2 hours			81,6%	23,7	49,1	66,1	208,2	2144,2			
6 hours			89,3%	25,9	64,3	65,2	352,9	2019,1			
8 hours			93,9%	27,2	66,1	74,2	360,0	2197,8			
Tailing									1,78	536	1436

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	4,3	11,0									
0 hours	6,8	11,0	2854,17	1514,46	0,00	0,00	0,00	0,00			
1 hours	6,8	10,9	1828,6	971,0	70,24	100,87	38,85	209,96	13,9%	1142,11	
2 hours	7,0	10,9	1662,9	883,0	80,48	105,13	93,32	278,93	18,4%	1068,61	
6 hours	7,3	10,8	1120,5	595,0	105,35	103,71	158,20	367,26	24,3%	804,07	
8 hours	6,4	11,0	952,9	506,0	108,28	117,92	161,40	387,60	25,6%	732,20	788,00
	NaCN consumption (kg/t)		1,7								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-012
Sample: Calcine
Description: PRE-LIME INFLUENCE - 0 hours WITH 100 gpt OF $Pb(NO_3)_2$
Date: 25/03/2004

Teste nº		% Sólidos	Pré-Lime (h)	Pré-Lime pH	Pré-Lime $Pb(NO_3)_2$ (gpt)	Pré-Lime CaO (kg/t)	Cianetação (h)	Cianetação pH	Cianetação NaCN (kg/t)	CIP Carvão (g/L)
LBII	012	50	0	11,0	100		8	11,0	2,5	0

Real Conditions

WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,6%	0	7,66	2,55	2719,66	1443,08	19,76	6,58	11	300,45	100,00

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3004,50	2816,53	5821,03						29,57		
0 hours			753,78	0,0	4,8	0,3	0,0	2080,0			
1 hours			783,84	20,4	52,0	51,0	225,0	2900,0			
2 hours			780,50	23,8	83,0	77,0	290,0	2700,0			
6 hours			773,51	25,5	98,0	101,0	427,0	2700,0			
8 hours			2729,40	28,6	98,0	101,0	465,0	2580,0			
Tailing									3,20	660	1625

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3004,50	3004,50							30,01	751,9	1719,7
0 hours			0,0%	0,0	4,5	0,3	0,0	1949,9			
1 hours			63,7%	19,1	48,7	47,8	210,9	2718,6			
2 hours			74,3%	22,3	77,8	72,2	271,9	2531,1			
6 hours			79,7%	23,9	91,9	94,7	400,3	2531,1			
8 hours			89,3%	26,8	91,9	94,7	435,9	2418,6			
Tailing									3,20	660	1625

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated $Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
Pre-Lime											
0 hours	3,6	10,7	2719,66	1443,08	0,00	0,00	0,00	0,00			
1 hours	6,6	10,3	1173,3	623,0	79,84	76,03	94,55	250,42	17,4%	778,86	
2 hours	6,5	10,7	1088,5	578,0	127,43	114,79	121,87	364,09	25,2%	820,22	
6 hours	6,0	10,7	817,3	434,0	150,46	150,56	179,44	480,46	33,3%	735,03	
8 hours	6,1	10,5	578,2	307,0	150,46	150,56	195,41	496,43	34,4%	608,03	633,00
	NaCN consumption (kg/t)		2,0								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-013**
Sample:	**Calcine**
Description:	**PRE-LIME INFLUENCE - 2 hours WITH 100 gpt OF $Pb(NO_3)_2$**
Date:	**25/03/2004**

Teste nº		**% Sólidos**	**Pré-Lime (h)**	**pH**	**$Pb(NO_3)_2$ (gpt)**	**CaO (kg/t)**	**Cianetação (h)**	**pH**	**NaCN (kg/t)**	**CIP Carvão (g/L)**
LBII	**013**	50	2	11,0	100		8	11,0	2,5	0

Real Conditions										
WT% solids	**Pre-lime (h)**	**NaCN (g)**	**NaCN (kg/t)**	**NaCN (mg/L)**	**CN (mg/L)**	**CaO (g)**	**CaO (kg/t)**	**pH**	**$Pb(NO_3)_2$ (mg)**	**$Pb(NO_3)_2$ (g/t)**
52,5%	2	7,71	2,55	2814,79	1493,56	21,46	7,09	11	302,00	99,84

Samples	**Solids**	**Volume**	**Pulp**	**Assays Solution**					**Assays Solids**		
Name	**Sólidos (g)**	**Solution (mL)**	**(g)**	**Au (mg/L)**	**Cu (mg/L)**	**Zn (mg/L)**	**SCN (mg/L)**	**$SO_4^=$ (mg/L)**	**Au (gpt)**	**Cu (g/t)**	**Zn (gpt)**
Feed Anal.	3024,70	2739,10	5763,80						29,57		
0 hours			798,30	0,0	0,2	0,0	0,0	2000,0			
1 hours			782,20	16,1	51,0	51,0	189,0	2400,0			
2 hours			792,90	16,8	31,0	17,7	196,0	1760,0			
6 hours			816,10	26,6	94,0	86,0	429,0	2900,0			
8 hours			2574,30	26,8	103,0	74,0	367,0	2640,0			
Tailing									4,66	630	1580

Calculation to 50% Solids

Samples	**Solids**	**Volume**	**Recovery**	**Assays Solution**					**Assays Solids**		
Name	**Sólidos (g)**	**Solution (mL)**	**Au (%)**	**Au (mg/L)**	**Cu (mg/L)**	**Zn (mg/L)**	**SCN (mg/L)**	**$SO_4^=$ (mg/L)**	**Au (gpt)**	**Cu (g/t)**	**Zn (gpt)**
Feed Calc.	3024,70	3024,70							28,88	723,3	1647,0
0 hours			0,0%	0,0	0,2	0,0	0,0	1811,2			
1 hours			50,6%	14,6	46,2	46,2	171,2	2173,4			
2 hours			52,7%	15,2	28,1	16,0	177,5	1593,8			
6 hours			83,5%	24,1	85,1	77,9	388,5	2626,2			
8 hours			83,9%	24,2	93,3	67,0	332,3	2390,7			
Tailing									4,66	630	1580

Samples	**DO**	**pH**	**$NaCN_{TRIT}$**	**CN^-_{TRIT}**	**Associated**					Calculated	Analyzed
Name	**(mg/L)**		**(mg/L)**	**(mg/L)**	**$Cu(CN)_3^{-4}$**	**$Zn(CN)_4^{-2}$**	**SCN^-**	**CN_{ASSOC}**	**% added**	**CN_{TOTAL}**	**CN_{TOTAL}**
Pre-Lime	4,85	11,0									
0 hours	5,4	11,0	2814,79	1493,56	0,00	0,00	0,00	0,00			
1 hours	6,2	11,0	1145,0	608,0	75,64	73,44	76,72	225,81	15,1%	757,08	
2 hours	7,2	11,1	685,5	364,0	45,98	25,49	79,57	151,03	10,1%	435,47	
6 hours	7,1	11,0	760,8	404,0	139,42	123,85	174,15	437,41	29,3%	667,26	
8 hours	7,4	10,6	721,3	383,0	152,76	106,56	148,98	408,31	27,3%	642,33	679,00
	NaCN consumption (kg/t)		1,9								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-014**
Sample:	**Calcine**
Description:	**PRE-LIME INFLUENCE - 6 hours WITH 100 gpt OF $Pb(NO_3)_2$**
Date:	**25/03/2004**

Teste nº		**% Sólidos**	**Pré-Lime**				**Cianetação**			**CIP**
			(h)	**pH**	**$Pb(NO_3)_2$ (gpt)**	**CaO (kg/t)**	**(h)**	**pH**	**NaCN (kg/t)**	**Carvão (g/L)**
LBII	**014**	50	6	11,0	100		8	11,0	2,5	0

Real Conditions										
WT% solids	**Pre-lime (h)**	**NaCN (g)**	**NaCN (kg/t)**	**NaCN (mg/L)**	**CN (mg/L)**	**CaO (g)**	**CaO (kg/t)**	**pH**	**$Pb(NO_3)_2$ (mg)**	**$Pb(NO_3)_2$ (g/t)**
51,3%	6	7,77	2,55	2687,92	1426,24	24,76	8,12	11	300,00	98,43

Samples	**Solids**	**Volume**	**Pulp**	**Assays Solution**					**Assays Solids**		
Name	**Sólidos (g)**	**Solution (mL)**	**(g)**	**Au (mg/L)**	**Cu (mg/L)**	**Zn (mg/L)**	**SCN (mg/L)**	**$SO_4^=$ (mg/L)**	**Au (gpt)**	**Cu (g/t)**	**Zn (gpt)**
Feed Anal.	3048,00	2890,71	5938,71						29,57		
0 hours			811,91	0,0	0,0	0,0	0,0	1940,0			
1 hours			791,40	21,3	63,0	63,0	354,0	2260,0			
2 hours			788,80	24,8	65,0	64,0	432,0	2140,0			
6 hours			840,50	27,5	80,0	65,0	440,0	2900,0			
8 hours			2706,10	29,0	83,0	71,0	461,0	3100,0			
Tailing									1,86	531	1466

Calculation to 50% Solids

Samples	**Solids**	**Volume**	**Recovery**	**Assays Solution**					**Assays Solids**		
Name	**Sólidos (g)**	**Solution (mL)**	**Au (%)**	**Au (mg/L)**	**Cu (mg/L)**	**Zn (mg/L)**	**SCN (mg/L)**	**$SO_4^=$ (mg/L)**	**Au (gpt)**	**Cu (g/t)**	**Zn (gpt)**
Feed Calc.	3048,00	3048,00							29,36	609,7	1533,3
0 hours			0,0%	0,0	0,0	0,0	0,0	1839,9			
1 hours			68,8%	20,2	59,7	59,7	335,7	2143,4			
2 hours			80,1%	23,5	61,6	60,7	409,7	2029,6			
6 hours			88,8%	26,1	75,9	61,6	417,3	2750,3			
8 hours			93,7%	27,5	78,7	67,3	437,2	2940,0			
Tailing									1,86	531	1466

Samples	**DO**	**pH**	**$NaCN_{TRIT}$**	**CN^-_{TRIT}**	**Associated**					Calculated	Analyzed
Name	**(mg/L)**		**(mg/L)**	**(mg/L)**	**$Cu(CN)_3^{-4}$**	**$Zn(CN)_4^{-2}$**	**SCN^-**	**CN_{ASSOC}**	**% added**	**CN_{TOTAL}**	**CN_{TOTAL}**
Pre-Lime	5	11,1									
0 hours	5,9	11,0	2687,92	1426,24	0,00	0,00	0,00	0,00			
1 hours	6,9	11,0	1606,4	853,0	97,86	95,01	150,50	343,37	24,1%	1045,87	
2 hours	6,4	11,0	1521,7	808,0	100,96	96,52	183,66	381,15	26,7%	1005,48	
6 hours	7,0	11,0	1062,1	564,0	124,26	98,03	187,06	409,35	28,7%	786,29	
8 hours	6,52	10,6	947,3	503,0	128,92	107,08	195,99	431,99	30,3%	739,00	788,00
	NaCN consumption (kg/t)		1,7								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-015
Sample: Calcine
Description: PRE-LIME INFLUENCE - 12 hours WITH 100 gpt OF $Pb(NO_3)_2$
Date: 29/03/2004

Teste nº		% Sólidos	Pré-Lime				Cianetação			CIP
			(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	015	50	12	11,0	0		8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
53,5%	12	7,66	2,55	2932,96	1556,26	25,32	8,43	11	300,00	99,85

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3004,60	2611,70	5616,30						29,57		
0 hours			840,00	0,0	0,1	0,0	0,0	1760,0			
1 hours			838,40	24,2	53,0	83,0	125,0	2400,0			
2 hours			837,30	27,5	55,0	79,0	440,0	2260,0			
6 hours			819,70	29,7	71,0	67,0	450,0	2460,0			
8 hours			2280,90	31,2	81,0	85,0	280,0	2580,0			
Tailing									2,32	570	1366

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3004,60	3004,60							29,44	640,4	1439,9
0 hours			0,0%	0,0	0,1	0,0	0,0	1529,9			
1 hours			71,5%	21,0	46,1	72,1	108,7	2086,2			
2 hours			81,2%	23,9	47,8	68,7	382,5	1964,5			
6 hours			87,7%	25,8	61,7	58,2	391,2	2138,3			
8 hours			92,1%	27,1	70,4	73,9	243,4	2242,6			
Tailing									2,32	570	1366

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	1,8	11,0									
0 hours	6,1	11,0	2932,96	1556,26	0,00	0,00	0,00	0,00			
1 hours	6,8	10,9	2000,0	1062,0	75,45	114,73	48,71	238,89	15,4%	1252,18	
2 hours	6,94	10,9	1728,8	918,0	78,30	109,20	171,45	358,95	23,1%	1105,50	
6 hours	7,3	11,0	1207,2	641,0	101,08	92,61	175,35	369,03	23,7%	834,69	
8 hours	6,9	11,0	924,7	491,0	115,31	117,49	109,10	341,91	22,0%	723,81	679,00
	NaCN consumption (kg/t)		1,7								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-016**
Sample: **Calcine**
Description: **NaCN INFLUENCE - 1.5 kpgt NaCN**
Date: **23/03/2004**

Teste n°		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	016	50	2	11,0	0		8	11,0	1,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,5%	2	4,62	1,53	1623,16	861,27	22,08	7,30	11	0,00	0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^{=}$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3023,30	2846,30	5869,60						29,57		
0 hours			801,70	0,0	0,1	0,0	0,0	1760,0			
1 hours			785,80	10,3	62,0	37,0	252,0	2400,0			
2 hours			808,00	13,1	92,0	76,0	275,0	2080,0			
6 hours			829,20	20,4	78,0	51,0	333,0	2260,0			
8 hours			2644,90	22,3	94,0	46,0	351,0	2580,0			
Tailing									9,14	600	1511

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^{=}$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3023,30	3023,30							30,13	688,5	1554,3
0 hours			0,0%	0,0	0,1	0,0	0,0	1657,0			
1 hours			32,2%	9,7	58,4	34,8	237,2	2259,5			
2 hours			40,9%	12,3	86,6	71,6	258,9	1958,2			
6 hours			63,7%	19,2	73,4	48,0	313,5	2127,7			
8 hours			69,7%	21,0	88,5	43,3	330,5	2429,0			
Tailing									9,14	600	1511

Samples	DO	pH	$NaCN_{TRIT}$	CN^{-}_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^{-}	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	4,9	11,0									
0 hours	5,2	11,0	1623,16	861,27	0,00	0,00	0,00	0,00			
1 hours	7,15	11,0	898,3	477,0	95,60	55,39	106,35	257,34	29,9%	627,99	
2 hours	7,15	11,0	713,7	379,0	141,86	113,78	116,06	371,70	43,2%	634,64	
6 hours	6,2	11,0	487,8	259,0	120,27	76,35	140,54	337,16	39,1%	455,62	
8 hours	7,1	10,6	497,2	264,0	144,94	68,87	148,13	361,94	42,0%	477,81	518,00
	NaCN consumption (kg/t)		1,1								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-017**
Sample: **Calcine**
Description: **NaCN INFLUENCE - 2.5 kpgt NaCN**
Date: **25/03/2004**

Teste nº		% Sólidos	Pré-Lime (h)	Pré-Lime pH	Pré-Lime $Pb(NO_3)_2$ (gpt)	Pré-Lime CaO (kg/t)	Cianetação (h)	Cianetação pH	Cianetação NaCN (kg/t)	CIP Carvão (g/L)
LBII	**017**	50	2	11,0	0		8	11,0	2,5	0

Real Conditions										
WT% solids	**Pre-lime (h)**	**NaCN (g)**	**NaCN (kg/t)**	**NaCN (mg/L)**	**CN (mg/L)**	**CaO (g)**	**CaO (kg/t)**	**pH**	**$Pb(NO_3)_2$ (mg)**	**$Pb(NO_3)_2$ (g/t)**
51,5%	2	7,78	2,55	2708,54	1437,18	21,07	6,91	11	0,00	0,00

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3050,60	2872,40	5923,00						29,57		
0 hours			798,00	0,0	0,3	0,0	0,0	2080,0			
1 hours			783,70	16,3	55,0	65,0	198,0	2460,0			
2 hours			801,60	21,9	67,0	87,0	206,0	2580,0			
6 hours			832,70	26,3	69,0	74,0	294,0	2460,0			
8 hours			2707,00	28,4	105,0	125,0	339,0	3020,0			
Tailing									1,97	584	1506

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3050,60	3050,60							28,71	682,9	1623,7
0 hours			0,0%	0,0	0,2	0,0	0,0	1958,5			
1 hours			53,5%	15,3	51,8	61,2	186,4	2316,3			
2 hours			71,8%	20,6	63,1	81,9	194,0	2429,3			
6 hours			86,3%	24,8	65,0	69,7	276,8	2316,3			
8 hours			93,1%	26,7	98,9	117,7	319,2	2843,6			
Tailing									1,97	584	1506

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated $Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
Pre-Lime	5,5	11,0									
0 hours	6,1	11,0	2708,54	1437,18	0,00	0,00	0,00	0,00			
1 hours	5,9	11,0	1485,9	789,0	84,82	97,33	83,57	265,72	18,5%	971,14	
2 hours	6,1	11,0	1201,5	638,0	103,32	130,27	86,95	320,54	22,3%	871,59	
6 hours	6,9	11,0	917,1	487,0	106,41	110,80	124,09	341,30	23,7%	704,21	
8 hours	6,2	10,9	711,9	378,0	161,92	187,17	143,09	492,18	34,2%	727,09	633,00
	NaCN consumption (kg/t)		1,9								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-018**
Sample: **Calcine**
Description: **NaCN INFLUENCE - 5 kpgt NaCN**
Date: **17/03/2004**

Teste n°		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	018	50	2	11,0	0		8	11,0	5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
50,7%	2	15,14	5,00	5136,84	2725,67	19,77	6,53	11	0,00	0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3028,00	2947,34	5975,34						29,57		
0 hours			725,88	0,0	2,4	0,4	0,0	1880,0			
1 hours			700,24	16,2	62,0	63,0	266,0	2580,0			
2 hours			770,37	30,3	64,0	66,0	209,0	3480,0			
6 hours			766,85	30,6	116,0	106,0	303,0	3340,0			
8 hours			3012,00	31,1	125,0	116,0	537,0	3660,0			
Tailing									6,49	639	1570

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3028,00	3028,00							36,76	760,7	1682,9
0 hours			0,0%	0,0	2,3	0,4	0,0	1829,9			
1 hours			42,9%	15,8	60,3	61,3	258,9	2511,3			
2 hours			80,2%	29,5	62,3	64,2	203,4	3387,3			
6 hours			81,0%	29,8	112,9	103,2	294,9	3251,0			
8 hours			82,3%	30,3	121,7	112,9	522,7	3562,5			
Tailing									6,49	639	1570

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	2,28	11,0									
0 hours		11,0	5136,84	2725,67	0,00	0,00	0,00	0,00			
1 hours		11,0	4212,8	2237,0	98,84	97,51	116,07	312,42	11,5%	2433,35	
2 hours	8,5	11,0	3131,8	1663,0	102,03	102,16	91,19	295,38	10,8%	1867,19	
6 hours	8,6	11,0	1939,7	1030,0	184,92	164,07	132,21	481,21	17,7%	1379,00	
8 hours	7,8	10,7	1774,0	942,0	199,27	179,55	234,31	613,13	22,5%	1320,82	1851,00
	NaCN consumption (kg/t)		3,3								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-019
Sample: Calcine
Description: NaCN INFLUENCE - 1.5 kpgt NaCN + 100 gpt $Pb(NO_3)_2$
Date: 22/03/2004

Teste nº		% Sólidos	Pré-Lime (h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	Cianetação (h)	pH	NaCN (kg/t)	CIP Carvão (g/L)
LBII	019	50	2	11,0	100		8	11,0	1,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,4%	2	4,65	1,55	1638,42	869,37	19,74	6,58	11	300,00	99,94

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3001,80	2838,10	5839,90						29,57		
0 hours			807,00	0,0	0,1	0,0	0,0	2000,0			
1 hours			780,70	11,4	73,0	66,0	191,0	2320,0			
2 hours			806,10	13,4	60,0	64,0	202,0	2200,0			
6 hours			810,00	19,5	84,0	49,8	329,0	2400,0			
8 hours			2636,10	21,9	96,0	52,0	378,0	2640,0			
Tailing									7,96	565	1518

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3001,80	3001,80							28,65	655,8	1567,2
0 hours			0,0%	0,0	0,1	0,0	0,0	1890,9			
1 hours			37,6%	10,8	69,0	62,4	180,6	2193,5			
2 hours			44,2%	12,7	56,7	60,5	191,0	2080,0			
6 hours			64,4%	18,4	79,4	47,1	311,1	2269,1			
8 hours			72,2%	20,7	90,8	49,2	357,4	2496,0			
Tailing									7,96	565	1518

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	5,7	11,0									
0 hours	6,1	11,0	1638,42	869,37	0,00	0,00	0,00	0,00			
1 hours	7,1	11,0	593,2	315,0	113,04	99,23	80,95	293,22	33,7%	527,27	
2 hours	7,4	11,0	467,0	248,0	92,91	96,22	85,61	274,75	31,6%	437,13	
6 hours	7,3	11,0	331,5	176,0	130,07	74,87	139,44	344,39	39,6%	380,95	
8 hours	7,9	10,4	389,8	207,0	148,65	78,18	160,21	387,04	44,5%	433,84	362,00
	NaCN consumption (kg/t)		1,2								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-020**
Sample:	**Calcine**
Description:	**NaCN INFLUENCE - 2.5 kpgt NaCN + 100 gpt Pb(NO3)2**
Date:	**22/03/2004**

Teste nº		% Sólidos	Pré-Lime (h)	pH	Pb(NO$_3$)$_2$ (gpt)	CaO (kg/t)	Cianetação (h)	pH	NaCN (kg/t)	CIP Carvão (g/L)
LBII	**020**	50	2	11,0	100		8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	Pb(NO$_3$)$_2$ (mg)	Pb(NO$_3$)$_2$ (g/t)
51,4%	2	7,69	2,55	2701,85	1433,63	21,65	7,18	11	300,00	99,50

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	SO$_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3015,00	2846,20	5861,20						29,57		
0 hours			827,60	0,0	0,2	0,0	0,0	2080,0			
1 hours			808,90	25,1	79,0	41,0	222,0	2260,0			
2 hours			790,50	25,8	84,0	76,0	310,0	2460,0			
6 hours			804,00	28,0	105,0	88,0	338,0	2580,0			
8 hours			2630,20	28,1	137,0	109,0	362,0	2580,0			
Tailing									1,98	531	1476

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	SO$_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3015,00	3015,00							28,51	660,3	1578,9
0 hours			0,0%	0,0	0,2	0,0	0,0	1963,5			
1 hours			83,1%	23,7	74,6	38,7	209,6	2133,5			
2 hours			85,4%	24,4	79,3	71,7	292,6	2322,3			
6 hours			92,7%	26,4	99,1	83,1	319,1	2435,6			
8 hours			93,1%	26,5	129,3	102,9	341,7	2435,6			
Tailing									1,98	531	1476

Samples Name	DO (mg/L)	pH	NaCN$_{TRIT}$ (mg/L)	CN$^-_{TRIT}$ (mg/L)	Associated Cu(CN)$_3^{-4}$	Zn(CN)$_4^{-2}$	SCN$^-$	CN$_{ASSOC}$	% added	Calculated CN$_{TOTAL}$	Analyzed CN$_{TOTAL}$
Pre-Lime	3,6	11,0									
0 hours	5,4	11,0	2701,85	1433,63	0,00	0,00	0,00	0,00			
1 hours	5,9	11,0	1600,8	850,0	122,14	61,55	93,95	277,64	19,4%	1033,69	
2 hours	5,9	11,0	1282,5	681,0	129,87	114,09	131,19	375,15	26,2%	924,96	
6 hours	6,4	11,0	1073,4	570,0	162,34	132,10	143,03	437,48	30,5%	864,44	
8 hours	6,8	10,7	951,0	505,0	211,82	163,63	153,19	528,64	36,9%	880,44	853,00
	NaCN consumption (kg/t)		1,7								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-021**
Sample:	**Calcine**
Description:	**NaCN INFLUENCE - 5 kpgt NaCN + 100 gpt Pb(NO3)2**
Date:	**17/03/2004**

Teste nº		% Sólidos	Pré-Lime (h)	Pré-Lime pH	Pré-Lime $Pb(NO_3)_2$ (gpt)	Pré-Lime CaO (kg/t)	Cianetação (h)	Cianetação pH	Cianetação NaCN (kg/t)	CIP Carvão (g/L)
LBII	021	50	2	11,0	100		8	11,0	5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,7%	2	15,03	5,00	5351,75	2839,70	20,41	6,79	11	300,00	99,77

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^{=}$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3006,90	2809,27	5816,17						29,57		
0 hours			781,43	0,0	2,6	0,0	0,0	1880,0			
1 hours			777,90	22,1	79,0	71,0	248,0	2580,0			
2 hours			793,97	21,3	80,0	69,0	287,0	2900,0			
6 hours			786,27	26,4	86,0	78,0	491,0	3160,0			
8 hours			2676,60	28,8	100,0	86,0	483,0	3980,0			
Tailing									3,50	637	1605

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^{=}$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3006,90	3006,90							30,41	730,4	1685,3
0 hours			0,0%	0,0	2,4	0,0	0,0	1756,4			
1 hours			67,9%	20,6	73,8	66,3	231,7	2410,4			
2 hours			65,4%	19,9	74,7	64,5	268,1	2709,4			
6 hours			81,1%	24,7	80,3	72,9	458,7	2952,3			
8 hours			88,5%	26,9	93,4	80,3	451,3	3718,4			
Tailing									3,50	637	1605

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated $Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
Pre-Lime	4,52	11,0									
0 hours		11,0	5351,75	2839,70	0,00	0,00	0,00	0,00			
1 hours		10,9	3431,3	1822,0	120,88	105,48	103,87	330,23	11,6%	2048,37	
2 hours	8,6	10,9	2647,8	1406,0	122,41	102,51	120,20	345,12	12,2%	1630,93	
6 hours	9,2	10,9	2256,1	1198,0	131,59	115,88	205,64	453,11	16,0%	1445,48	
8 hours	7,9	10,9	2013,2	1069,0	153,02	127,77	202,29	483,07	17,0%	1349,78	2018,00

NaCN consumption (kg/t)	3,1

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-022**
Sample:	**Calcine**
Description:	**$Pb(NO3)2$ INFLUENCE - 5 gpt $Pb(NO3)2$**
Date:	**22/03/2004**

Teste n°		% Sólidos	Pré-Lime (h)	Pré-Lime pH	Pré-Lime $Pb(NO_3)_2$ (gpt)	Pré-Lime CaO (kg/t)	Cianetação (h)	Cianetação pH	Cianetação NaCN (kg/t)	CIP Carvão (g/L)
LBII	**022**	50	2	11,0	5		8	11,0	2,5	0

Real Conditions										
WT% solids	**Pre-lime (h)**	**NaCN (g)**	**NaCN (kg/t)**	**NaCN (mg/L)**	**CN (mg/L)**	**CaO (g)**	**CaO (kg/t)**	**pH**	**$Pb(NO_3)_2$ (mg)**	**$Pb(NO_3)_2$ (g/t)**
51,5%	2	7,65	2,55	2711,42	1438,71	21,25	7,08	11	15,00	5,00

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3001,00	2821,40	5822,40						29,57		
0 hours			808,40	0,0	0,2	0,0	0,0	1940,0			
1 hours			829,50	18,8	52,0	59,0	256,0	2520,0			
2 hours			801,70	23,9	80,0	89,0	340,0	2460,0			
6 hours			800,90	26,8	90,0	80,0	342,0	2640,0			
8 hours			2581,90	27,1	103,0	92,0	464,0	2640,0			
Tailing									5,92	599	1480

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3001,00	3001,00							31,43	695,8	1566,5
0 hours			0,0%	0,0	0,2	0,0	0,0	1823,9			
1 hours			56,1%	17,6	48,9	55,5	240,7	2369,2			
2 hours			71,4%	22,5	75,2	83,7	319,7	2312,8			
6 hours			80,0%	25,1	84,6	75,2	321,5	2482,0			
8 hours			81,2%	25,5	96,8	86,5	436,2	2482,0			
Tailing									5,92	599	1480

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated $Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
Pre-Lime	5,7	11,0									
0 hours	5,3	11,0	2711,42	1438,71	0,00	0,00	0,00	0,00			
1 hours	6,5	11,0	1354,0	719,0	80,07	88,21	107,89	276,17	19,2%	887,28	
2 hours	7,3	11,0	1169,5	621,0	123,18	133,06	143,29	399,53	27,8%	877,24	
6 hours	7,2	11,0	742,0	394,0	138,58	119,60	144,14	402,32	28,0%	652,18	
8 hours	7,8	10,7	811,7	431,0	158,60	137,54	195,55	491,69	34,2%	727,14	730,00
	NaCN consumption (kg/t)		1,8								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-023
Sample: Calcine
Description: Pb(NO3)2 INFLUENCE - 25 gpt Pb(NO3)2
Date: 23/03/2004

Teste nº		% Sólidos	Pré-Lime (h)	Pré-Lime pH	Pré-Lime $Pb(NO_3)_2$ (gpt)	Pré-Lime CaO (kg/t)	Cianetação (h)	Cianetação pH	Cianetação NaCN (kg/t)	CIP Carvão (g/L)
LBII	023	50	2	11,0	25		8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
52,8%	2	7,70	2,55	2850,27	1512,39	23,68	7,84	11	75,00	24,83

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3020,30	2701,50	5721,80						29,57		
0 hours			828,00	0,0	0,1	0,0	0,0	1680,0			
1 hours			822,50	17,4	79,0	79,0	277,0	2460,0			
2 hours			804,20	21,1	82,0	73,0	338,0	2700,0			
6 hours			806,50	24,4	76,0	49,0	384,0	3280,0			
8 hours			2460,60	26,5	150,0	129,0	404,0	3160,0			
Tailing									5,22	605	1572

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3020,30	3020,30							28,92	739,2	1687,4
0 hours			0,0%	0,0	0,0	0,0	0,0	1502,7			
1 hours			53,8%	15,6	70,7	70,7	247,8	2200,3			
2 hours			65,3%	18,9	73,3	65,3	302,3	2415,0			
6 hours			75,5%	21,8	68,0	43,8	343,5	2933,8			
8 hours			82,0%	23,7	134,2	115,4	361,4	2826,5			
Tailing									5,22	605	1572

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated $Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
Pre-Lime	5,0	11,0									
0 hours	5,9	11,0	2850,27	1512,39	0,00	0,00	0,00	0,00			
1 hours	7,3	11,0	1598,9	849,0	115,73	112,37	111,07	339,16	22,4%	1077,10	
2 hours	7,5	11,0	1423,7	756,0	120,12	103,83	135,52	359,48	23,8%	979,96	
6 hours	6,9	11,0	941,6	500,0	111,33	69,70	153,97	335,00	22,2%	681,03	
8 hours	7,6	11,0	764,6	406,0	219,74	183,48	161,99	565,21	37,4%	809,22	730,00
	NaCN consumption (kg/t)		1,9								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-024**
Sample:	**Calcine**
Description:	**Pb(NO3)2 INFLUENCE - 50 gpt Pb(NO3)2**
Date:	**23/03/2004**

Teste nº		**%**	**Pré-Lime**				**Cianetação**			**CIP**
		Sólidos	**(h)**	**pH**	**$Pb(NO_3)_2$ (gpt)**	**CaO (kg/t)**	**(h)**	**pH**	**NaCN (kg/t)**	**Carvão (g/L)**
LBII	**024**	50	2	11,0	50		8	11,0	2,5	0

Real Conditions										
WT% solids	**Pre-lime (h)**	**NaCN (g)**	**NaCN (kg/t)**	**NaCN (mg/L)**	**CN (mg/L)**	**CaO (g)**	**CaO (kg/t)**	**pH**	**$Pb(NO_3)_2$ (mg)**	**$Pb(NO_3)_2$ (g/t)**
51,5%	2	7,69	2,55	2701,09	1433,23	20,80	6,89	11	151,00	50,03

Samples	**Solids**	**Volume**	**Pulp**	**Assays Solution**					**Assays Solids**		
Name	**Sólidos (g)**	**Solution (mL)**	**(g)**	**Au (mg/L)**	**Cu (mg/L)**	**Zn (mg/L)**	**SCN (mg/L)**	**$SO_4^=$ (mg/L)**	**Au (gpt)**	**Cu (g/t)**	**Zn (gpt)**
Feed Anal.	3017,90	2847,00	5864,90						29,57		
0 hours			794,20	0,0	0,2	0,0	0,0	1880,0			
1 hours			791,60	15,4	48,0	65,0	199,0	2400,0			
2 hours			818,30	23,0	60,0	80,0	264,0	2700,0			
6 hours			808,80	27,6	74,0	96,0	271,0	2400,0			
8 hours			2652,00	27,9	90,0	117,0	348,0	3100,0			
Tailing									2,04	571	1480

Calculation to 50% Solids

Samples	**Solids**	**Volume**	**Recovery**	**Assays Solution**					**Assays Solids**		
Name	**Sólidos (g)**	**Solution (mL)**	**Au (%)**	**Au (mg/L)**	**Cu (mg/L)**	**Zn (mg/L)**	**SCN (mg/L)**	**$SO_4^=$ (mg/L)**	**Au (gpt)**	**Cu (g/t)**	**Zn (gpt)**
Feed Calc.	3017,90	3017,90							28,36	655,9	1590,4
0 hours			0,0%	0,0	0,2	0,0	0,0	1773,5			
1 hours			51,2%	14,5	45,3	61,3	187,7	2264,1			
2 hours			76,5%	21,7	56,6	75,5	249,1	2547,1			
6 hours			91,8%	26,0	69,8	90,6	255,7	2264,1			
8 hours			92,8%	26,3	84,9	110,4	328,3	2924,5			
Tailing									2,04	571	1480

Samples	**DO**	**pH**	**$NaCN_{TRIT}$**	**CN^-_{TRIT}**	**Associated**					Calculated	Analyzed
Name	**(mg/L)**		**(mg/L)**	**(mg/L)**	**$Cu(CN)_3^{-4}$**	**$Zn(CN)_4^{-2}$**	**SCN^-**	**CN_{ASSOC}**	**% added**	**CN_{TOTAL}**	**CN_{TOTAL}**
Pre-Lime	4,8	11,0									
0 hours	6,1	11,0	2701,09	1433,23	0,00	0,00	0,00	0,00			
1 hours	6,2	11,0	1504,7	799,0	74,16	97,51	84,16	255,83	17,8%	970,67	
2 hours	6,6	11,0	1391,7	739,0	92,70	120,01	111,64	324,36	22,6%	951,72	
6 hours	6,8	11,0	917,1	487,0	114,33	144,02	114,60	372,95	26,0%	745,35	
8 hours	6,6	10,8	711,9	378,0	139,05	175,52	147,17	461,74	32,2%	692,57	679,00
	NaCN consumption (kg/t)		1,9								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID:	**LBII-025**
Sample:	**Calcine**
Description:	**Pb(NO3)2 INFLUENCE - 100 gpt Pb(NO3)2**
Date:	**23/03/2004**

Teste nº		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	025	50	2	11,0	100		8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,2%	2	7,77	2,55	2676,63	1420,25	25,84	8,48	11	300,00	98,49

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3046,10	2902,90	5949,00						29,57		
0 hours			808,70	0,0	0,0	0,0	0,0	2080,0			
1 hours			790,90	18,3	90,0	77,0	223,0	2400,0			
2 hours			818,50	23,4	125,0	105,0	316,0	2260,0			
6 hours			800,00	26,6	131,0	107,0	359,0	2260,0			
8 hours			2730,90	27,8	90,0	70,0	352,0	2400,0			
Tailing									2,16	521	1458

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3046,10	3046,10							28,65	606,8	1524,7
0 hours			0,0%	0,0	0,0	0,0	0,0	1982,2			
1 hours			60,9%	17,4	85,8	73,4	212,5	2287,2			
2 hours			77,8%	22,3	119,1	100,1	301,1	2153,8			
6 hours			88,5%	25,3	124,8	102,0	342,1	2153,8			
8 hours			92,5%	26,5	85,8	66,7	335,5	2287,2			
Tailing									2,16	521	1458

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	3,5	11,1									
0 hours	5,2	11,0	2676,63	1420,25	0,00	0,00	0,00	0,00			
1 hours	5,8	11,0	1248,6	663,0	140,47	116,69	95,27	352,43	24,8%	920,16	
2 hours	6	11,0	1563,1	830,0	195,10	159,12	135,00	489,22	34,4%	1184,22	
6 hours	6,5	11,0	1054,6	560,0	204,46	162,15	153,37	519,98	36,6%	926,62	
8 hours	6,7	10,7	941,6	500,0	140,47	106,08	150,38	396,93	27,9%	746,55	788,00
	NaCN consumption (kg/t)		1,7								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-026**
Sample: **Calcine**
Description: **Pb(NO3)2 INFLUENCE - 200 gpt Pb(NO3)2**
Date: **23/03/2004**

Teste nº		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	**026**	50	2	11,0	200		8	11,0	2,5	0

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
52,1%	2	7,65	2,55	2772,14	1470,93	21,51	7,17	11	600,20	200,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3001,00	2759,60	5760,60						29,57		
0 hours			787,90	0,0	1,1	0,0	0,0	1820,0			
1 hours			787,80	24,0	53,0	85,0	271,0	2580,0			
2 hours			779,50	24,3	69,0	66,0	290,0	2700,0			
6 hours			789,40	26,1	91,0	71,0	339,0	2700,0			
8 hours			2616,00	28,1	92,0	96,0	468,0	2960,0			
Tailing									2,54	686	1715

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3001,00	3001,00							28,38	770,6	1803,3
0 hours			0,0%	0,0	1,0	0,0	0,0	1673,6			
1 hours			77,8%	22,1	48,7	78,2	249,2	2372,5			
2 hours			78,7%	22,3	63,4	60,7	266,7	2482,8			
6 hours			84,6%	24,0	83,7	65,3	311,7	2482,8			
8 hours			91,0%	25,8	84,6	88,3	430,4	2721,9			
Tailing									2,54	686	1715

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	3,8	11,1									
0 hours	6,5	11,0	2772,14	1470,93	0,00	0,00	0,00	0,00			
1 hours	7,7	11,0	1568,7	833,0	79,82	124,30	111,71	315,83	21,5%	1037,12	
2 hours	7,3	11,0	1382,3	734,0	103,92	96,51	119,54	319,97	21,8%	934,43	
6 hours	6,2	11,0	952,9	506,0	137,05	103,82	139,74	380,62	25,9%	746,87	
8 hours	5,8	10,7	738,2	392,0	138,56	140,38	192,92	471,85	32,1%	670,94	730,00
	NaCN consumption (kg/t)		1,9								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: **LBII-027**
Sample: **Calcine**
Description: **ACTIVATED CARBON IN THE LEACHING INFLUENCE**
Date: **30/03/2004**

Teste nº		%	Pré-Lime				Cianetação			CIP
		Sólidos	(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	027	50	2	11,0			8	11,0	2,5	24

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,5%	2	7,69	2,55	2713,38	1439,75	23,95	7,94	11		0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^{=}$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3015,30	2834,10	5849,40						29,57		
0 hours			832,00	0,0	1,8	1,2	0,0	1500,0			
1 hours			830,10	0,9	55,0	40,0	148,0	1680,0			
2 hours			804,70	0,2	65,0	40,0	171,0	1760,0			
6 hours			838,00	0,03	72,0	53,0	232,0	1680,0			
8 hours			2544,60	0,0	76,0	54,0	185,0	1760,0			
Tailing									2,01	535	1321

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^{=}$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3015,30	3015,30							29,57	606,4	1371,8
0 hours			0,0%	0,0	1,7	1,1	0,0	1409,9			
1 hours			2,9%	0,8	51,7	37,6	139,1	1579,0			
2 hours			0,7%	0,2	61,1	37,6	160,7	1654,2			
6 hours			0,1%	0,0	67,7	49,8	218,1	1579,0			
8 hours			0,1%	0,0	71,4	50,8	173,9	1654,2			
Tailing									2,01	535	1321

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	4,29	11,1									
0 hours	5,5	11,0	2713,38	1439,75	0,00	0,00	0,00	0,00			
1 hours	5,7	11,0	1555,6	826,0	84,67	59,79	62,36	206,81	14,4%	970,45	
2 hours	6,6	11,0	1265,5	672,0	100,06	59,79	72,05	231,89	16,1%	831,85	
6 hours	6,3	11,0	638,4	339,0	110,83	79,22	97,75	287,80	20,0%	529,05	
8 hours	7,8	10,7	557,4	296,0	116,99	80,71	77,95	275,65	19,1%	493,70	518,00
	NaCN consumption (kg/t)		2,0								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-028
Sample: Calcine
Description: ACTIVATED CARBON IN THE LEACHING INFLUENCE
Date: 30/03/2004

Teste nº		% Sólidos	Pré-Lime (h)	Pré-Lime pH	Pré-Lime $Pb(NO_3)_2$ (gpt)	Pré-Lime CaO (kg/t)	Cianetação (h)	Cianetação pH	Cianetação NaCN (kg/t)	CIP Carvão (g/L)
LBII	028	50	2	11,0			8	11,0	2,5	24

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
50,5%	2	7,70	2,55	2601,00	1380,12	24,53	8,12	11		0,00

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3021,00	2960,40	5981,40						29,57		
0 hours			814,00	0,0	0,1	0,0	0,0	1680,0			
1 hours			813,20	0,9	55,0	45,0	191,0	1880,0			
2 hours			811,40	0,3	60,0	55,0	206,0	1940,0			
6 hours			819,00	0,03	73,0	55,0	217,0	2080,0			
8 hours			2723,80	0,0	78,0	57,0	234,0	1680,0			
Tailing									2,01	549	1367

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Assays Solids Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3021,00	3021,00							29,57	625,4	1422,9
0 hours			0,0%	0,0	0,1	0,0	0,0	1646,3			
1 hours			3,0%	0,9	53,9	44,1	187,2	1842,3			
2 hours			1,0%	0,3	58,8	53,9	201,9	1901,1			
6 hours			0,1%	0,0	71,5	53,9	212,6	2038,3			
8 hours			0,1%	0,0	76,4	55,9	229,3	1646,3			
Tailing									2,01	549	1367

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated $Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
Pre-Lime	4,29	11,1									
0 hours	5,5	11,0	2601,00	1380,12	0,00	0,00	0,00	0,00			
1 hours	5,7	11,0	1476,5	784,0	88,27	70,12	83,90	242,30	17,6%	942,40	
2 hours	6,6	11,0	1122,4	596,0	96,30	85,71	90,49	272,50	19,7%	778,00	
6 hours	6,3	11,0	708,1	376,0	117,16	85,71	95,32	298,19	21,6%	578,87	
8 hours	7,8	10,7	548,0	291,0	125,19	88,82	102,79	316,80	23,0%	505,01	590,00
	NaCN consumption (kg/t)		2,0								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-029
Sample: Calcine
Description: DESTILATED WATER INFLUENCE
Date: 30/03/2004

Teste nº		% Sólidos	Pré-Lime				Cianetação			CIP
			(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	029	50	2	11,0			8	11,0	2,5	

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,5%	2	7,74	2,55	2708,69	1437,26	17,42	5,74	11		0,00

Samples	Solids	Volume	Pulp	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	(g)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3035,00	2857,47	5892,47						29,57		
0 hours			835,91	0,0	0,1	0,0	0,0	1760,0			
1 hours			802,42	17,7	69,0	67,0	272,0	2320,0			
2 hours			793,26	21,4	90,0	57,0	301,0	2400,0			
6 hours			806,38	26,1	77,0	58,0	415,0	2780,0			
8 hours			2654,50	26,1	90,0	59,0	466,0	2700,0			
Tailing									2,39	582	1502

Calculation to 50% Solids

Samples	Solids	Volume	Recovery	Assays Solution					Assays Solids		
Name	Sólidos (g)	Solution (mL)	Au (%)	Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3035,00	3035,00							26,99	666,7	1557,5
0 hours			0,0%	0,0	0,1	0,0	0,0	1657,1			
1 hours			61,8%	16,7	65,0	63,1	256,1	2184,3			
2 hours			74,6%	20,1	84,7	53,7	283,4	2259,6			
6 hours			91,1%	24,6	72,5	54,6	390,7	2617,4			
8 hours			91,1%	24,6	84,7	55,5	438,7	2542,1			
Tailing									2,39	582	1502

Samples	DO	pH	$NaCN_{TRIT}$	CN^-_{TRIT}	Associated					Calculated	Analyzed
Name	(mg/L)		(mg/L)	(mg/L)	$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added	CN_{TOTAL}	CN_{TOTAL}
Pre-Lime	4,8	11,1									
0 hours	3,5	10,9	2708,69	1437,26	0,00	0,00	0,00	0,00			
1 hours	6	11,0	1244,8	661,0	106,40	100,31	114,80	321,51	22,4%	867,71	
2 hours	6,6	11,0	1175,1	624,0	138,78	85,34	127,04	351,16	24,4%	848,12	
6 hours	7,1	10,8	758,9	403,0	118,73	86,84	175,15	380,72	26,5%	608,57	
8 hours	7,6	10,7	723,2	384,0	138,78	88,33	196,68	423,79	29,5%	611,11	730,00
	NaCN consumption (kg/t)		1,9								

CONTROL SHEET - LABORATORY TESTWORK - ANNEX 1

Leach Test ID: LBII-030
Sample: Calcine
Description: DESTILATED WATER INFLUENCE
Date: 30/03/2004

Teste nº		% Sólidos	Pré-Lime				Cianetação			CIP
			(h)	pH	$Pb(NO_3)_2$ (gpt)	CaO (kg/t)	(h)	pH	NaCN (kg/t)	Carvão (g/L)
LBII	030	50	2	11,0			8	11,0	2,5	24

Real Conditions										
WT% solids	Pre-lime (h)	NaCN (g)	NaCN (kg/t)	NaCN (mg/L)	CN (mg/L)	CaO (g)	CaO (kg/t)	pH	$Pb(NO_3)_2$ (mg)	$Pb(NO_3)_2$ (g/t)
51,8%	2	7,65	2,55	2739,49	1453,61	18,56	6,18	11		0,00

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Pulp (g)	Assays Solution					Assays Solids		
				Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Anal.	3000,90	2792,49	5793,39						29,57		
0 hours			821,70	0,0	1,8	2,3	0,0	2140,0			
1 hours			809,10	14,5	64,0	73,0	214,0	2260,0			
2 hours			869,40	20,4	83,0	118,0	216,0	2640,0			
6 hours			807,20	26,6	88,0	96,0	271,0	2580,0			
8 hours			2485,99	27,0	92,0	103,0	258,0	2640,0			
Tailing									2,20	588	1541

Calculation to 50% Solids

Samples Name	Solids Sólidos (g)	Volume Solution (mL)	Recovery Au (%)	Assays Solution					Assays Solids		
				Au (mg/L)	Cu (mg/L)	Zn (mg/L)	SCN (mg/L)	$SO_4^=$ (mg/L)	Au (gpt)	Cu (g/t)	Zn (gpt)
Feed Calc.	3000,90	3000,90							27,32	673,6	1636,8
0 hours			0,0%	0,0	1,7	2,1	0,0	1991,4			
1 hours			49,4%	13,5	59,6	67,9	199,1	2103,0			
2 hours			69,5%	19,0	77,2	109,8	201,0	2456,7			
6 hours			90,6%	24,8	81,9	89,3	252,2	2400,8			
8 hours			91,9%	25,1	85,6	95,8	240,1	2456,7			
Tailing									2,20	588	1541

Samples Name	DO (mg/L)	pH	$NaCN_{TRIT}$ (mg/L)	CN^-_{TRIT} (mg/L)	Associated					Calculated CN_{TOTAL}	Analyzed CN_{TOTAL}
					$Cu(CN)_3^{-4}$	$Zn(CN)_4^{-2}$	SCN^-	CN_{ASSOC}	% added		
Pre-Lime	5,2	11,1									
0 hours	5,5	11,0	2739,49	1453,61	0,00	0,00	0,00	0,00			
1 hours	5,9	11,0	1354,0	719,0	97,54	108,02	89,27	294,83	20,3%	924,56	
2 hours	6,2	11,0	1194,0	634,0	126,50	174,61	90,10	391,21	26,9%	935,11	
6 hours	7	11,0	725,0	385,0	134,12	142,06	113,05	389,22	26,8%	661,17	
8 hours	6,8	10,8	636,5	338,0	140,21	152,42	107,62	400,25	27,5%	630,63	679,00
	NaCN consumption (kg/t)		2,0								

TESTES
DE CONFIRMAÇÃO

LABORATORY CONFIRMATION TEST

CIANETAÇÃO CONCENTRADO COMPOSTO

SEM PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro Teor em g/t				REAGENTES			
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)	NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
1	Cianetação	54,6	26,06	19,71	6,35	75,6	0,0100	5,7	6067,5	5984,3
2	Cianetação	54,2	26,07	19,52	6,55	74,9	0,0110	5,5	6074,9	5981,9
3	Cianetação	53,9	26,29	19,75	6,54	75,1	0,0120	5,5	6073,8	5971,1
						75,2		5,6	6072,0	5979,1

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
1	Cianetação	261,4	23,7	1841,0	314,3	2416,7	575,7	10,2	1,780	1,907
2	Cianetação	270,8	23,1	2046,2	320,5	2637,5	591,3	10,4	1,760	1,947
3	Cianetação	262,0	23,1	1739,8	306,4	2308,2	568,4	10,1	1,700	1,861

Condições da cianetação:
Pré - aeração / Duração de 0 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO CONCENTRADO COMPOSTO

2 HORAS PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro				REAGENTES			
			Teor em g/t							
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)	NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
4	Cianetação	53,6	25,79	19,15	6,64	74,3	0,0130	6,7	6075,8	5963,1
5	Cianetação	54,0	25,48	19,56	5,92	76,8	0,0140	6,3	6059,1	5940,0
6	Cianetação	54,1	24,25	18,12	6,13	74,7	0,0144	6,0	6097,0	5975,1
						75,2		6,3	6077,3	5959,4

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
4	Cianetação	290,4	22,1	1727,1	335,1	2352,6	625,5	10,2	2,230	2,036
5	Cianetação	253,3	23	1735,0	297,9	2286,2	551,2	10,2	1,886	1,805
6	Cianetação	296,8	21,4	1326,2	350,5	1973,5	647,3	10,2	2,100	2,137

Condições da cianetação:
Pré - aeração / Duração de 2 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO CONCENTRADO COMPOSTO

6 HORAS PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro				REAGENTES			
			Teor em g/t				NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)				
7	Cianetação	54,2	25,30	19,84	5,47	78,4	0,0206	7,2	6042,1	5868,2
8	Cianetação	54,7	25,63	19,81	5,82	77,3	0,0230	6,5	6059,9	5869,3
9	Cianetação	53,6	25,30	19,62	5,68	77,5	0,0260	8,7	6051,8	5827,1
Médias		54,2				77,7		7,4	6051,2	5854,8

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
7	Cianetação	273,0	23,5	1900,6	323,4	2497,0	596,4	11,0	2,330	1,954
8	Cianetação	332,3	23,9	1743,4	401,0	2476,7	733,3	10,8	2,600	2,430
9	Cianetação	253,5	22,7	1760,8	293,3	2307,6	546,8	11,4	2,540	1,775

Condições da cianetação:
Pré - aeração / Duração de 6 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO CONCENTRADO COMPOSTO

16 HORAS PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro Teor em g/t				REAGENTES			
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)	NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
10	Cianetação	54,0	26,42	20,83	5,59	78,8	0,0520	9,1	6062,9	5620,7
11	Cianetação	54,2	26,48	20,68	5,80	78,1	0,0600	9,2	6043,1	5536,7
12	Cianetação	54,4	26,21	20,24	5,97	77,2	0,0590	8,4	6051,4	5555,9
						78,1		8,9	6052,5	5571,1

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
10	Cianetação	262,4	24,5	1580,8	308,6	2151,8	571,0	11,3	2,821	1,871
11	Cianetação	266,6	24,5	1658,9	315,9	2241,4	582,5	11,5	2,910	1,909
12	Cianetação	261,2	24,1	1597,1	311,0	2169,3	572,2	11,5	2,610	1,882

Condições da cianetação:
Pré - aeração / Duração de 16 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO CALCINADO (AS2C)

SEM PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro Teor em g/t				REAGENTES			
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)	NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
1	Cianetação	48,0	42,00	39,40	2,60	93,8	0,1954	4,9	3208,0	1093,1
2	Cianetação	49,9	41,09	38,59	2,50	93,9	0,1997	4,7	3190,3	1183,4
3	Cianetação	49,6	39,00	36,60	2,40	93,8	0,2000	4,7	3199,4	1166,3
						93,9		4,8	3199,3	1147,6

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
1	Cianetação	319,5	36,4	1744,0	295,2	2358,7	614,7	10,47	1,460	0,947
2	Cianetação	262,4	38,4	1761,2	261,1	2284,7	523,5	10,46	1,220	0,833
3	Cianetação	288,5	36	1841,3	283,8	2413,6	572,3	10,57	1,330	0,908

Condições da cianetação:
Pré - aeração / Duração de 0 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO CALCINADO (AS2C)

2 HORAS PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro Teor em g/t				REAGENTES			
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)	NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
4	Cianetação	49,4	40,02	37,47	2,55	93,6	0,2226	4,9	3200,9	922,2
5	Cianetação	49,3	38,52	36,22	2,30	94,0	0,2280	5,1	3203,5	857,4
6	Cianetação	49,3	39,67	37,22	2,45	93,8	0,2240	4,1	3208,0	905,1
						93,8		4,7	3204,2	894,9

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
4	Cianetação	268,0	36,6	1323,2	261,8	1853,0	529,8	10,7	1,280	0,838
5	Cianetação	269,5	35,2	1727,6	261,9	2259,0	531,4	10,9	1,340	0,839
6	Cianetação	281,7	36,2	1735,4	274,0	2291,1	555,7	10,7	1,130	0,879

Condições da cianetação:
Pré - aeração / Duração de 2 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO CALCINADO (AS2C)

6 HORAS PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro Teor em g/t				REAGENTES			
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)	NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
7	Cianetação	49,5	40,18	37,48	2,70	93,3	0,2100	5,8	2884,1	739,5
8	Cianetação	49,1	40,79	38,19	2,60	93,6	0,2360	5,0	3247,9	805,4
9	Cianetação	49,3	40,28	37,53	2,75	93,2	0,2380	5,1	3203,6	756,7
						93,4		5,3	3111,9	767,2

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
7	Cianetação	259,9	36,7	1740,9	254,5	2255,3	514,4	10,9	1,483	0,734
8	Cianetação	298,9	36,9	1900,7	288,8	2488,4	587,7	10,9	1,450	0,938
9	Cianetação	285,3	36,5	2046,8	277,5	2609,6	562,8	10,9	1,413	0,889

Condições da cianetação:
Pré - aeração / Duração de 6 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO CALCINADO (AS2C)

16 HORAS PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro				REAGENTES			
			Teor em g/t							
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)	NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
10	Cianetação	48,9	40,46	37,86	2,60	93,6	0,2440	6,6	3193,5	641,8
11	Cianetação	49,5	37,28	34,58	2,70	92,8	0,2515	6,4	3201,1	635,4
12	Cianetação	48,9	41,76	39,36	2,40	94,3	0,2460	6,3	3196,2	628,1
						93,5		6,4	3196,9	635,1

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
10	Cianetação	301,6	36,2	1546,5	288,4	2136,5	590,0	11,1	1,890	0,921
11	Cianetação	288,1	33,9	1522,1	282,4	2092,6	570,5	11,0	1,800	0,904
12	Cianetação	296,9	37,7	1625,5	284,4	2206,8	581,3	11,1	1,790	0,909

Condições da cianetação:
Pré - aeração / Duração de 16 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO ALIMENTAÇÃO LIXIVIAÇÃO - ALC (finos do concentrado+calcinado)

SEM PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro Teor em g/t				REAGENTES			
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)	NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
1	Cianetação	45,8	34,98	32,63	2,35	93,3	0,1712	8,3	4059,7	2036,0
2	Cianetação	47,9	31,59	29,21	2,38	92,5	0,1752	7,4	3917,7	2014,9
3	Cianetação	44,9	33,09	30,77	2,32	93,0	0,1634	6,6	4201,9	2198,7
Média	-	46,2	33,2	30,9	2,4	92,9	0,1699	7,4	4059,7	2083,2

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução	Au solução	garrafa	minério	Peso final	Peso final	Lixiviação	Consumo (g)	
		(ml)	(mg/l)	(g)	(g)	garrafa + polpa	polpa	pH final	Cal	Cianeto
1	Cianetação	277,2	27,6	1761,3	234,5	2273,0	511,7	11,8	1,937	0,952
2	Cianetação	266,4	26,9	1740,3	245,3	2252,0	511,7	11,1	1,820	0,961
3	Cianetação	290,3	25,1	1743,8	236,8	2270,9	527,1	11,2	1,560	0,995

Condições da cianetação:
Pré - aeração / Duração de 0 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO ALIMENTAÇÃO LIXIVIAÇÃO - ALC (finos do concentrado+calcinado)

2 HORAS PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro				REAGENTES			
			Teor em g/t				NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)				
1	Cianetação	47,1	32,16	29,80	2,36	92,7	0,1600	8,3	3736,8	1937,5
2	Cianetação	46,4	31,84	29,47	2,37	92,6	0,1644	8,3	3766,7	1866,9
3	Cianetação	46,3	33,13	30,75	2,38	92,8	0,1842	7,3	4303,1	2165,9
Média	-	46,6	32,4	30,0	2,4	92,7	0,1695	8,0	3935,5	1990,1

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução	Au solução	garrafa	minério	Peso final	Peso final	Lixiviação	Consumo (g)	
		(ml)	(mg/l)	(g)	(g)	garrafa + polpa	polpa	pH final	Cal	Cianeto
1	Cianetação	287,1	26,5	1546,3	255,3	2088,7	542,4	11,0	2,120	0,954
2	Cianetação	293,3	25,5	1658,6	253,8	2205,7	547,1	11,1	2,110	0,956
3	Cianetação	267,2	26,5	1625,4	230,3	2122,9	497,5	11,2	1,670	0,991

Condições da cianetação:
Pré - aeração / Duração de 2 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO ALIMENTAÇÃO LIXIVIAÇÃO - ALC (finos do concentrado+calcinado)

6 HORAS PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro				REAGENTES			
			Teor em g/t							
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)	NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
1	Cianetação	46,5	31,89	29,32	2,57	91,9	0,1880	9,2	3960,4	1798,9
2	Cianetação	46,8	27,18	24,89	2,29	91,6	0,1886	10,7	4326,1	2182,6
3	Cianetação	47,3	33,61	31,23	2,38	92,9	0,1888	9,2	3977,5	1871,7
Média	-	46,9	30,9	28,5	2,4	92,1	0,1885	9,7	4088,0	1951,1

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
1	Cianetação	296,4	25,5	1597,3	257,8	2151,5	554,2	11,2	2,360	1,021
2	Cianetação	253,0	21,9	1841,4	222,6	2317,0	475,6	11,3	2,380	0,963
3	Cianetação	297,8	28,0	1580,7	267,0	2145,5	564,8	11,2	2,450	1,062

Condições da cianetação:
Pré - aeração / Duração de 6 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos

LABORATORY CONFIRMATION TEST

CIANETAÇÃO ALIMENTAÇÃO LIXIVIAÇÃO - ALC (finos do concentrado+calcinado)

16 HORAS PRÉ LIME NA POLPA

Ensaio	Etapa	Polpa final	Ouro				REAGENTES			
			Teor em g/t				NaCN Livre (%)	Cal (Kg/t)	NaCN Adicionado (g/t)	NaCN (g/t) Consumido
		% de sólidos	Alim. calc.	Solução	Rejeito	Extração (%)				
1	Cianetação	46,2	35,28	32,90	2,38	93,3	0,2024	18,0	4032,4	1679,7
2	Cianetação	45,9	36,97	34,17	2,80	92,4	0,2060	11,5	4054,5	1627,6
3	Cianetação	45,9	36,47	33,86	2,61	92,8	0,2080	11,6	4014,2	1560,5
Média	-	46,0	36,2	33,6	2,6	92,8	0,2055	13,7	4033,7	1622,6

Entrada de dados da cianetação

Ensaio	Etapa	vol. solução (ml)	Au solução (mg/l)	garrafa (g)	minério (g)	Peso final garrafa + polpa	Peso final polpa	Lixiviação pH final	Consumo (g)	
									Cal	Cianeto
1	Cianetação	272,7	28,3	2046,2	234,6	2553,5	507,3	11,2	4,220	0,946
2	Cianetação	289,7	29	1521,8	245,9	2057,4	535,6	11,1	2,830	0,997
3	Cianetação	283,0	28,7	1900,5	239,9	2423,4	522,9	11,3	2,790	0,963

Condições da cianetação:
Pré - aeração / Duração de 16 horas
24 horas cianetação
pH > 11.0 (10,5 -11,0)
50% sólidos