Estimation of the potential for the formation of acid rock drainage of the talc deposit Rabenwald

Thesis submitted for the degree of Master of Science

Diplomarbeit zur Erlangung des akademischen Grades eines
Diplomingenieurs

Stefan Neumeister

November, 2009

Supervisors

Ao. Univ.-Prof. Dr. phil. Walter Prochaska(1)
Dr. Michaela Wurm(2)

(1) Department for Applied Geosciences and Geophysics
Chair of Geology and Economic Geology, University of Leoben, Austria
(2) Rio Tinto Minerals

EIDESSTATTLICHE ERKLÄRUNG

Ich erkläre an Eides statt, dass ich diese Arbeit selbstständig verfasst, andere als die angegebenen Quellen und Hilfsmittel nicht benutzt und mich auch sonst keiner unerlaubten Hilfsmittel bedient habe.

AFFIDAVIT

I declare in lieu of oath, that I wrote this thesis and performed the associated research myself, using only literature cited in this volume.

Leoben, November 2009 Stefan Neumeister

TABLE OF CONTENT

1. **ABSTRACT** 5

2. **ZUSAMMENFASSUNG** 6

3. **INTRODUCTION** 7

4. **ACID ROCK DRAINAGE** 8

4.1 GENERAL INTRODUCTION 8
4.2 ACID GENERATION DUE TO THE OXIDATION OF PYRITE 10
4.3 BIOCHEMICAL AND GEOCHEMICAL INFLUENCE ON THE PYRITE OXIDATION 11
4.4 ACID CONSUMING PROCESSES 13
4.5 NET ACID GENERATION POTENTIAL AND REACTION KINETICS 16
4.6 MIGRATION OF THE ARD 16
4.7 INFLUENCE OF THE ARD ON THE ENVIRONMENT 17
4.8 ACID GENERATION PREDICTION 17
4.9 ACID GENERATION AND LEACHATE MIGRATION CONTROL MEASURES 18
4.10 IN-WASTE MONITORING 18

5. **GEOLOGY AND HYDROGEOLOGY** 20

5.1 GEOLOGICAL OVERVIEW 20
5.2 CLIMATE, HYDROLOGY AND HYDROGEOLOGY 24

6. **ANALYTICAL METHODS** 34

6.1 SAMPLING 34
6.2 PREPARATION AND TREATMENT OF THE SAMPLES 34
6.3 THIN SECTIONS, POLISHED SECTIONS 36
6.4 LECO ANALYSIS 36
6.5 LEACHING TESTS 38
  6.5.1 General Introduction 38
  6.5.2 Leaching Test of the Waste Rock Sample (LT 1) 39
  6.5.3 Leaching Test of the ‘artificial’ Bulk Samples (LT 2) 40
  6.5.4 Leaching Test Actinolite Gneiss (LT 3) 41
  6.5.5 Leaching Test Acid Rock Drainage (LT 4) 42

6.6 ION CHROMATOGRAPHY 43

7. **RESULTS** **46**

7.1 PETROGRAPHY 46
7.1.1 Transmitted Light Microscopy 46
7.1.1.1 Leucophyllite/"Kornstein" (LP 1, LP 4) 46
7.1.1.2 Marble (MA 1, MA 2) 47
7.1.1.3 Granite Gneiss (GG 1, GG 2) 48
7.1.1.4 Paragneiss (PG 1) 50
7.1.1.5 Hornblende Gneiss (HG 2, HG 4) 51
7.1.1.6 Calcsilicate (CS 1, CS 3) 52
7.1.1.7 Actinolite Gneiss (AG 1, AG 3, AG 6) 54
7.1.2 Ore Microscopy 57
7.2 LECO ANALYSIS 60
7.3 LEACHING TESTS 62
7.3.1 Leaching Test of the Waste Rock Sample (LT 1) 62
7.3.2 Leaching Test of the 'artificial' Bulk Sample (LT 2) 65
7.3.3 Leaching Test of Actinolite Gneiss (LT 3) 68
7.3.4 Leaching Test Acid Rock Drainage (LT 4) 70
7.4 ION CHROMATOGRAPHY 72
7.5 OPEN PIT MINE WATER QUALITY 74

8. **DISCUSSION** **87**

9. **REFERENCES** **105**

10. **APPENDIX** **109**

1. ABSTRACT

The aim of this work was an estimation of the potential of the Rabenwald talc deposit regarding the formation of acid rock drainage (ARD). ARD is a term which describes leachate, seepage and drainage that can be caused by the activity of the oxidation reaction of sulphide minerals with water and oxygen.

The methods applied in the scope of this work were: microscopy, Leco analysis, leaching tests and ion chromatography.

Of all samples of the deposit investigated only actinolite gneisses and calcsilicates showed elevated contents of sulphur and hence the theoretical potential for ARD formation. The sulphides contained in these rocks are pyrrhotite, chalcopyrite, and pyrite.

Kinetic tests like leaching tests make it possible to find evidence about the rate of acid generation and the amount of acid that will ultimately be produced. In addition, changes in the ion concentrations of the applied solutions can give information about the activity of the ARD process. All in all four leaching tests with rock samples with different compositions were carried out. The results of the tests are clear. According to these, acid generation through actinolite gneisses and calcsilicates takes place in the open pit mining and the waste rock management units. The acid produced is neutralised by alkaline rocks on site however. Consequently no migration of acidic solutions occurs.

Besides acidity, sulphate is a product of the oxidation of sulphides. The analysed basis fertiliser solutions of the leaching tests showed increasing sulphate concentrations until the end of the tests. This circumstance explains the elevated sulphate concentrations of the open pit mine waters at the measuring points of the Rabenwald. Those are thus a product of the oxidation of sulphides in actinolite gneisses and calcsilicates.

Therefore the obtained results cannot allow one to assume potential for the formation of ARD at the Rabenwald talc deposit as generated acid is neutralised in situ. Sulphate, as a product of the oxidation process, raises problems however since it migrates with seeping open pit mine waters and therefore leads to elevated concentrations at the measuring points.

2. ZUSAMMENFASSUNG

Ziel dieser Arbeit war eine Bewertung des Potentials der Talklagerstätte Rabenwald hinsichtlich der Bildung von Acid Rock Drainage (ARD). ARD beschreibt Laugung, Versickerung und Entwässerung, welche durch den Ablauf von Oxidationsreaktionen von Sulfidmineralen mit Wasser und Sauerstoff verursacht werden kann.

Die im Rahmen dieser Arbeit angewandten Methoden waren: Mikroskopie, Leco-Analysen, Laugungsversuche und Ionenchromatographie.

Von allen untersuchten Proben der Lagerstätte ergaben sich nur für Aktinolithgneise und Kalksilikate erhöhte Schwefelkonzentrationen und somit das theoretische Potential zur Formation von ARD. Die in diesen Gesteinen enthaltenen Sulfide sind Magnetkies, Kupferkies und Pyrit.

Kinetische Tests wie Laugungsversuche ermöglichen es, Aussagen über die Rate der Säuregenerierung und die Menge an schlussendlich gebildeter Säure zu treffen. Außerdem können auch die Veränderungen der Ionenkonzentrationen der verwendeten Lösungen Aufschlüsse über den Ablauf des ARD-Prozess geben. Es wurden insgesamt vier Laugungsversuche mit Gesteinsproben unterschiedlicher Zusammensetzung durchgeführt. Die Ergebnisse der Versuche sind eindeutig. Demnach erfolgt Säuregenerierung durch Aktinolithgneise und Kalksilikate im Tagebau und in den Halden. Die gebildete Säure wird aber in situ durch alkaline Gesteine neutralisiert. Es kommt folglich zu keiner Migration von acidischen Lösungen.

Neben Acidität ist Sulphat ein Produkt der Oxidation von Sulfiden. Die untersuchten Stammlösungen der Laugungsversuche zeigen bis zum Versuchsende steigende Sulphatkonzentrationen. Dieser Umstand erklärt die erhöhten Sulphatkonzentrationen der Tagebauwässer an den Messstellen des Rabenwalds. Diese sind demnach ein Produkt der Oxidation von Sulfiden in Aktinolithgneisen und Kalksilikaten.

Die erhaltenen Ergebnisse lassen also kein Potential der Talklagerstätte Rabenwald für das Auftreten von ARD vermuten da generierte Säure in situ neutralisiert wird. Sulphat, als Produkt des Oxidationsprozesses, verursacht allerdings Probleme da es mit den versickernden Tagbauwässern migriert und folglich zu erhöhten Konzentrationen an den Messstellen führt.

3. INTRODUCTION

The Rabenwald talc deposit is located in the east of Styria on an elevation between 900 and 1080 m. Some 120,000 t of talc is mined in the open pit mining operations annually. The Rabenwald surface mining is operated by the company Luzenac Naintsch, which is a part of the internationally active Luzenac group.

The quality of open pit mine waters has been measured at Rabenwald since 1999 at a total of 17 locations. Beginning in 2000/2001 a significant increase of the sulphate concentrations at the measuring points has been recorded. Yet in 2001 the tolerable value of 250 mg/l SO_4 according to the regulatory guideline for drinking water was considerably exceeded at no fewer than six measuring points. This trend of elevated concentrations has continued to today (Nov. 2009) and thus caused potential problems regarding the use of springs and wells for drinking water supply.

The formation of acid rock drainage (ARD) was assumed by Luzenac Naintsch staff as a possible reason for the elevated sulphate concentrations. In the course of the ARD process the reaction of sulphide minerals with oxygen and water generates acidity and sulphate as products.

The task of the work at hand therefore was an estimation of the potential of the Rabenwald deposit for the formation of ARD. For this purpose sampling was carried out twice in 2007 at the open pit. In the process representative samples of the eight rock types present at the open pit mining, as well as one waste rock sample, were obtained. Furthermore 'artificial' bulk samples on the basis of a drillhole database, which should reflect the composition of the entire surface mining, were prepared in the laboratory. The following methods were applied for the analysis of the different samples: microscopy, Leco analyses, leaching tests, ion chromatography.

Particular attention was put on the leaching tests, as those provide information on the rate of sulphide oxidation and hence acid generation, as well as on the quality of produced solutions. For the interpretation of data regarding acid generation, the interaction between acidity, loss of alkalinity, pH and sulphate is essential. This information, which is obtained from the tests mentioned permit a reliable estimation of the ARD potential of the Rabenwald talc deposit.

4. ACID ROCK DRAINAGE

This chapter is based on and includes quotations of the textbook MINE WASTE MANAGEMENT. This book was composed by the authors HUTCHINSON and ELLISON, was sponsored by the California Mining Association and was published by the company Lewis Publishers in the year 1992. The findings of these authors that were taken from the book are not marked extra in the text. Sources that were used by HUTCHINSON and ELLISON are mentioned with quotations in this chapter.

4.1 General introduction

Acid rock drainage (ARD) is a term used to describe leachate, seepage or drainage that has been affected by the natural oxidation of sulphide minerals contained in rocks, which are exposed to air and water. The potential occurrence of ARD is basically determined by three factors: the presence of sulphide minerals in the rock and the availability of oxygen and water. Additionally, biological activity is a factor that can speed up and reinforce this process.

ARD need not imperatively occur when sulphide-containing rocks are exposed. Arid climate, non-reactive sulphide minerals as well as alkaline rocks (for acidity buffering) can prevent or minimize the formation of ARD. The factor time must be considered as important too. Thus a waste rock management unit may contain alkaline rocks with a certain buffer capacity for generated acid. If this capacity has been used up, under certain circumstances years or even decades after close-down of the mine, ARD may still occur.

Therefore, in order to be able to make predictions about acid generation in surface mining, the interaction of acidity, loss of alkalinity, change of the pH value and built sulphates must be considered, because these are the products of sulphide oxidation.

ARD caused by mining activities can have different sources:

- Waste rock management units.
- Ore stockpiles.

- Tailings waste management units.
- Underground mine workings.
- Open mine pits.
- Spent heap leach piles.

ARD develops when the following conditions are fulfilled:

- The waste rock material contains sulphides which react chemically and biologically to acid leachate. This happens to an extent that alkaline rocks contained in the dump cannot neutralize the generated acid.

- The physical characteristics of the dump must allow sufficient quantities of oxygen and water to flow through the rock material in order to enhance the chemical and biological reactions.

- There is enough precipitation, so that the infiltrating rainwater can transport the generated acidic drainage out of the waste rock management unit into the surroundings. A second possibility of transport is provided by surface water which runs through the dump.

Of the sources mentioned above, above all the potential development of ARD in waste rock management units and open mine pits are relevant concerning the Rabenwald deposit. The specific problems in this context are briefly described in the following.

The development of ARD is possible in waste rock management units which show significant portions of sulphide-containing rocks. This is particularly true for humid climates, where sufficient water in the forms of rainwater, surface runoff and ground water exists in order to transport the products of the acid generation.

In open pits it is particularly the large, exposed rock surface, the drawdown of the ground water level and the continuous clearing of fresh rock which poses problems concerning the generation of ARD.

4.2 Acid generation due to the oxidation of pyrite

Basically the acid generation process can be described with the following rule of thumb: ‘pyrite plus water plus oxygen forms acid’. However, this formula only presents a simplification of a complex chemical and biological process.

Pyrite is a sulphide mineral occurring frequently in deposits; therefore its oxidation is a main cause of acid generation. Additionally, further aspects must be considered like the oxidation of other metal sulphides or acid generation through anaerobe processes.

The oxidation of pyrite can either happen directly, through the reaction of the mineral with air and water, or indirectly, through the Fe^{3+} ion. The direct oxidation is considered decisive for the generation of ARD in deposits.
The oxidation of pyrite is determined by four reactions which are defined by the pH value, the sulphate and iron concentrations and the acidity:

1. $2\ FeS_2 + 7\ O_2 + 2\ H_2O = 2\ Fe^{2+} + 4\ SO_4^{2-} + 4\ H^+$
2. $4\ Fe^{2+} + 10\ H_2O + O_2 = 4\ Fe(OH)_3 + 8\ H^+$
3. $2\ Fe^{2+} + O_2 + 2\ H^+ = 2\ Fe^{3+} + H_2O$
4. $FeS_2 + 14\ Fe^{3+} + 8\ H_2O = 15\ Fe^{2+} + 2\ SO_4^{2-} + 16\ H^+$

The above reactions occur in three distinguishable stages (with falling pH value):

Stage 1 is defined by a pH value of >4.5, high sulphate and low iron concentrations and a low acidity level. The first reaction equation works abiotically and also through direct bacterial oxidation. Reaction n° 2 also works abiotically and slows down with falling pH value.

The chemical conditions for stage 2 are defined by a pH between 2.5 and 4.5, high sulphate values, acidity, increased iron contents and a low Fe^{3+}/Fe^{2+} ratio. The processing of reaction n° 1 does not change compared to stage 1. The rate of the second reaction depends on the activity of the *Thiobacillus ferro-oxidans* (*T. ferro-oxidans*) bacterium.

Stage 3 is characterized by a pH value <2.5, high sulphate levels, acidity, high iron concentrations and a high Fe^{3+}/Fe^{2+} ratio. Reaction n° 3 is here determined by the activity of *T. ferro-oxidans*. Reaction 4 in turn depends on the activity of reaction 3.

According to WILLIAMS et al. (1982), the indirect oxidation (reaction n° 4) of the pyrite is caused by dissolved trivalent ferric iron (Fe^{3+}). This kind of acid generation from pyrite takes place without oxygen.

4.3 Biochemical and geochemical influence on the pyrite oxidation

It is important to consider pyrite's dependence on several biological and chemical processes. These are described in the following.

The biochemical influence on the oxidation process was – amongst others – examined by WALSH and MITCHELL (1975). According to this study, the oxidation of pyrite is controlled chemically until a pH value of 3.5 to 4. If it falls below this pH value, the *T. ferro-oxidans* bacterium catalyzes the process. According to KNAPP (1987), the rate of the biochemical reaction is quicker at pH values of about 3 to 3.5 than the rate of the abiotical reaction. The biochemical influence must not be underestimated, because the *T. ferro-oxidans* bacterium, which catalyzes the process, is wide-spread in natural systems. However, low temperatures limit the efficiency and the frequency of *T. ferro-oxidans*.

The following geochemical factors influence the oxidation of the pyrite: the pH value, the availability of oxygen, the alkalinity, the frequency of pyrite in the rock, the formation of the pyrite minerals, the temperature and microenvironments. WILLIAMS et al. (1982) directly links the initial acid generation at a pH value of >4.5 with the total acid generation potential, which is under certain circumstances used up only after several years or decades. According to this study, a high initial rate of acid generation will, in the end, favour the generation of bigger amounts of acid.

The role of the pH value within the acid generation process is not clearly defined. It has not been clearly found out whether the pH value can decisively control or even prevent the oxidation of the pyrite. Maybe it only affects the oxidation rate or serves to announce buffering processes. The following examples shall illustrate the different views on this topic.

KNAPP (1987) declares proportionality between the rate of chemical oxidation and the pH value. According to this theory, the rate of oxidation sinks up to 50% at falling pH value. WILLIAMS et al. (1982) on the other hand, referencing in turn works of KLEINMANN et al. (1981), states a very slow processing of the oxidation reaction at a pH value of 6 to 8, with production of insignificant amounts of acid. Data from field tests confirm the second theory.

Oxygen is a prerequisite for the direct oxidation of the pyrite after reaction n° 1. The amount of oxygen needed is very small, being less than 10% of the oxygen content of the air. Water can as well serve as a source for oxygen in liquid form.

The influence of the alkalinity on the pyrite oxidation was described by WILLIAMS et al. (1982). He assumed three systems with different buffering capacity and water characteristics.

A system without alkalinity, thus without buffering capacity, under influence of oxygen-saturated water, reaches a final pH value of approximately 3.2 after the oxidation process. If calcite as a source of alkalinity is present, pyrite will be oxidized through oxygen, but the pH value remains at about 7.4 after the reaction. The produced acid is thus neutralized. According to BUCKMANN and BRADY (1960), the carbon dioxide concentration in the ground water is up 10 to 100 times higher than in surface water. Consequently, in this case an even more distinct alkalinity and thus a pH value of >8 can be expected. Summing up, it can be said that alkaline minerals maintain the pH value constant around 7 to 8, and consequently, according to the theory by WILLIAMS et al. (1982) described above, they can significantly slow down the pyrite oxidation.

According to CARUCCIO and GEIDEL (1981), there is a relation between the amount of pyrite in a rock, the exposed surface area of pyrite grains and the amount of acid produced in a dump. In other words, the rate of acid generation in a waste rock management unit directly depends on the amount and the exposed surface areas of the contained pyrite minerals. Consequently, also the formation of the pyrite in the rock is important. Finely spread, disseminated grains have a bigger surface, thus offer a bigger reaction surface and are consequently a potentially better acid generator than massive grains. Furthermore the extent of encapsulation of the mineral grains has to be considered. Pyrite in a massive, unweathered silicate matrix may be protected from the influence of water and oxygen and may not react as a consequence.

The temperature significantly influences the process of chemical and biochemical reactions. Thus a raise of the temperature by 10°C corresponds to a doubling of the pyrite oxidation rate (SMITH and SHUMATE, 1971).

The rock material in a waste rock management unit is often heterogeneous and irregular. The results are different geochemical characteristics and the appearance of different microenvironments. The geochemical characteristics of these microenvironments influence the acid generation of the dump. This means that a waste rock management unit can show local acid generation even if it does not have the potential for the formation of ARD in general.

SILVER (1987) examined the biochemical oxidation of other metal sulphides. Similar to pyrite, these are oxidable either directly through bacteria in presence of air and water, or indirectly through Fe^{3+} ions

4.4 Acid consuming processes

Acid generation in mine waste must not occur obligatorily, even if reactive sulphides are contained and if there is sufficient water to transport the acid leachate. There are two ways to prevent acid: by reaction with other components contained in the dump, by water which streams through the rock material. The reactions most important for these processes are:

- Reactions with carbonates.
- Reactions with alumosilicates.
- Reactions with, or formation of, other compounds.

Carbonate minerals are widely-spread in many rocks and therefore play an important role in the preventing of acid generation. The carbonate minerals can hereby react in solution or as solid phase. For calcite and dolomite as solid-phase carbonates, the acid consuming reactions can be written as follows:

$$2\ H^+ + CaCO_3 = Ca^{2+} + CO_2 + H_2O \text{ (calcite)}$$

$$2\ H^+ + [CaMg(CO_3)_2] = Ca^{2+} + Mg^{2+} + CO_2 + H_2O \text{ (dolomite)}$$

Under saturated conditions, the gaseous carbon dioxide cannot escape. In this case the reactions for calcite and dolomite under formation of bicarbonate ions are as follows:

$$H^+ + CaCO_3 = Ca^{2+} + HCO_3^- \text{ (calcite)}$$

$$2\ H^+ + [CaMg(CO_3)_2] = Ca^{2+} + Mg^{2+} + 2\ HCO_3^- \text{ (dolomite)}$$

The combined reaction for the formation of acid through the oxidation of pyrite through oxygen and water and the simultaneous acid consuming reaction with calcite was described by WILLIAMS et al. (1982) as follows:

$$4\ FeS_2 + 8\ CaCO_3 + 15\ O_2 + 6\ H_2O = 4\ Fe(OH)_3 + 8\ SO_4^{2-} + 8\ Ca^{2+} + 8\ CO_2$$

From this equation it can be deduced that 2 moles of calcite are needed to neutralize the generated acid from the oxidation of 1 mol pyrite. This corresponds to 200 g calcite for 120 g pyrite.

The product side of the above reaction equation shows a circumstance which can confine the process of acid consuming of the calcite. Through precipitation of the iron hydroxide and gypsum built from the reaction, a ‘coat’ can evolve around the carbonate minerals which can gravely affect the neutralisation reactions. This in turn can lead to an underestimation of the acid generation, through assumption of a too big neutralisation potential. What makes it even more difficult is the fact that the surface of the sulphide minerals and consequently the acid generation are not that severely affected by this ‘coating’.

Next to the carbonates, it is mostly the alumosilicates which play an important role at the neutralization of acid. The most important traits hereby are:

- Alumosilicates in contact with water tend to produce an alkaline pH.
- Alumosilicates dissociate at contact with acid, consume hydrogen ions and build clay minerals.
- Some silicates, particularly alumosilicates like mica, clay minerals, etc. can remove hydrogen ions by ion exchange.

Acid consuming processes involving silicates are not as effective as those involving carbonates. However, according to WILLIAMS et al. (1982) these processes are of high importance when the acid generation runs more slowly, because of the enormous amounts of silicate minerals in waste rock material and natural environments. The reaction equations for the acid consuming process for calcic and potassic feldspar are (LAPAKKO, 1987; WILLIAMS et al., 1982):

$$2\ KAlSi_3O_8 + 2\ H^+ + H_2O = Al_2Si_2O_5(OH)_4 + 4\ SiO_2 + 2\ K^+ \text{ (calcic feldspar)}$$

$$CaAl_2Si_2O_8 + 2\ H^+ + H_2O = Al_2Si_2O_5(OH)_4 + Ca^{2+} \text{ (potassic feldspar)}$$

In both cases a cation (calcium, potassium), which remains in solution, as well as an Al-rich clay mineral are created on the product side. The processes run relatively slowly according to BUSENBERG and CLEMENCY (1976).

To sum up, it can be stated that the content of carbonate minerals as well as the content of alumosilicates in dump material considerably influences the prediction of acid generation. This is especially relevant for systems with slow acid producing rates.

4.5 Net acid generation potential and reaction kinetics

The processes of acid generation and of acid consumption described above can be subsumed under the term net acid generation potential of rock material. The net acid generation potential of a rock material describes the difference between the acid built by oxidation of sulphide minerals and the acid used up by neutralisation reactions at a certain point of time or during a certain period. Consequently, through this approach the net rate of acid generation is only indirectly affected due to the whole acid generation potential. Because of that, the rate of the acid generation and acid consumption, or the reaction kinetics, become most important for the determination of the net acid generation potential. A system can, for example, initially possess enough alkalinity to neutralize the produced acid. However, after some time this alkalinity may have been used up, and an acidic system is formed. For this reason, middle- to long-term kinetic tests must be applied, in order to gain potentially plausible values when estimating the net acid generation potential.

4.6 Migration of the ARD

In case a mine waste has acid generation potential, the next step is to consider the migration of the acid rock drainage (ARD). The formed ARD has a low pH and can release heavy metals from the mine waste and carry them along as pore water. This polluted pore water has a great influence on the environment if it migrates from the dump and runs off in surrounding aquifers or flows off as surface water. The migration of ARD and the solubility of metals are controlled by different physical, chemical and biological characteristics of the waste rock material. For the mobilisation of the metals, it is particularly chemical and, to a certain degree, also physical and biological factors that play a role. The controlling processes of the ARD migration on the other hand are of physical and biological origin. These factors determining the migration of ARD are briefly described in the following:

The physical traits of the dump material mainly influence the migration of the ARD. The amount of infiltrating rainwater, the permeability of the waste rock material, the availability of pore water and the pore water pressure are the factors that have to be considered. Furthermore, the way the ARD is transported is of relevance. The transport can take place by means of discrete channel flow at very good porosity, as porous media flow and/or as molecular diffusion. The way of flowing controls the migration rate, as well as the dilution and the mixing of the ARD. The physical influence on the solubility of the metals is restricted to grain size, grain form, and the surrounding temperature.

The solubility of metals is controlled chemically, especially through the pH value. If the pH sinks, more soluted metals are carried along in the pore water. Further factors are the redox potential, adsorption characteristics of the minerals, the structure of the mobilised metal and the composition of the ARD. The chemical traits of the waste rock material determine the amount of generated acid and the rate of neutralisation reactions, and thus the quality of the water flowing off.

The biological influence on the migration is of less importance. Alongside the migration route, bacteria can cause metal leaching from sulphides; adsorption and precipitation of metals can occur as well.

4.7 Influence of the ARD on the environment

Systems have of course a certain potential to neutralize the ARD. The ARD can be neutralized on the migration route by alkaline rocks, or it can be diluted by waters. The influence the ARD has on the environment is a function of time and depends on the quality of the formed ARD, the dilution and neutralisation capacity of the bedrock and soils surrounding the deposit, and the quantity and quality of potentially influenced waters.

4.8 Acid generation prediction

The prediction of the amount of acid generation is immensely important in mining. Based on that, precaution and protection measures can be taken to prevent the formation or the migration of ARD. FERGUSON and ERICKSON (1986) differ between five different methods to predict the formation of ARD. In reality only two methods, static and kinetic tests are applied. The test method used to compile this thesis, including the sampling, is described in the chapter ANALYTICAL METHODS.

4.9 Acid generation and leachate migration control measures

For minings with a potential for the formation of ARD, controlling measures are very important. They should if possible prevent the ARD process or at least limit it afterwards. Basically it is easier to prevent the formation of ARD from the beginning than to constrict it afterwards. There are three different approaches to prevent or limit acid generation and leachate migration:

- Control of the acid generation process.

- Control of the ARD migration.

- Collection and treatment of the acid drainage.

The possibilities above are listed in the order of their preferred application. Logically, the acid generation process is controlled most effectively if the formation of ARD is prevented. If this cannot be done any more, it must be attempted to prevent the formation of an eluate or at least its migration from the waste management unit. The third method finally is applied if none of the earlier control mechanisms works. The ARD control mechanism which is most effective for a certain mining is determined by kinetic tests. Hereby, the effectiveness of the planned method is tested in the lab under simulation of the natural conditions.

4.10 In-waste monitoring

The permanent monitoring of mine waste management units is a precondition to gain up-to-date data about acid generation, or about the effectiveness of control measures to prevent the formation of ARD. The targets of in-waste monitoring regarding ARD are thus the following:

- Provide information about the functioning of the control mechanism.

- Provide early hints about possible problems, so that the existing control mechanism can be modified.

- Provide information in order to improve the existing control mechanism if necessary.

In-waste monitoring uses several parameters to detect products of the acid generation process:

- Temperature: the oxidation of sulphides produces heat. Changes of temperature can consequently hint at acid generation.

- Gases: oxygen is used up at the production of acid. The accumulation of oxygen in the gas phase can thus indicate acid generation.

- Dilution products: different soluble products like acidity, sulphates, iron, and different secondary trace metals can hint at acid generation.

Through permanent control of these parameters, one is able to detect acid generation early and to begin with appropriate countermeasures.

5. GEOLOGY AND HYDROGEOLOGY

5.1 Geological overview

In Styria there are two places where talc (Rabenwald) or leucophyllite (Weißkirchen bei Judenburg) is mined. The Rabenwald surface mining, whose rocks were examined in the context of this thesis, is situated at the border of the regions of Weiz and Hartberg, at the eastern edge of the Alps.

From a geological point of view, the Rabenwald area belongs to the Lower Austro Alpine Unit (LAA). The LAA is characterized by a north oriented relief with polymetamorphic cores and permomesozoic covers (FLÜGEL, 1984). In the Styrian region, this structure of nappes is subordinated by not exposed penninic units and overlaid by the Middle Austro Alpine Unit (MAA) as well as the Upper Austro Alpine Unit (UAA) (TOLLMANN, 1977). The LAA is framed in the north and the west by the Greywacke zone and the Palaeozoic of Graz (both belonging to UAA). In the south and in the east the rocks of the LAA are overlaid by neogenic sediments. In the area of Rechnitz, in the Burgenland region, penninic rocks emerge in the Rechnitzer Fenster. Remains of the MAA overlie the Grobgneiss Unit in form of eroded thrust outliers (PROCHASKA, 1988). The MAA thins out on the eastern end of the Alps and misses in parts (TOLLMANN, 1977). Figure 5-1 shows a tectonic map of the position of the LAA on the eastern end of the Alps according to WIESENEDER (1971) and TOLLMANN (1976, 1977). Figure 5-2 shows a geological map of the eastern end of the Alps (according to SCHUSTER, 2001; out of WURM, 2005).

The MAA can be divided into the deeper Wechsel and into the higher Semmering system. The Semmering system can further be categorized into the Stuhleck-Kirchberg, the Mürz-Tachenberg and the Roßkogel nappe (FLÜGEL, 1984). The rocks appearing at the Rabenwald are part of the Stuhleck-Kirchberg nappe and can, according to FRIEDRICH (1947), be divided into two different series:

- Grobgneiss Unit

- Strallegger Gneiss and Schist Unit

The Grobgneiss Unit is mainly composed of granite and coarse-grained granite gneisses of different compositions and different strains, as well as of micaschists. These rocks build up lenses, blocks, and bigger masses and are widely-spread at Rabenwald (FRIEDRICH, 1947). The Grobgneiss Unit is covered by low metamorphic, permomesozoic sediments. Phyllites at the basis are followed by greywackes, metabreccia and porphyroides (KIESL, KLUGER and WIESENEDER, 1986).

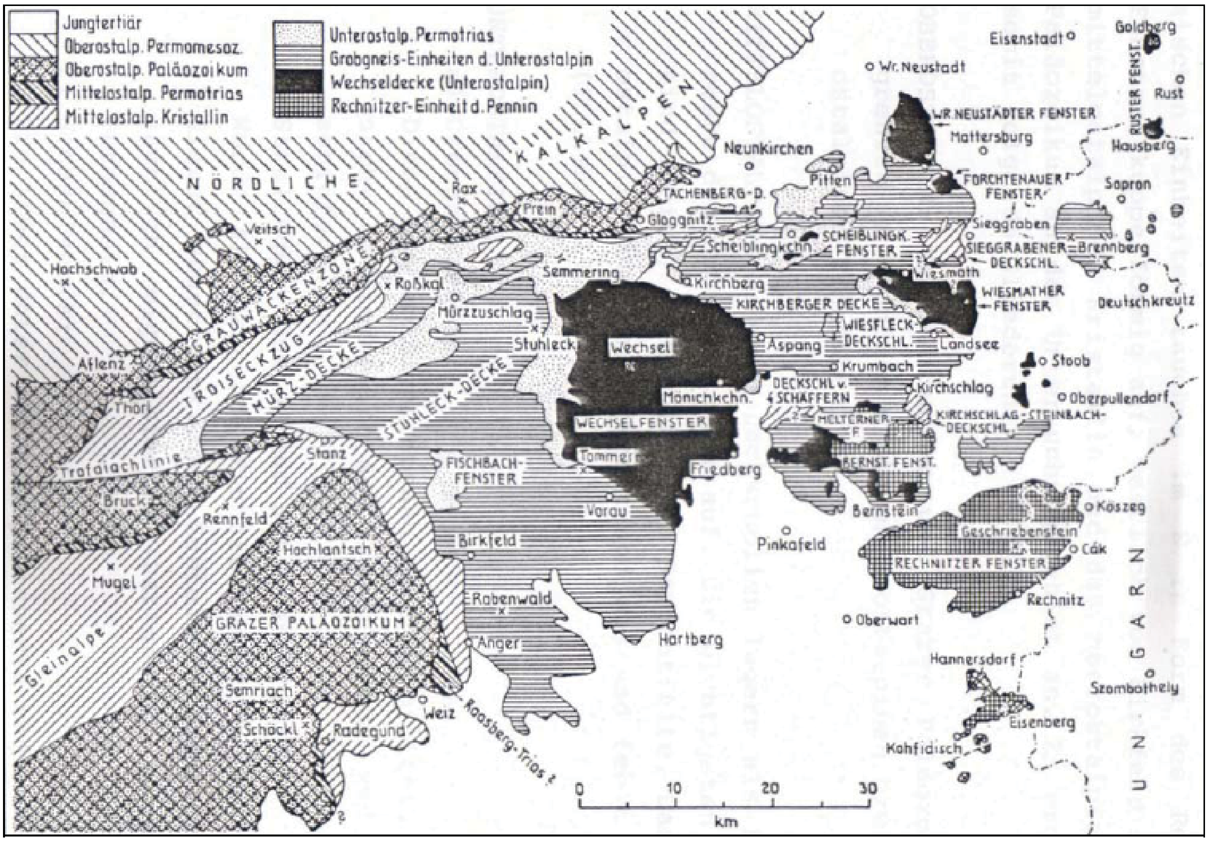

Figure 5-1: Tectonically sketch of the position of the Lower Austro Alpine Unit (LAA) at the eastern end of the Alps according to WIESENEDER (1971) and TOLLMANN (1976, 1977).

The “Strallegger Gneisses” of the Strallegger Gneiss and Schist Unit are partly migmatic biotite schists and biotite gneisses (FLÜGEL, 1984) which overlie the rocks of the Grobgneiss Unit (FRIEDRICH, 1947). According to FRIEDRICH (1947) furthermore amphibolites, amphibole gneisses and in contrast to the carbonate-free Grobgneiss Unit also lenses of marbles and massy calcsilicate rocks occur in the

Strallegger Gneiss and Schist Unit. A particularly striking rock of this unit is the leucophyllite or “Kornstein” (see also chapter PETROGRAPHY) which often appears in the vicinity of the talc ore body and is widely distributed at Rabenwald deposit (FRIEDRICH, 1947). This rock type forms on the one hand long veins at tectonically stressed zones and on the other hand boundary layers, mainly in the hanging wall of granitic rocks (FRIEDRICH, 1947).

The rocks of the LAA experienced an alpidic metamorphose in greenschist-facies which transformed a variscian metamorphose event of amphibolite-facies that led to local anatectic melting (FLÜGEL, 1984). The alpidic orogeny caused an intensive retrograde transformation of the predominantly variscian mineral content as well as the formation of faults and shear zones where hydrothermal alteration of the rocks and widespread formation of leucophyllite took place (PROCHASKA, 1988). According to PROCHASKA (1988) the occurrence of magnesite relics in the deposit is particularly important as they are the Mg-source that permitted talc formation exactly and only at Rabenwald. The deposit is thereby located at the boundary between the lower Grobgneiss Unit, which consists of augengneiss, orthogneiss and metabasite, and the higher Strallegger Gneiss and Schist Unit which is assembled of “Strallegger gneiss” (biotite-paragneiss), “Tommer schist” (garnet-micaschist) and metagranite (PROCHASKA, 1988).

The talc mineralisation (talc and leucophyllite) is bound to a large scale shear zone in the LAA and follows this highly tectonized zone of alpidic age (WURM, 2005). According to WURM (2005) the shear zone dips about 8 to 10° to the south and has an average thickness of about 20 to 60 m. As already mentioned above, the mineralization was effected by hydrothermal alteration of the host rocks to talc, at which existing magnesite relics and dolomites served as Mg-source (PROCHASKA, 1988). Furthermore, amphibolites and amphibole gneisses (actinolite gneisses) that can contain considerable amounts of sulphide minerals are encountered in the ore body as well. The ore itself is a mixture of talc and chlorite in equal parts (WURM, 2005).

Different assumptions and theories were established regarding the genesis of Rabenwald talc deposit in the course of the last decades.

FRIEDRICH (1947) and MOREAU (1981) reasoned a metasomatic formation of the talc by Mg-rich hydrothermal fluids.

PROCHASKA (1989) tested several Mg-carbonate-hosted and Mg-ultramafic-hosted deposits for their specific geochemistry. As the two deposit types vary significantly in respect to their element distribution, it was possible for him to classify the Rabenwald as Mg-carbonate-hosted talc deposit. The talc forming process with magnesite as source material proceeded as follows (PROCHASKA, 1988):

$$3\ MgCO_3 + 4\ SiO_2 + H_2O = Mg_3Si_4O_{10}(OH)_2 + 3\ CO_2$$

SCHROLL et al. (1986) proved the sedimentary origin of the magnesite relics in the ore body by means of their low content of Fe^{2+}. Magnesites of metamorphic origin have higher FeO-contents of about 4 to 5% (WURM, 2005).

The magnesites were tectonically transported into a complex fault system (PROCHASKA, 1988). There, the metamorphic-hydrothermal transformation of the magnesites to talc and the alteration of the host rocks to leucophyllite syntectonically took place (WURM, 2005).

KIESL et al. (1983) determined with a value of 80 Ma the alpidic age of metamorphose on the basis of K/Ar dating. There are different opinions concerning the formation temperature and pressure of the talc schists. HERITSCH (1967) emanates formation temperatures from 450-500°C; MOINE et al. (1989) assumes temperatures of 500-550°C and pressures of 8 to 9 kbars; BELOCKY (1992) on the other hand supposes temperatures >350°C and pressures >3 kbars for this process.

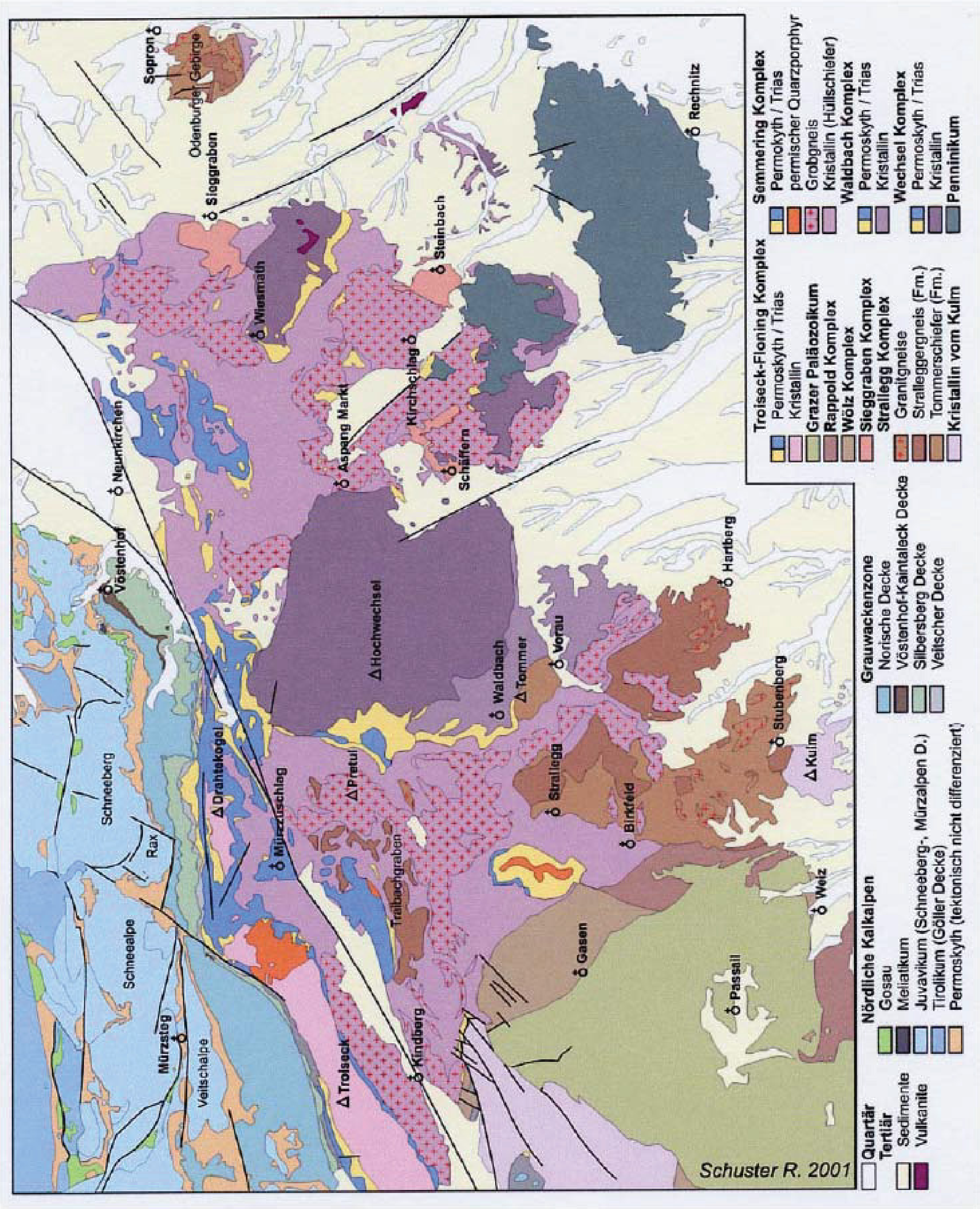

Figure 5-2: Geological map of the eastern end of the Alps (according to SCHUSTER, 2001; out of WURM, 2005).

5.2 Climate, hydrology and hydrogeology

The formation of acid rock drainage (ARD) is mainly determined by three factors. Besides the occurrence of sulphide minerals and oxygen, the availability of sufficient water is a prerequisite for the production of acidic drainage. Thus for a reliable determination of the ARD potential of a deposit, the hydrologic and hydrogeologic characteristics as well as the climate of the investigated area are of particular importance. According to HUTCHINSON and ELLISON (1992), these factors above all have a great influence on the migration of the produced ARD. The migration of the acidic drainage is mainly determined by the rate of rainfall infiltration, the availability of pore water, pore water pressure, the flow mechanism and the flow velocity of pore water, the degree of dilution of the acidic drainage and the location of ground water tables. In addition, the temperature as a further climatic factor influences the rate of chemical and biochemical processes.

In general, Austria is climatically allocated to the European moderate climate zone. This zone is shaped by oceanic as well as continental and pannonic influences respectively. In the east of Austria this interaction is defined by cold winters and hot summers with moderate precipitation all-season. The months with the highest precipitation rates are June, July and August, whereas in the winter months between November and March/April there is only low precipitation (figure 5-3). The Rabenwald with its 15-year-precipitation-average (1992-2008) of 1082 mm/a (figure 5-4) shows considerably higher values than valleys in the vicinity. This instance is caused by the altitude of the open pit mining between ca. 900-1100 m, as an average increase of precipitation of 27.8 mm/a occurs with an increase of altitude of 100 m (POLTNIG and REINSDORFF, 2000). Figure 5-5 shows the temperature distribution at Rabenwald from 1998 to 2008.

The migration of the acidic drainage as well as the potential dimension of contamination of the environment are codetermined by the runoff of open pit mine waters and dump seepage waters and the sizes of the drainage areas of the particular recipients. The morphology of the site is necessarily changed because of surface extraction of a resource. Additionally, interferences by means of activities in the context of dump formation occur regarding the discharge characteristics of seepage water flowing through the dump. These factors have in further succession a direct influence on the hydrologic and hydrogeologic characteristics of the open pit mining and its surrounding environments.

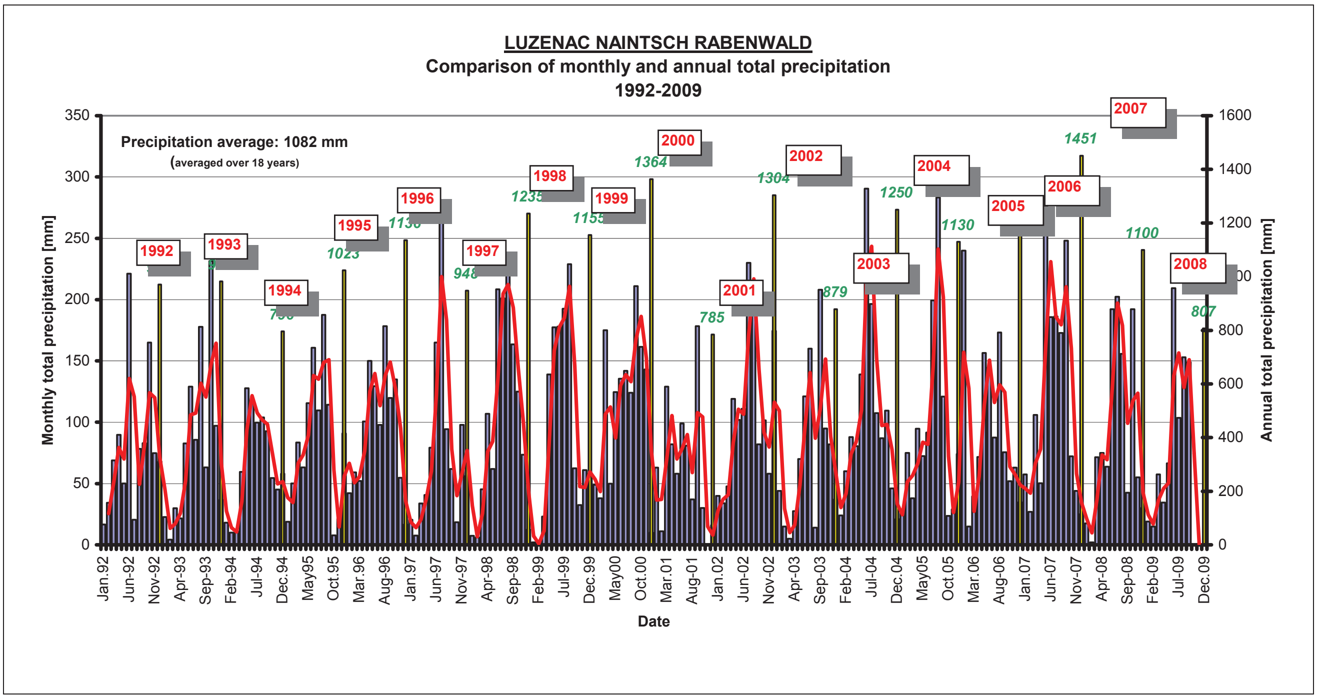

Figure 5-3: Comparison of monthly and annual total precipitation 1992-2008: monthly total precipitation (blue), total annual precipitation (yellow), moving average of two periods of the monthly total precipitation (red) (data from LUZENAC NAINTSCH Rabenwald).

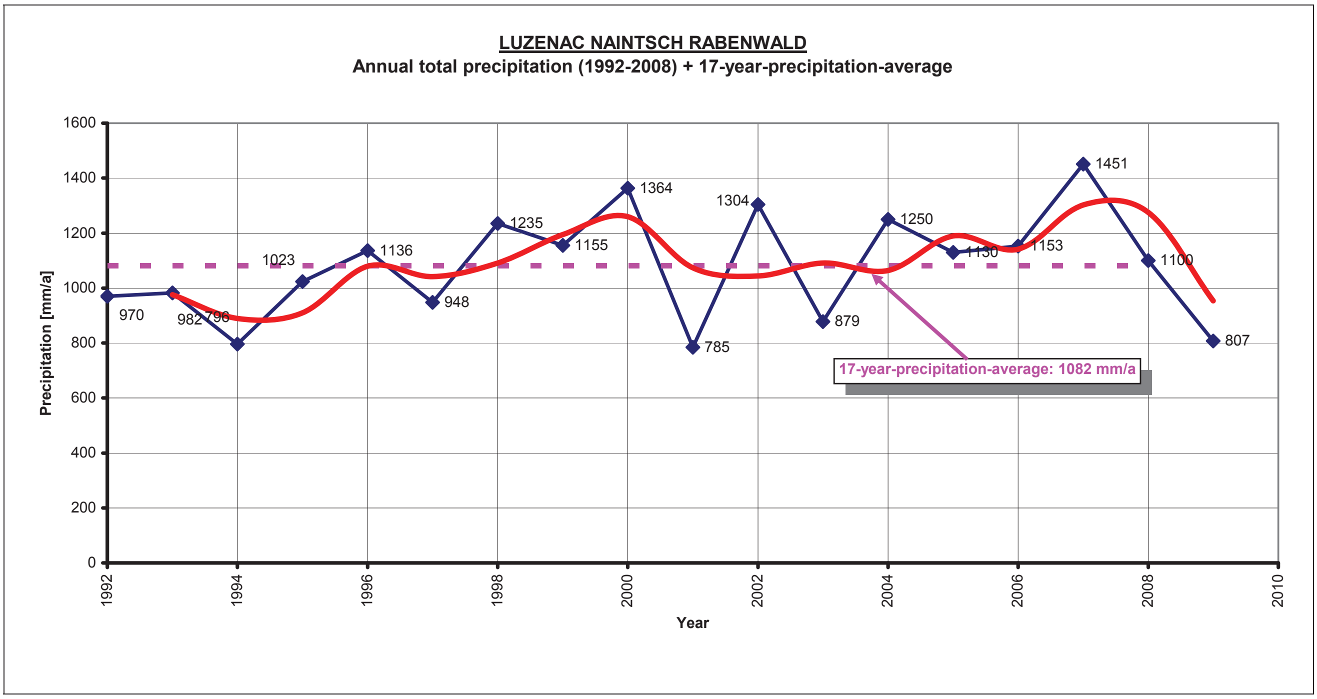

Figure 5-4: Precipitation pattern of the Rabenwald: total annual precipitation (blue), moving average of precipitation of two periods (red), 17-year-precipitation-average (purple) (data from LUZENAC NAINTSCH Rabenwald).

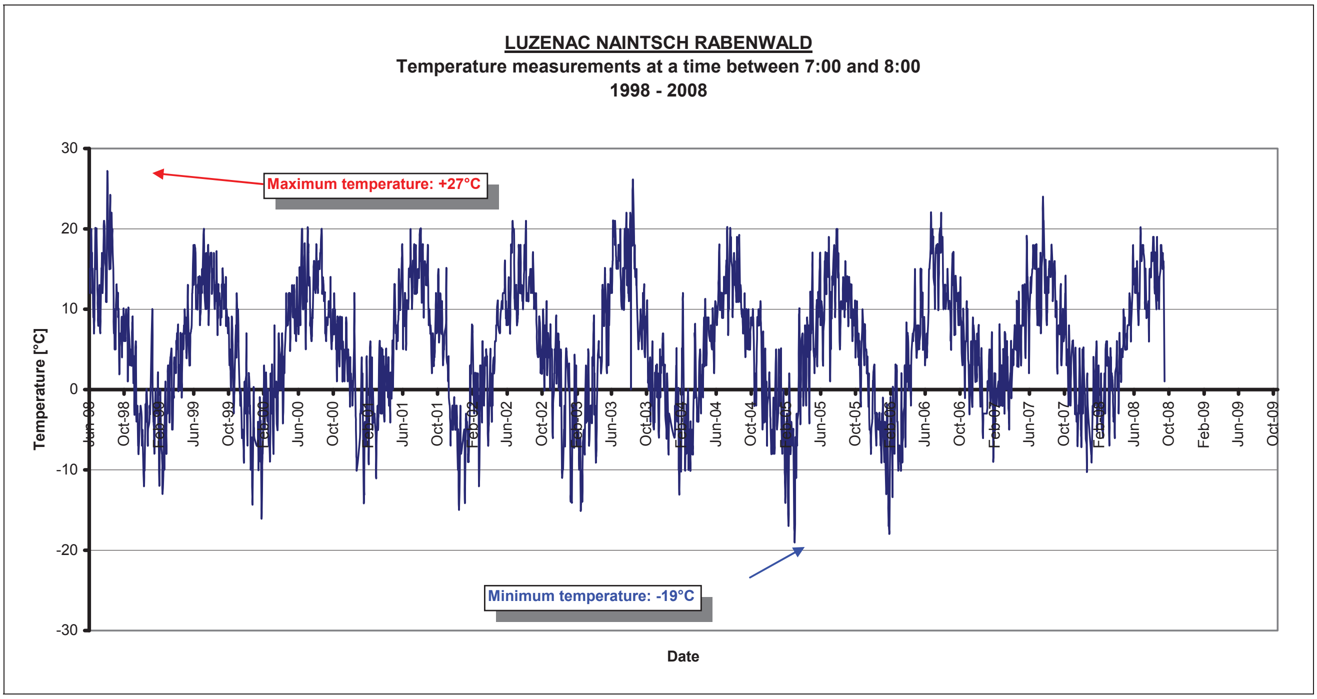

Figure 5-5: Temperature distribution of the Rabenwald from 1998 to 2008 with measured maximum temperature and minimum temperature (data from LUZENAC NAINTSCH Rabenwald).

Rabenwald open pit mining strikes four drainage areas which are briefly described in the following (according to POLTNIG and REINSDORFF, 1998):

The largest part of Rabenwald surface mining is drained by the catchment area of the Floingbach. The mining activities underground and on the surface in the past already led to changes compared to the natural orographic drainage area. In particular the initiation of the retention basin “Wasserspiegel Brunnen” resulted in a deficit concerning the runoff at the measuring points of the Floingbach within the scope of the hydrologic balance equation (see below).

The Lambach drainage area is located east to northeast of the open pit mining and shows no surface runoff in the vicinity of the deposit. Below an altitude of about 900 m two drain ways appear which join and form the Lambach at a sea level of about 750 m.

The drainage area of the Schmidbach is located adjacent to the south to that of the Lambach. Surface runoff only can be observed in the lower part of the drainage area. According to SUETTE (2004) there are several used springs (Sternmichl I-III, Lacknerquelle, Kothgasserquelle) appearing within this area.

The western part of the mining is drained by the catchment area of the Dunstbach.

To permit an assumption of the magnitude of changes conditionally to the hydrologic and hydrogeologic conditions, a hydrologic measuring point network was installed by POLTNIG and REINSDORFF (2000), in the scope of a hydrogeologic estimation of the influence of future mining activities on the drainage areas and recipients of the Rabenwald surface mining. From the obtained data about precipitation and runoff characteristics of the most important recipients (Dunstbach, Floingbach, Schmidbach, Lambach), hydrologic balance equations for the drainage areas of the Floingbach and the Dunstbach could be realized (see below). The sizes of the catchment areas of the individual recipients were determined by POLTNIG and REINSDORFF (1998) by means of fixed measuring points for the year 1998 as well as estimated ones by means of a future mining plan for the year 2006. The values received thereby are shown in table 5-1.

Drainage area	Size 1998 [km^2]	Size 2006 [km^2]
Floingbach	0,56	0,54
Schmidbach	0,73	0,72
Dunstbach	0,98	0,99
Lambach	1,77	1,79

Table 5-1: Sizes of the drainage areas of the most important recipients of the Rabenwald (according to POLTNIG and REINSDORFF, 1998).

As there were no existing runoff measurement results for the drainage areas of the Schmidbach and the Lambach, only for those of the Floingbach and the Dunstbach a realization of the hydrologic balance equation was possible. The results of these equations are displayed in table 5-2 (POLTNIG and REINSDORFF, 2000). Furthermore the results of the hydrologic equation for the Höhenhanselbach of FANK et al. (1999) are shown too. The drainage area of the Höhenhanselbach features similar topographic parameters as the catchment areas of the Rabenwald and was therefore used for the correlation of the results.

Drainage area	MQ	MoMNQ$_T$	Mq	A$_o$	MoMNq$_T$ = A$_u$	P	E$_{Ta}$ (+ deficit)*
	[l/s]	[l/s]	[mm/a]	[mm/a]	[mm/a]	[mm/a]	[mm/a]
Floingbach	5,78	4,27	325,72	85,09	240,63	1034,69	708,97
Dunstbach	14,52	11,08	467,57	110,77	356,79	1043,89	576,32
Höhenhanselbach	6,53		527,61	117,48	409,8	1064	537,49

Table 5-2: Results of the hydrologic equations of the drainage areas of the Floingbach and the Dunstbach for the years 1995-1998 (according to POLTNIG and REINSDORFF, 2000) as well as the results of the hydrologic equation of the Höhenhanselbach according to FANK et al. (1999).

MQ ... Mean total runoff during the observation period.
MoMNQ$_T$ ... Monthly mean “Niederabfluss”.
Mq ... Standardised mean total runoff.
A$_o$ ... Surface runoff.
MoMNq$_T$ ... Standardised monthly mean “Niederabfluss”.
A$_u$ ... Ground water recharge rate.
P ... Precipitation.
E$_{Ta}$ ... Evaporation.
* ... Just at the Floingbach (see text).

POLTNIG and REINSDORFF (2000) used the drainage area of the Höhenhanselbach, which was investigated by FANK et al. (1999), for the correlation of their results as it shows similar parameters as the catchment areas of the Rabenwald. The hydrologic balance equation of the Dunstbach fits well with that of the Höhenhanselbach; the results of the Floingbach on the contrary differ clearly from the results of the other two drainage areas. According to POLTNIG and REINSDORFF (2000) the hydrologic characteristics of the drainage area of the Floingbach are mainly changed to a large extent by the water drainage at the mining area. Nearly the whole open pit mining is drained by the closed systems of the Ernst August Stollen and the Lilly Stollen. In addition to this, surface water coming in because of precipitation events is also brought into these systems by using pumps. According to POLTNIG and REINSDORFF (2000) the different discharge of those two systems is now the reason for the deficit in the hydrologic equation of the Floingbach. The Ernst August Stollen is drained into the retention basin “Wasserspiegel Brunnen” through a 600 m channel. It is possible that loss of water already occurs on the way because of seepage. The whole incoming water seeps away at the retention basin; even though there is heavy precipitation, no overflow into the Floingbach can be observed according to POLTNIG and REINSDORFF (2000). In contrast to that, the entire runoff of the Lilly Stollen discharges into the Floingbach and is therefore recognized in the runoff measurements of the Floingbach. The discharge of the Ernst August Stollen is missing in the hydrologic balance equation of the Floingbach and is thus responsible for the deficit appearing there. According to POLTNIG and REINSDORFF (2000) the occurrence of small emergences (e.g.: Rohrquelle Feldhofer) down the slope of Wasserspiegel Brunnen can be seen in direct relation to the initiation of this retention basin. It is assumed that only a minor part of the water seeping away at Wasserspiegel Brunnen gets into the Floingbach, whereas the major part flows off subterraneously and is not recognized at the measuring points.

Because of the existing geological conditions the Rabenwald features scores of springs of low yield (mostly <1 l/s) which vary strongly with the meteorological circumstances (SUETTE, 2004). This is seen as a direct indication for only low water storage capacity of the bedrock. The trend of the measured spring flows at Rabenwald follows the yearly precipitation pattern. According to POLTNIG and

REINSDORFF (2000) a significant increase of discharge can be recorded from April/May and a continuous decrease of discharge from September/October.

The future mining activities at Rabenwald will change the morphology of the crest areas and in the following the hydrologic conditions and the sizes of the drainage areas as well. The waste rock material which is accumulated with the mining of talc is back tipped. In the process it is considered that the originally crest morphology is being reconstructed if possible. According to POLTNIG and REINSDORFF (2000) infiltrating rainfall water will find good permeability in the dump but hit a low permeable rock formation in the footwall of the mined talc. The better part of the seepage water will consequently follow this depression axis of the mined talc towards south (recipient Floingbach) and hence won’t be available for the other recipients.

According to POLTNIG and REINSDORFF (2000) the sizes of the drainage areas of the Dunstbach and the Lambach only change marginally during planned future mining activities. As the workings are progressed in western direction, the size of the drainage area of the Floingbach will increase by an amount of >30% at the expense of the Schmidbach drainage area, which will be scaled down by about 10% until 2030. The expected sizes of the drainage areas in the year 2030 and the changes per centum to the values of 1998 are demonstrated in table 5-3 (according to POLTNIG and REINSDORFF, 2000).

Drainage area	Size [km^2]	Changing to 1998 [%]
Floingbach	0,73	+30,3
Schmidbach	0,66	-10
Dunstbach	0,97	-1,1
Lambach	1,76	-0,1

Table 5-3: Expected sizes of the drainage areas of the most important recipients of the Rabenwald in the year 2030 and the changes in per cent compared to the values of 1998 (according to POLTNIG and REINSDORFF, 1998).

The sizes of the drainage areas of course have a direct influence on the amount of discharging water. According to SUETTE (2004) the runoff of the Floingbach will increase by about 23%; at the Schmidbach on the contrary there will be a reduction of the runoff of about 11%. Thus the Floingbach will exert an increasing influence on the runoff conditions of the deposit, especially those of the dump waters.

According to SUETTE (2004) the changes in water supply will lead to quantitative detraction of springs in the Schmidbach drainage area and to qualitative and quantitative impacts in the area of the Floingbach. For reasons of perpetuating testimony and of monitoring the water quality, the seepage of several springs at Rabenwald is conducted an examination of its mining specific (nitrate, nitrite, ammonium, hydrocarbons, cadmium, lead) and its general parameters (colour, suspension, temperature, electrical conductance, pH value, chloride, sulphate). The results of these measurements are displayed in the chapter RESULTS and in the APPENDIX and serve as a completion of the yielded laboratory data in the chapter DISCUSSION.

6. ANALYTICAL METHODS

6.1 Sampling

For the measurements relevant to this diploma thesis, samples from Rabenwald open pit mining were recovered twice. The first sampling was carried out in January 2007, the second in June 2007. The rock samples were not drawn from bedrock, but from the dump “Mittelbau”, the Wiedenhofer area and the talc stockpile.

In the course of the first sampling, one waste rock sample as well as four to five representative pieces of leucophyllite (LP 1-5), marble (MA 1-5), granite gneiss (GG 1-4), hornblende gneiss (HG 1-5), paragneiss (PG 1-4), calcsilicate (CS 1-3) and actinolite gneiss (AG 1-6) were taken from the dump “Mittelbau”. The six talc samples (respectively three light and three dark samples; TA 1-6) were taken from the talc stockpile located next to the office building of the open pit mining; the four actinolite gneiss samples especially rich in sulphide (AG 3-6) are from the Wiedenhofer area. The sampling of the waste rock sample was achieved by ‘blind’ collection of differently-sized rock pieces of the dump “Mittelbau”.

The second sampling served as a complement of the first one. Here, further six pieces of calcsilicate (CS 4-9), five actinolite gneiss samples (AG 7-11) and representative rocks of all rock types occurring at the open pit were drawn, in order to create ‘artificial’ bulk samples.

6.2 Preparation and treatment of the samples

First all rock samples were cleaned in the laboratory. Then the representative samples of the different rock types were divided into three pieces with a saw. One piece was stored as a retain sample. The other both pieces served as a model for a thin section, or respectively to further processing. The samples selected for further processing were broken with a jaw crusher with the smallest jaw setting.

After purification, the rocks of the waste rock sample were allocated to the individual formations, and then weighed separately in order to gain the parts of the individual fractions in per cent (see table 6-1).

Rock type	Single pieces	Mass [g]	Share [%]
Actinolite gneiss/calcsilicate	22	11186,6	46,02
Paragneiss	26	10970	45,13
Hornblende gneiss	1	757	3,11
Leucophyllit	2	559,9	2,30
Granite gneiss	2	503,7	2,07
Marble	1	331,6	1,36
Total	**54**	**24308,8**	**100**

Table 6-1: Number of single pieces, mass und per cent part of the different rock types of the waste rock sample.

Subsequently, the waste rock sample was broken with a jaw crusher with the widest jaw setting. After homogenisation of the broken material, 500 g were separated for the realization of the Leco analysis and then broken once again with a jaw crusher with the smallest jaw setting. The rest of the sample was used for the first leaching test (LT 1).

The composition of the waste rock sample did not correspond to the actual ‘approximate’ rock composition of the whole open pit mining. For that reason, three ‘artificial’ bulk samples were compiled with the aid of a chart, which is based on drillhole data representing the average rock composition of the entire surface mining. Those rocks whose part in the bulk composition was less than 2% were not included here. The table 6-2 shows the data based on the drillhole data as well as the composition of the samples ABS 1-3 (‘artificial’ bulk samples 1-3).

The three bulk samples were broken with a jaw crusher with smallest jaw setting, and then they were further processed for the second leaching test (LT 2) and the Leco analysis.

Drillhole Database		ABS 1		ABS 2		ABS 3	
Rock type	Share [%]	Mass [g]	Share [%]	Mass [g]	Share [%]	Mass [g]	Share [%]
Leucophyllite	11,1	1001,9	10,2	1000,3	10,2	999,2	10,2
Talc	11,1	998,6	10,2	999,2	10,2	1000,6	10,2
Hornblende gn.	19,7	1998,7	20,4	2000,6	20,5	2000,9	20,5
Actinolite gneiss/ calcsilicate	7,8	1044,0	10,7	1019,2	10,4	1019,8	10,4
Paragneiss	31,8	2992,9	30,6	3007,4	30,7	2999,4	30,8
Talc/gneiss Mix	13,7	1500,5	15,3	1501,1	15,4	1501,2	5,3
Granite gneiss	2,1	252,2	2,6	251,7	2,6	251,6	2,6
Sum	**97,3**	**9788,8**	**100**	**9772,7**	**100**	**9779,5**	**100**

Table 6-2: Drillhole database Rabenwald and composition of the bulk samples (ABS 1-3).

6.3 Thin sections, polished sections

Total fourteen thin sections were produced of all different rock types: respectively two of leucophyllite (LP 1, LP 4), marble (MA 1, MA 2), granite gneiss (GG 1, GG 2), hornblende gneiss (HG 2, HG 4), calcsilicate (CS 1, CS 3) and one of paragneiss (PG 1). Of actinolite gneiss (AG 1, AG 3, AG 6) three thin sections were provided.
For the four polished sections the sulphide-rich samples AG 3-6 were selected.

6.4 Leco analysis

By means of the Leco analysis, the level of organically bound carbon as well as the content of sulphur in a sample can be determined. Since this thesis centres on sulphide minerals and acid rock drainage, only the sulphur content of the rock samples was measured.

The percentage of sulphur in a sample is determined by burning the rock material at 1500°C under oxygen atmosphere. When the sample is being burned, the sulphur reacts to SO_2. By means of an infrared detector the concentration of SO_2 is measured, from which the sulphur content of the sample can be deduced. The determination of the sulphur content was achieved by means of a LECO 300 CSTM analyzer.

For the analyses, approximately 30 g of the different samples were ground for 10 minutes each with an agate mortar in order to produce grain sizes apt for analysis. Subsequently, respectively 100 mg of the rock powder was weighed into crucibles and mixed with alcohol. After one night in the drying oven, wolfram and iron was added to the crucibles, and finally the measurements were carried out.

For the first trial of the Leco analyses, respectively two double-samples of the rocks were analyzed. The samples were divided into two groups according to expected sulphur content. For granite gneiss (4 samples), leucophyllite (5 samples), talc (6 samples) and marble (5 samples) low sulphur contents were assumed. At the measurement a standard of 0.0044% S was applied in order to guarantee exactest results. For the analysis of paragneiss (4 samples), hornblende gneiss (5 samples), calcsilicate (3 samples), and actinolite gneiss (2 samples), of which a higher sulphur content was expected, a standard of 0.360% S was used. As the examined paragneisses and hornblende gneisses showed only low contents of sulphur, these rocks were repeatedly measured with the lower standard. Furthermore, the sulphur content of three double samples out of the waste rock sample, which had been gathered by ‘blind’ collection of the rocks, was determined with the higher standard (0.360% S).
The data of these measurements are partly shown in chapter RESULTS as well as in the APPENDIX.

In the context of the first Leco-measurements, it was particularly the calcsilicates and the actinolite gneisses which showed increased contents of sulphur. For that reason, measurements with six further calcsilicate samples (CS 4-9) and nine further actinolite gneisses (AG 3-11) were effected, with the aim to be able to make more accurate and precise statements about the average as well as the variation of the sulphur contents of these rocks. Furthermore the three bulk samples (ABS 1-3) compiled from drillhole database were analysed. The Leco analyses of this run were performed with the higher sulphur standard (0.360% S) and with 3 measurements per samples. The measured data are shown in the APPENDIX.

6.5 Leaching tests

6.5.1 General introduction

The purpose of leaching tests is to enable predictions about the acid generation potential in a deposit. These tests should provide two kinds of data (cf. HUTCHINSON and ELLISON):

- If acid generation occurs in certain deposit material, and if so then to which extent.
- Information about quantity and quality of the acidic drainage which migrates from the deposit under controlled or uncontrolled conditions.

There are several methods to evaluate the acid generation potential of a deposit. However, in practice only geochemical kinetic tests and - secondarily - geochemical static tests are relevant. These two methods of testing are similar in design and execution. The striking difference between the tests consists in testing duration. Static tests are short-term lab experiments which compare acid generation with neutralisation capacity of samples. Kinetic tests are long-term, produce acid leachate and serve to predict the rate of acid generation of the examined samples. Both methods thus provide a reliable way to characterize deposit material regarding its capacity to produce acid. Furthermore, with these methods adequate inspection measures to prevent or reduce acid generation can be defined. Due to their longer duration, kinetic tests provide more reliable data on the convertibility of acid generation in the course of time, and consequently better interpretation possibilities. For that reason, kinetic test arrangements were chosen for the experiments carried out within the scope of this thesis.

In total three leaching tests were performed with rocks of different composition and grain size of the Rabenwald open pit mining. One further experiment with ore rocks from Corinthia, Australia and Poland served as a complementation of the results and to retain the possibilities to compare. Above all the purpose of the leaching tests carried out for this thesis was the determination of the development of the pH-value with time. During the experiments the pH values of the basis fertiliser solutions were measured in intervals by means of a pH meter (Lesto 230). In addition, samples of the different solutions were taken and were examined regarding ions with an ion chromatograph (see 6.6). The different leaching tests varied in duration. An experiment was finished when no significant change was to be observed at a pH measurement in comparison to the former measurements. The samples and test arrangements are described in the following sections.

6.5.2 Leaching test of the waste rock sample (LT 1)

For the first leaching test (LT 1) rocks of the waste rock sample were used which had been gathered by 'blind' collection at the Mittelbau dump. The rocks, broken by means of a jaw crusher, were first sieved and consequently divided into the grain size categories <5 mm, 5-20 mm, >20 mm. This differentiation of the fractions should enable predictions of the interrelation between grain size and grain surface in regard to acid generation. Then respectively two times 1500 g (<5 mm, 5-20 mm) of the different fractions as well as, due to the bigger grains, also respectively two times 3000 g (>20 mm) were weighed in. These sample amounts were put into six buckets with lids and each filled up with 3 l basis fertiliser solution. For the >20 mm fraction, the same amount of basis fertiliser solution was used despite the bigger sample quantity, because in this case, compared to the other samples, a lower rate of acid generation was assumed. In order to characterize the pH dependence of the acid generation, different basis fertiliser solutions were chosen. For every grain size category, the experiment was performed both with distilled water (pH of 7) as well as with solution acidified with HCl (pH of 3.6). The closable buckets should prevent evaporation of the basis fertiliser solutions. Additionally, the original levels of the solutions were marked in the buckets. If necessary, distilled water was added up to this level during the experiment. The detailed weights samples of the different fractions as well as the type and quantity of the used basis fertiliser solutions are shown in table 6-3.

Sample name	Grain size [mm]	Weighted sample [g]	Basis fertiliser solution
1	<5	1500,1	3 l dist. H_2O
2	5-20	1500,3	3 l dist. H_2O
3	>20	3002,7	3 l dist. H_2O
4	<5	1500,1	3 l dist. H_2O + HCl
5	5-20	1499,9	3 l dist. H_2O + HCl
6	>20	3001,4	3 l dist. H_2O + HCl

Table 6-3: Grain size fractions, weighted samples and type and quantity of used basis fertiliser solutions for the first leaching test (LT 1).

The first leaching test was started on 27.02.2007 and lasted for a total 3840 hrs or 160 days. The pH value was measured a total nine times; samples of the solutions were taken ten times. The measurements results are shown in the chapter RESULTS and in the APPENDIX.

6.5.3 Leaching test of the bulk samples (LT 2)

The composition of the waste rock sample cannot be transformed onto the whole open pit mining. Consequently, the results of the first leaching test can only restrictedly be used for interpretations concerning acid generation. For this reason, a second leaching test (LT 2) was started with three bulk samples (BS 1-3) compiled with use of drillhole data. The composition of the bulk samples BS 1 and BS 2 as well as the amount of distilled water added for the tests is shown in table 6-4. The tests were carried out in closable buckets in order to prevent the evaporation of the basis fertiliser solutions.

The two samples in the chart below only differ in regard to their quantity of marble. In contrast to BS 1, BS 2 contains 20 g of marble. It is common knowledge that carbonates act as an acid buffer. In order to examine the effects of this puffer capacity on the pH value, the same test was run with the same conditions, except that the marble quantity in the samples was altered. The calcsilicate used for the compilation of the bulk samples was taken from the sample CS 7 with a sulphur content of 0.86% S. This value approximately corresponds to the average of all measured calcsilicate and actinolite gneiss samples; it can therefore be considered representative for this sulphide-bearing rocks. The grain sizes of both overall samples are <3 cm.

Rock type	**BS 1**	**BS 2**
	Initial weight [g]	
Leucophyllite	111	111
Talc	111	111
Hornblende gneiss	197	197
Calcsilicate (CS 7)	78	78
Paragneiss	318	318
Talc/gneiss-Mix	137	137
Granite gneiss	21	21
Marble	0	20
Sum	**973**	**993**
Distilled water [ml]	**1950**	**2000**

Table 6-4: Composition of the bulk samples (BS 1, BS 2) and quantity of used basis fertiliser solutions for the second leaching test (LT 2).

BS 3 then should simulate a worst case scenario for the Rabenwald open pit mining. The ABS 1_3 sample was chosen as the provider of the material for this test, because this sample showed with 0.120% the highest sulphur values according to Leco analysis. In addition, rock powder was used to achieve a larger exposed surface and thus optimal reaction potential and acid generation. For the test, 150 g of rock powder was mixed with 300 ml of distilled water.

The second leaching test started on 30.10.2007 and lasted for a total 1870 hrs or 78 days. In the process, the pH values were measured ten times; six times samples of the basis fertiliser solutions were drawn for analyses with the ion chromatograph. The results of the measurements are shown in the chapter RESULTS as well as in the APPENDIX.

6.5.4 Leaching test actinolite gneiss (LT 3)

For all rock types occurring at Rabenwald open pit mining, Leco analyses were carried out to determine the content of sulphur. Of all examined rock samples, actinolite gneisses showed the significantly highest quantities. For that reason, these gneisses should possess the highest acid generation potential of all rocks in the Rabenwald surface mining. The objective of the third leaching test was to determine this potential under the most ideal conditions for acid generation which can possibly occur in Rabenwald open pit.

For the test the AG 11 sample was chosen, because it showed with a sulphur content of 2.64% the highest amongst all examined rocks. The experiment was started on 23.07.2007 with 500.14 g of rock powder and one litre of distilled water in a covered beaker. The experiment was ended after 4386 hrs or 183 days and eleven pH measurements. The results of this leaching test are shown in the chapter RESULTS as well as in the APPENDIX.

6.5.5 Leaching test acid rock drainage (LT 4)

The rocks of the Rabenwald surface mining approximately only show a small percentage of sulphide minerals. In order to be able to compare the results of the experiments described above, a forth leaching test was carried out with rocks from such mineral deposits for which acid generation potential could be assumed. Furthermore, this experiment should show the ‘standard’ development of the pH-value of rocks with distinct potential for acid generation. The selected samples are taken from Waldenstein in Corinthia, Pine Point in Australia and from Poland. The sample of Waldenstein is a hematite with high amounts of pyrite. Pine Point is a Mississippi valley-type Pb/Zn-deposit. The sample of Poland is a copper schist consisting of about 10% chalcopyrite and pyrite.

The test was started on 05.06.2007 and was ended after 3691 hrs or 154 days. During this period, the pH value was measured fourteen times. The grain sizes of the samples were <3 cm; as basis fertiliser solution, respectively 300 ml of distilled water was added. The exact weighted samples can be seen in table 6-5.
The results of this leaching test are shown in the chapter RESULTS as well as in the APPENDIX.

Sample name	Initial weight [g]
Waldenstein	150
Pine Point	150,1
Cu-schist	150,2

Table 6-5: Initial weights of the samples used for LT 4.

6.6 Ion chromatography

Ion chromatography serves to measure the cations Li, Na, K, Mg and Ca as well as the anions F, Cl, Br, NO_3, SO_4 and PO_4 of fluid samples. For the analysis, only approximately 20 µl of the sample solution is necessary. For the measurement of the samples a Dionex DX-120 ion chromatograph was used that performs isocratic ion analyses applications using conductivity detection. This method is based on the attitude of ions in solution to conduct electrical current when voltage is applied between electrodes contacting the solution. Furthermore the magnitude of this current is nearly proportional to the concentration of dissolved ions in the sample. The conductivity of the sample is measured by applying an alternating voltage between two electrodes in a conductivity cell so that the negatively charged anions migrate towards the positive electrode and positively charged cations migrate towards the negative electrode. The solution resistance is then calculated from Ohm’s law:

$$R = \frac{E}{I}[\Omega]$$

(R ... resistance, E ... voltage, I ... amperage).

Respectively the conductance G in siemens [S] is therefore:

$$G = \frac{1}{R} = \frac{I}{E}[S]$$

(G ... conductance, R ... resistance, I ... amperage, E ... voltage).

This measured conductance has to be corrected by the conductivity cell constant K:

$$\kappa = K \times G \left[\frac{S}{cm} \right]$$

(κ ... conductivity, K ... conductivity cell constant, G ... conductance).

According to Kohlrausch's law of independent migration, conductivity is directly proportional to concentration (DIONEX CORPORATION, 1997).

The detection limits of the Dionex DX-120 as well as the analysed cations and anions are shown in table 6-6.

Parameter	Detection limit [ppb]	Parameter	Detection limit [ppb]
Li	0,1	F	5
Na	5	Cl	100
K	5	Br	5
Mg	20	NO_3	10
Ca	20	SO_4	20

Table 6-6: Analysed parameters and detection limits.

The contained chromatograms were automatically, electronically integrated. With some samples, additionally, manual integration was required. Figure 6-1 shows, as examplification, a chromatogram of the measured anions of an analysed waste rock sample.

The ion chromatography was used to determine the changes and contents in cations and anions of the solutions used in the leaching tests. There was a particular focus on the SO_4 values of the samples. The samples and solutions used in the leaching tests are described in chapter 6.5.

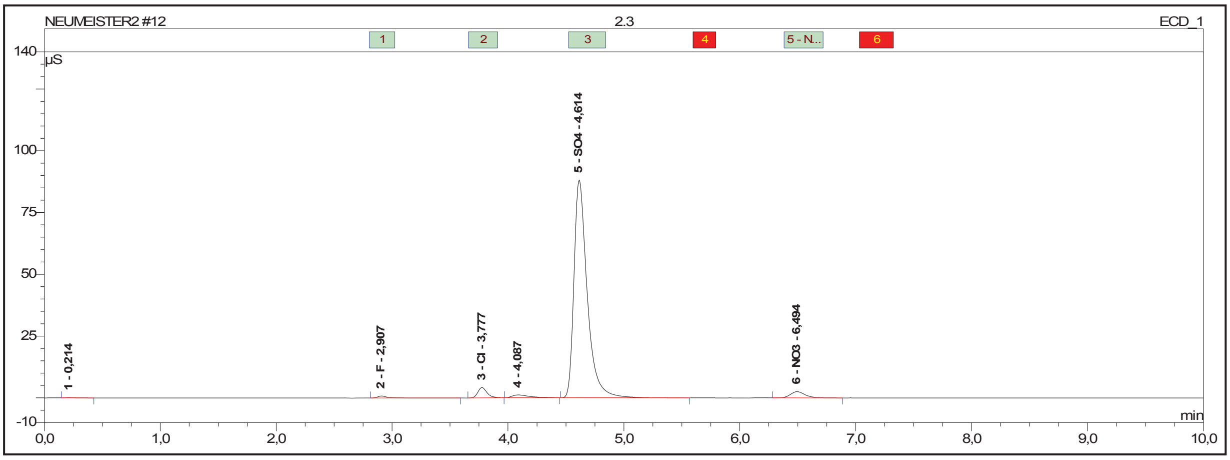

Figure 6-1: Anion chromatogram of an analysed solution of LT 1.

Before the samples were drawn, the containers for the samples of the leaching tests were shaken in order to homogenize the solutions. The drawing of the sample itself was performed with an injection with a fitted filter. This should ensure that only pure samples of the solution, without particles, would be obtained. The sample quantity taken from the containers was subsequently levelled with distilled water. In total, samples of the solution of the first leaching test were taken ten times (with the waste rock sample taken from the Mittelbau dump, see 6.5.2). Six samples of the solutions of the second leaching test with artificially compiled bulk samples were taken (see 6.5.3). The dates the samples were taken and the results are shown in the chapter RESULTS and in the APPENDIX.

7. RESULTS

7.1 Petrography

Description of the main rocks of the Rabenwald talc deposit.

7.1.1 Transmitted light microscopy

7.1.1.1 Leucophyllite/"Kornstein" (LP 1, LP 4)

When examined with the naked eye, the thin sections of the leucophyllite show distinct orientation. However, the sections slightly differ from one another in their mineral composition.
Quartz, muscovite and chlorite constitute the major minerals of the rock (figure 7-1). They appear in alternating, aligned layers and thus define the lepidoblastic structure of the samples. Coarser grains constitute nearly mono-mineral layers or lenses. In contrast to that, the intermediate layers are composed of very fine-grained mineral grains.

Turmaline, apatite, rutile and zircon appear as subordinated components in LP 1. Apatite is characterized by its positive relief as well as by its round, hypidiomorphic grain shape. The pleochroitic turmaline could be identified as a further hypidiomorphic component of the sample (figure 7-2).

In contrast to that, tremolite, apatite, turmaline, titanite, zircon and opaque phases constitute further mineral phases in LP 4.
The opaque phases are oriented in fine veins following the cleavage. The turmaline shows weak pleochroism, brown areas in the centre of the grains and color zoning within the interference colors. The brown areas mark the Fe-rich variety of the turmaline, the color zoning can be ascribed to dravite (Mg variety) accretion seams. Tremolite (Fe-poor actinolite) appears in xenomorphic, colorless aggregates, also transverse to the cleavage.

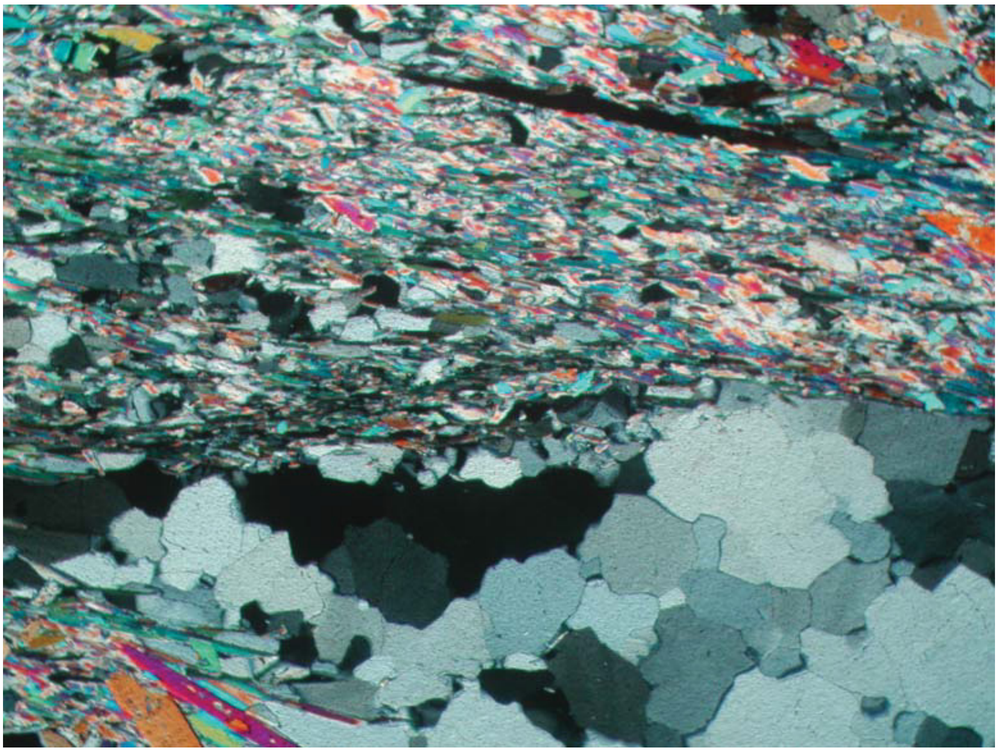

Figure 7-1: Leucophyllite (Kornstein)
CPLx4 (Picture size 75%).

Figure 7-2: Turmaline in sample LP 1
PPLx25 (Picture size 50%).

7.1.1.2 Marble (MA 1, MA 2)

The marble of the Rabenwald deposit shows a typical granoblastic structure and is nearly mono-minerally composed of calcite (figure 7-3). Muscovite and chlorite figure as accessory components.

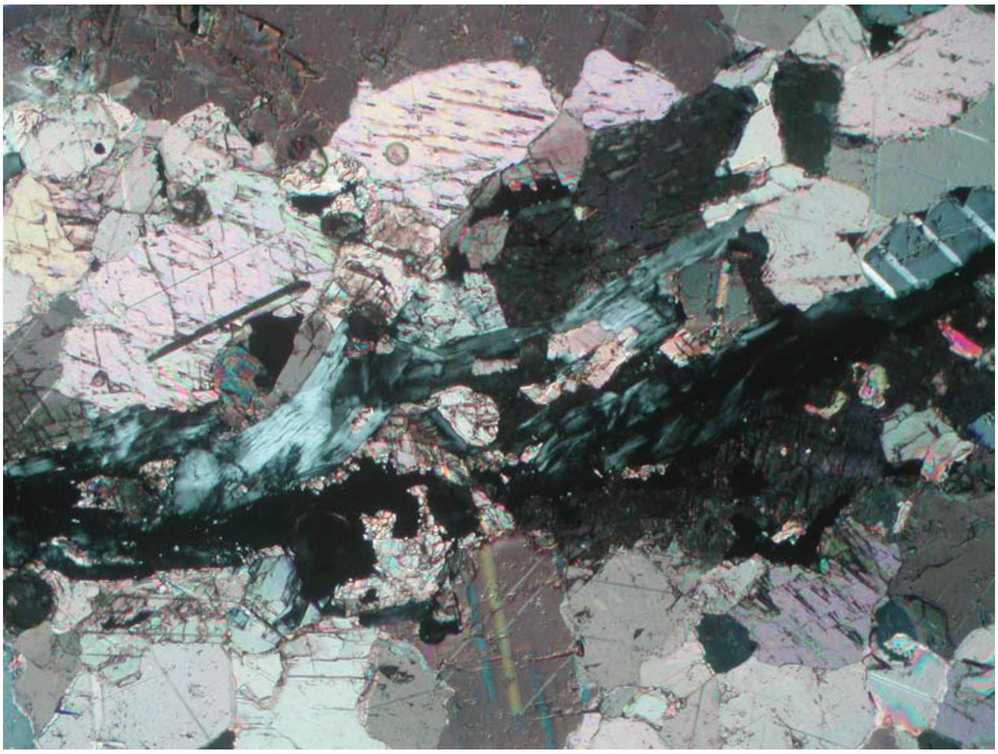

Figure 7-3: Calcite and chlorite in the thin section MA 1
CPLx10 (Picture size 60%).

7.1.1.3 Granite gneiss (GG 1, GG 2)

Already without examination under microscope, the two thin sections of the granite gneiss show significant differences. The GG 1 sample only shows minor bedding, GG 2 reveals veins and an orientation of the mineral grains.

GG 1 consists of quartz, muscovite, clinozoisite, potassium feldspar, garnet, rutile, apatite, zircon and opaque phases. Striking is the formation of the clinozoisite. It builds dark, unaligned ‘needles’ with higher relief, particularly in the feldspar (figure 7-4). The muscovite partly shows slight alignment and defines, together with mono-mineral quartz areas, the orientation of the rock. The opaque phases can be identified as finely spread, small grains.

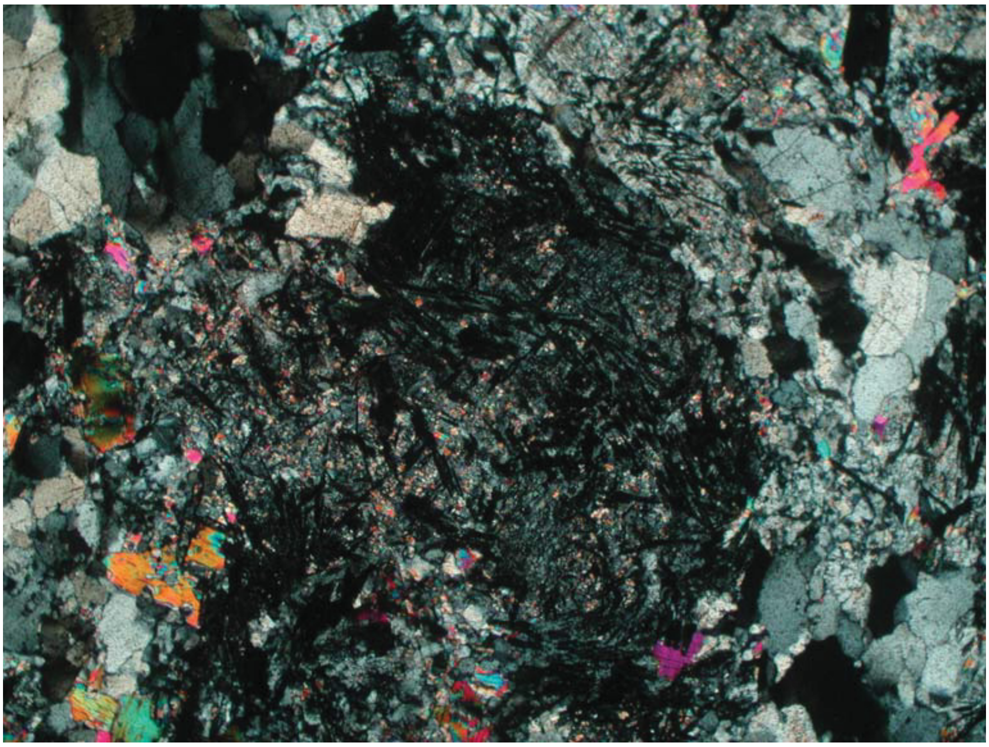

Figure 7-4: ‘Needles’ of clinozoisite in feldspar in the sample GG 1
CPLx4 (Picture size 75%).

The GG 2 sample consists of quartz, muscovite, garnet, potassium feldspar, plagioclase, clinozoisite, biotite, apatite and zircon (figure 7-5). The orientation of the rock is defined by the aligned muscovite, which partly occurs in veins.

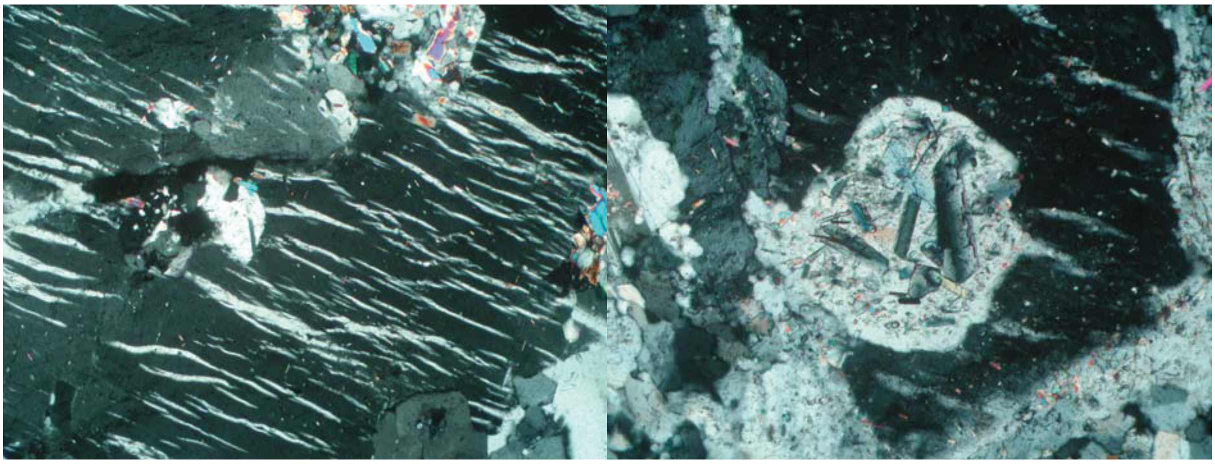

Figure 7-5: Sample GG 2.
Left: Perthitic decomposition of potassium feldspar; CPLx10 (Picture size 40%).
Right: Grains of clinozoisite in feldspar; CPLx10 (Picture size 40%).

The clinozoisite appears completely differently compared to sample GG 1; it is only subsidiarily contained in shape of longish grains. The beginning conversion of the clinozoisite to epidote is already suggested by the partly more colorful interference colors of the mineral. The potassium feldspar sporadically shows a typical microcline lattice as well as perthitic decomposition (veined perthite) in polarized light (figure 7-5). A further difference from GG 1 appears in the high content of garnet grains.

7.1.1.4 Paragneiss (PG 1)

The sample of the paragneiss is fine-grained and shows distinct orientation defined by quartz and mica (figure 7.6). Quartz is the main component of the section; muscovite and biotite appear as mica representatives. Further components are actinolite, garnet, chlorite, plagioclase, rutile, zircon and opaque phases. The bedding of the rock is defined by mica, actinolite and opaque phases, whose mineral grains are oriented.

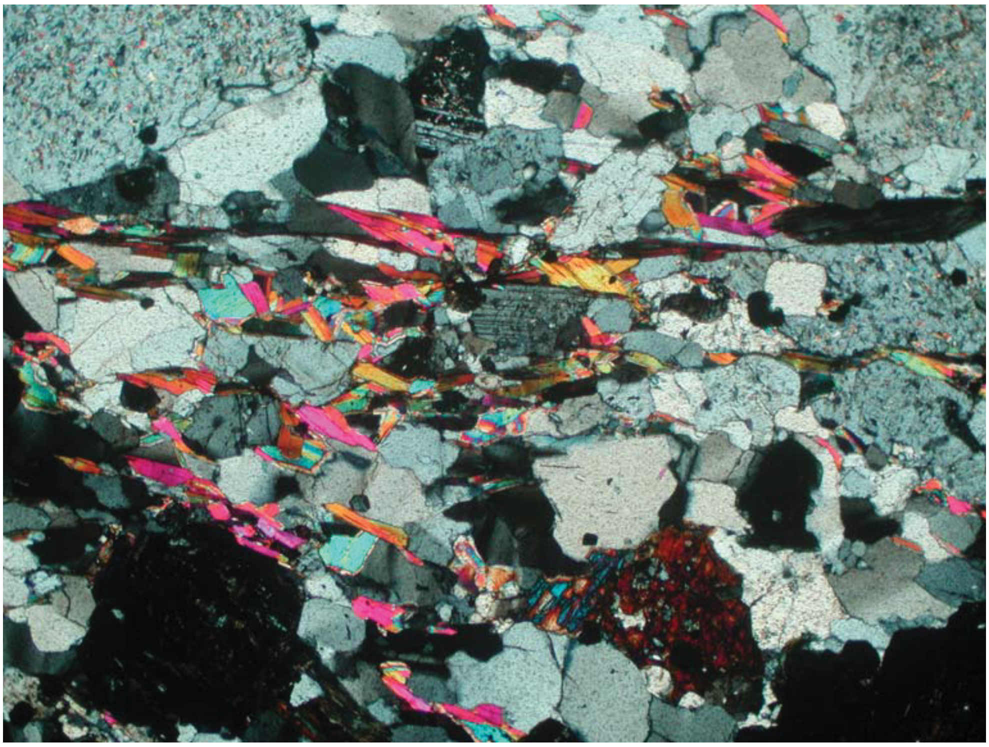

Figure 7-6: Image of the sample PG 1
CPLx4 (Picture size 75%).

The garnet partly generates big, idiomorphic grains and is overgrown with xenomorphic actinolite. Plagioclase is – due to its distinctly developed poly-synthetic lamellas – easily identifiable as a feldspar representative. Hypidiomorphic rutile and idiomorphic zircon constitute the accessory components of the thin section. Rutile appears in longish, round, gold-yellow grains; zircon is only visible under great magnification. The opaque phases are scattered across the whole section as single, irregular aggregates with small grain sizes.

7.1.1.5 Hornblende gneiss (HG 2, HG 4)

Under examination without microscope the thin sections HG 2 and HG 4 show different structures.

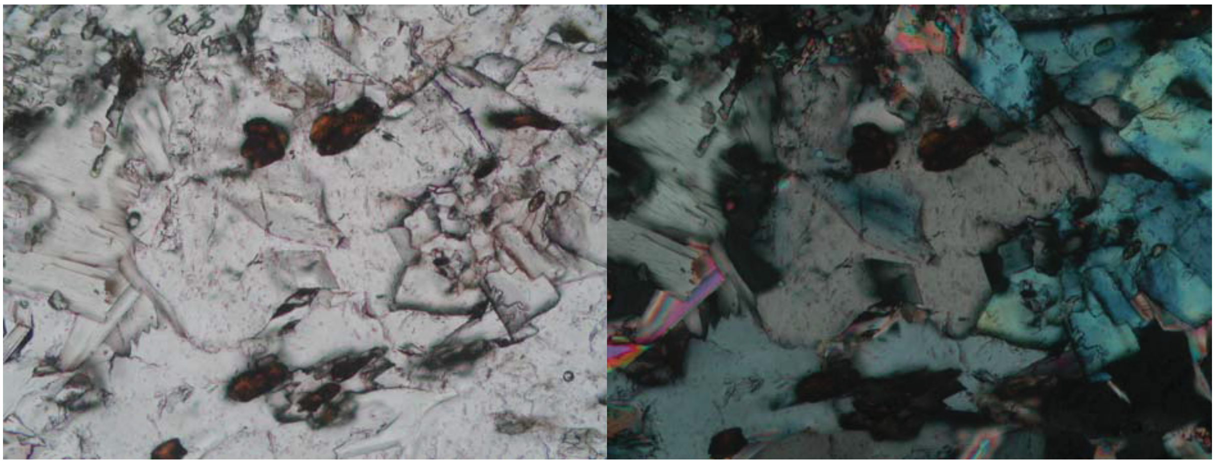

Figure 7-7: Rutile and clinozoisite in sample HG 2
Left: PPLx10 (Picture size 45%); right: CPLx10 (Picture size 45%).

In HG 2 the mineral grains are mainly unaligned; areas in which quartz appears mono-minerally alternate with fine-grained areas (figure 7-7). Next to quartz, hornblende, muscovite, chlorite, clinozoisite, rutile, biotite, plagioclase, garnet and zircon occur as further phases. Chlorite and muscovite show only slight orientation.

The HG 4 sample is better oriented than HG 2; however, no persistent layers are visible. Similar to sample HG 2, areas with quartz alternate with areas with fine-grained aggregates. Additionally, muscovite, hornblende, chlorite, plagioclase, potassium feldspar and - as minor minerals - garnet, clinozoisite, rutile and zircon are contained. The potassium feldspar shows perthitic decomposition and can thus easily be identified. Garnet appears in small grains irregularly spread over the whole section. In contrast to the thin section HG 2, HG 4 shows much less clinozoisite, but distinctly more garnet and feldspar.

Compared to the other Rabenwald rocks, the high amount of rutile contained in both hornblende samples is striking. Rutile occurs in idiomorphic, longish-round aggregates.

7.1.1.6 Calcsilicate (CS 1, CS 3)

When the thin section CS 1 is examined without microscope, no orientation can be detected. The sample contains two optically distinctly different areas. Large parts of the section are overgrown by clinozoisite and show a very irregular and dull appearance, without visible orientation of the mineral grains (figure 7-8). In contrast to that, very fine-grained areas with larger hornblende grains show significant orientation. The main components of the section are calcite, muscovite, hornblende, clinozoisite and titanite. Calcite builds xenomorphic mineral grains and can easily be identified by its abnormally high interference colors and its characteristic twin formation (figure 7-9). In contrast to the other rocks of the Rabenwald, the section contains considerable amounts of titanite, which is characterized by its high relief, its hypidiomorphic constitution and its twin formation. Further occurring mineral phases are biotite, apatite, quartz, opaque phases, and zircon. The opaque phases appear as xenomorphic grains of different sizes, irregularly spread across the whole section. Apatite constitutes round, hypidiomorphic grains, quartz is only subsidiarily contained.

Main components of the sample CS 3 are quartz and muscovite (figure 7-10). The section contains more biotite and large garnet grains (up to 0.5 cm) than CS 1; further minerals are clinozoisite, titanite, apatite, hornblende and calcite. Extremely fine-grained layers alternate with mono-mineral, coarser quartz layers.

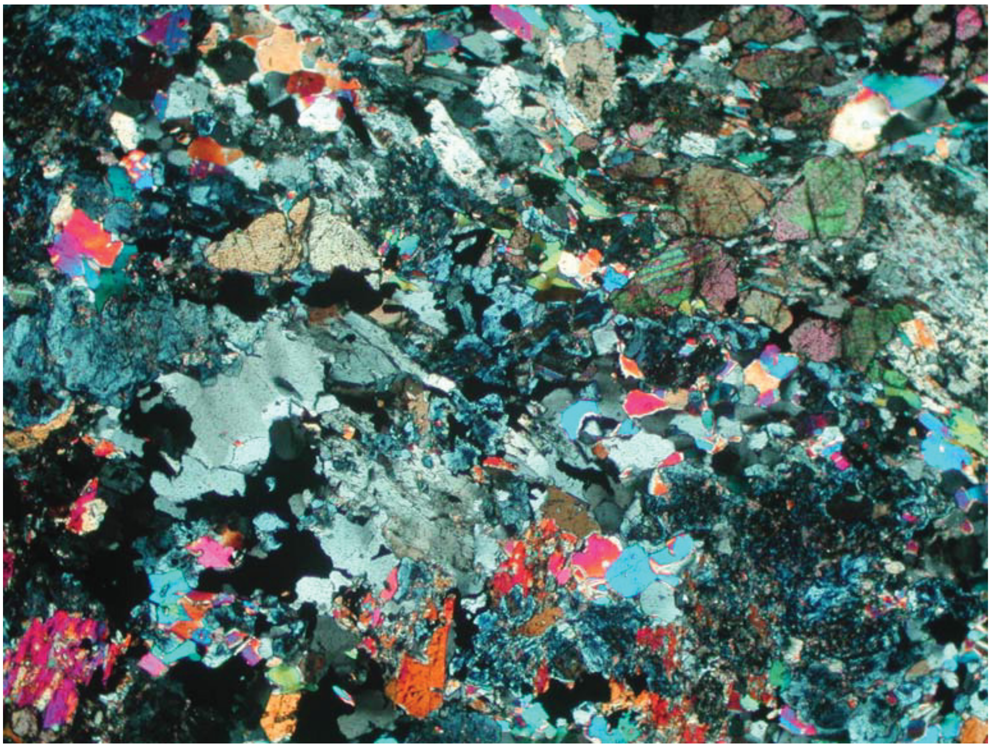

Figure 7-8: Image of the sample CS 1
CPLx4 (Picture size 75%).

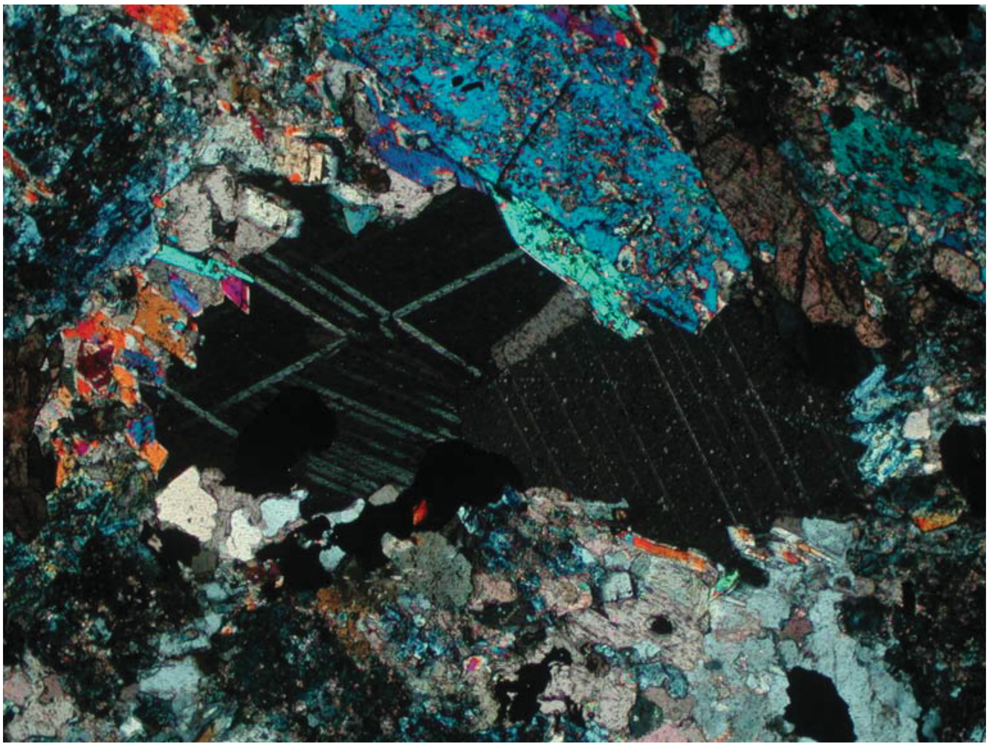

Figure 7-9: Twinning of calcite (Sample CS 1)
CPLx4 (Picture size 75%).

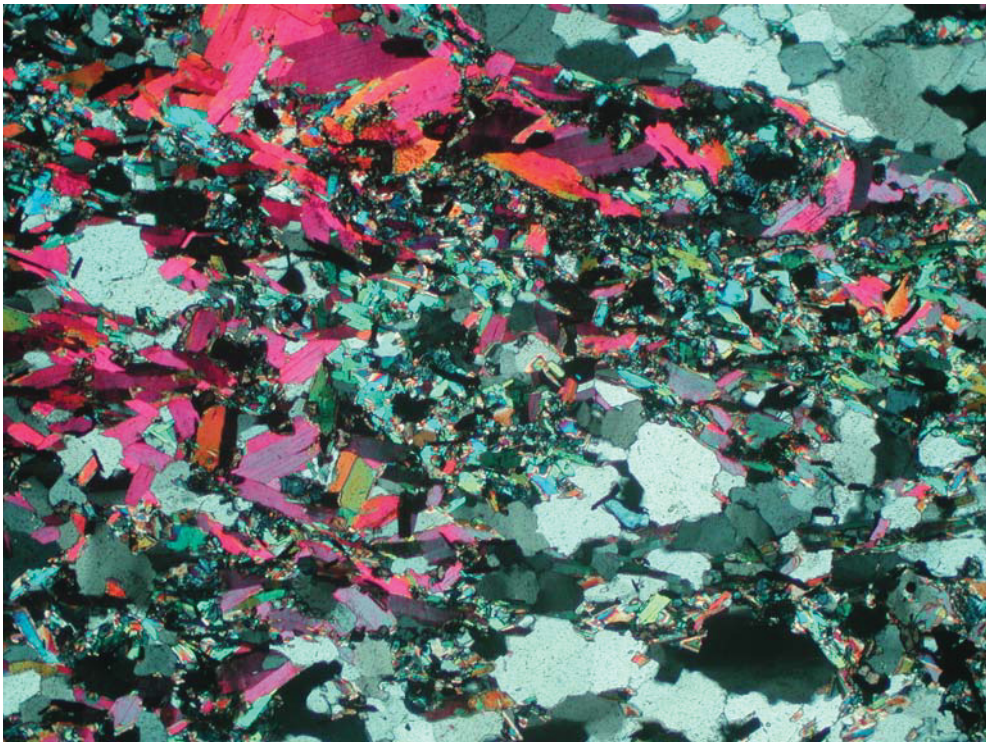

Figure 7-10: Image of the sample CS 3
CPLx4 (Picture size 75%).

The clinozoisite covers large areas of the section, which, together with the appearance of calcite, proves a similarity to CS 1. Some clinozoisite grains show yellowish colors, which is a sign for the starting conversion of the mineral to epidote. The opaque phases constitute a considerable part of the sample. Large, xenomorphic aggregates are present as well as fine, only partly aligned 'needles'.

7.1.1.7 Actinolite gneiss (AG 1, AG 3, AG 6)

The samples of the actinolite gneiss already show considerable amounts of sulphide minerals under macroscopic examination.

The minerals of the section AG 1 show significant orientation, which is characterized by coarser quartz layers and very fine-grained layers consisting of mainly quartz, actinolite and mica. Main minerals of the sample are quartz, actinolite, mica, clinozoisite and opaque phases. The undulatory quartz constitutes irregular, coarser grains in layers and lenses as well as very fine-grained components in the intermediate layers. Biotite and muscovite (the existing glimmers) are responsible for the orientation of the rock, together with quartz, the characteristically pleochroitic green actinolite gneiss and the opaque phases. The opaque phases appear as fine, long-stretched veins as well as needle-shaped grains, spread across the whole section. Vertical to this orientation, alternating veins with clinozoisite, chlorite and opaque phases can be identified (figure 7-11).

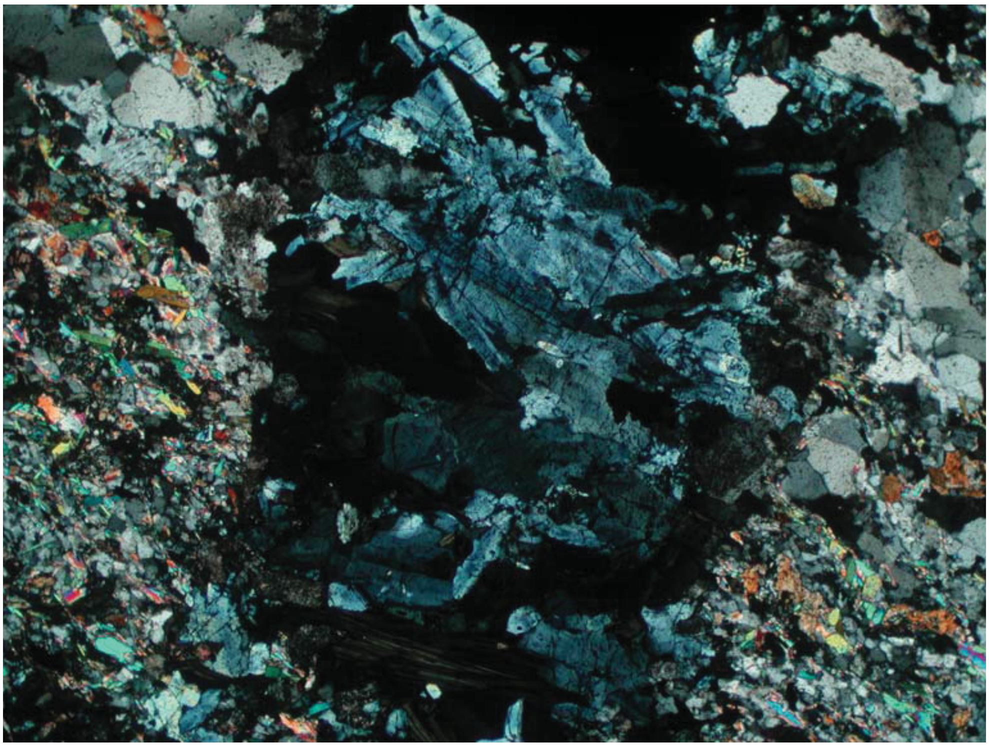

Figure 7-11: Vein of clinozoisite, chlorite and opaque phases in sample AG 1 CPLx4 (picture size 75%).

Thence the occurrence of the opaque phases in the section AG 1 is marked by two directions standing vertically on one another (figure 7-12). Garnet, plagioclase, titanite, rutile and zircon are the further components of the rock which are classifiable in the thin section.

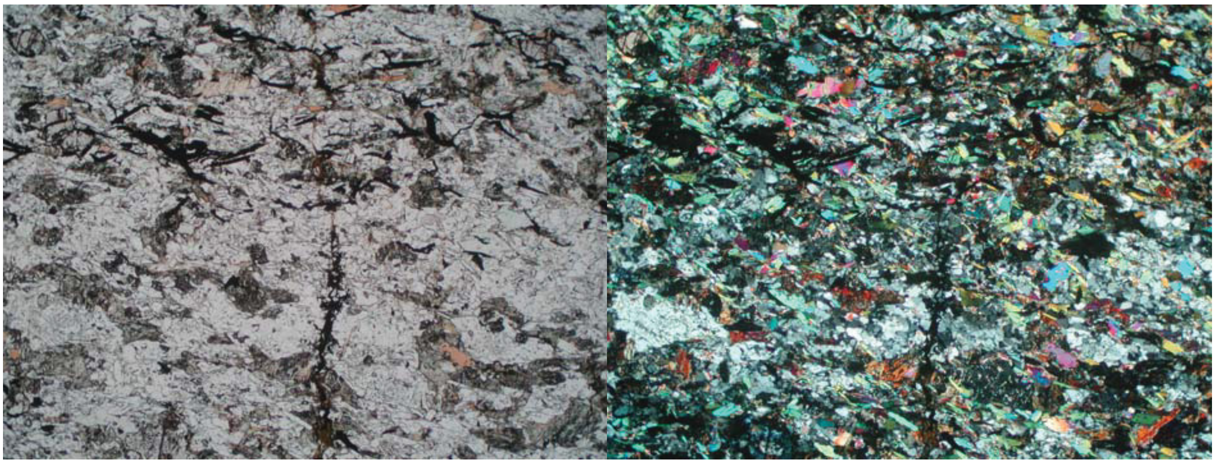

Figure 7-12: Two directions of oriented opaque phases in AG 1
Left: PPLx4 (Picture size 45%); right: CPLx4 (Picture size 45%).

For the thin sections AG 3 and AG 6 samples were chosen which showed considerable sulphide phases already from the outside and could thus be considered potential main causers of the formation of ARD. In other words: the samples are actinolith gneisses with elevated contents of sulphide phases.

Both sections show distinct orientation under examination without microscope. Mono-mineral layers of bigger, undulatory quartz grains alternate with fine-grained layers of quartz, muscovite, actinolite, plagioclase and opaque phases and thus define the lepidoblastic structure of the rock. Clinozoisite, titanite, rutile, apatite and zircon are the minor components of the sections. In sample AG 6 additionally garnet can be found.

Opaque phases form a considerable part in these rocks compared to the other rock types of the deposit. In AG 3, ‘networks’ between quartz grains, but also big, roundish grains characterize the irregularly appearing opaque phases (figure 7-13). In AG 6, the opaque phases constitute long, thin, branching veins and single, small, longish mineral grains which are spread across the whole section as well as bigger lenses (figure 7-14). Both samples are characterized by a distinct orientation of the opaque phases.

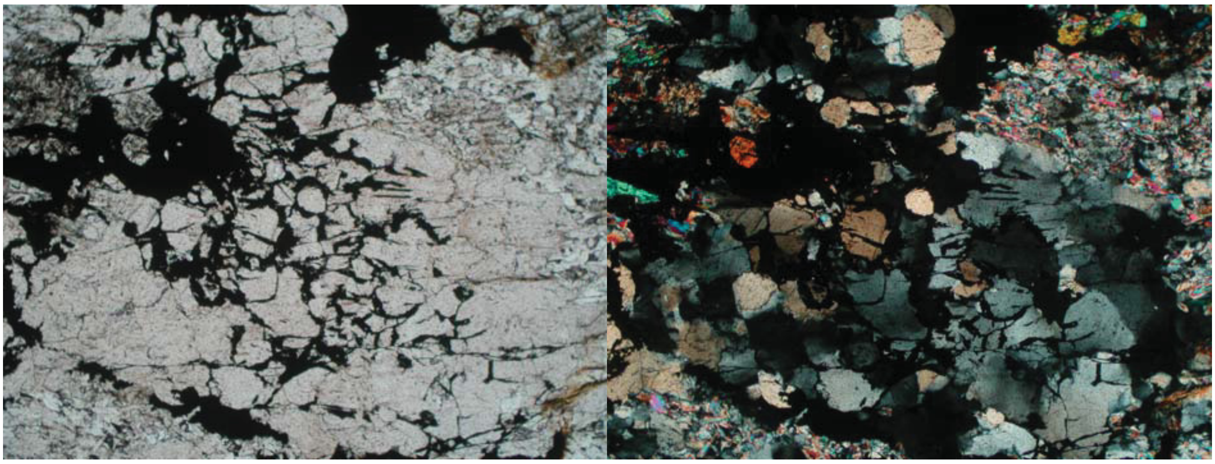

Figure 7-13: ‘Network’ of opaque phases in sample AG 3
Left: PPLx4 (Picture size 45%); right: CPLx4 (Picture size 45%).

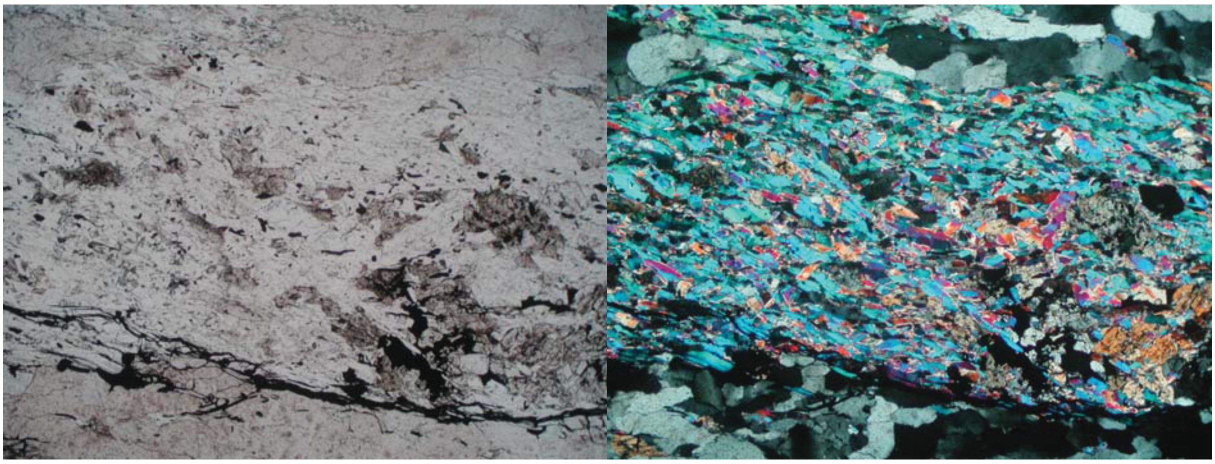

Figure 7-14: Image of sample AG 6
Left: PPLx4 (Picture size 45%); right: CPLx4 (Picture size 45%).

The talc samples of the deposit were not examined under microscope, since it was presumed that no sulphide minerals were contained therein. For that reason, talc is not considered a potential causer of the formation of ARD.

7.1.2 Ore microscopy

Four polished thin sections were prepared for the examination of the sulphide phases. Such actinolite gneiss samples (AG 3-6) were chosen which already showed sulphides under macroscopic examination.

Description of the microsections:

The four examined samples do not differ significantly from one another. For that reason, single descriptions of each section are neglected and a general description of the results is provided instead.

The distinguishable sulphides of the polished thin sections are pyrrhotite, chalcopyrite and pyrite. Furthermore, rutile, graphite and titanite can be classified under reflected light.

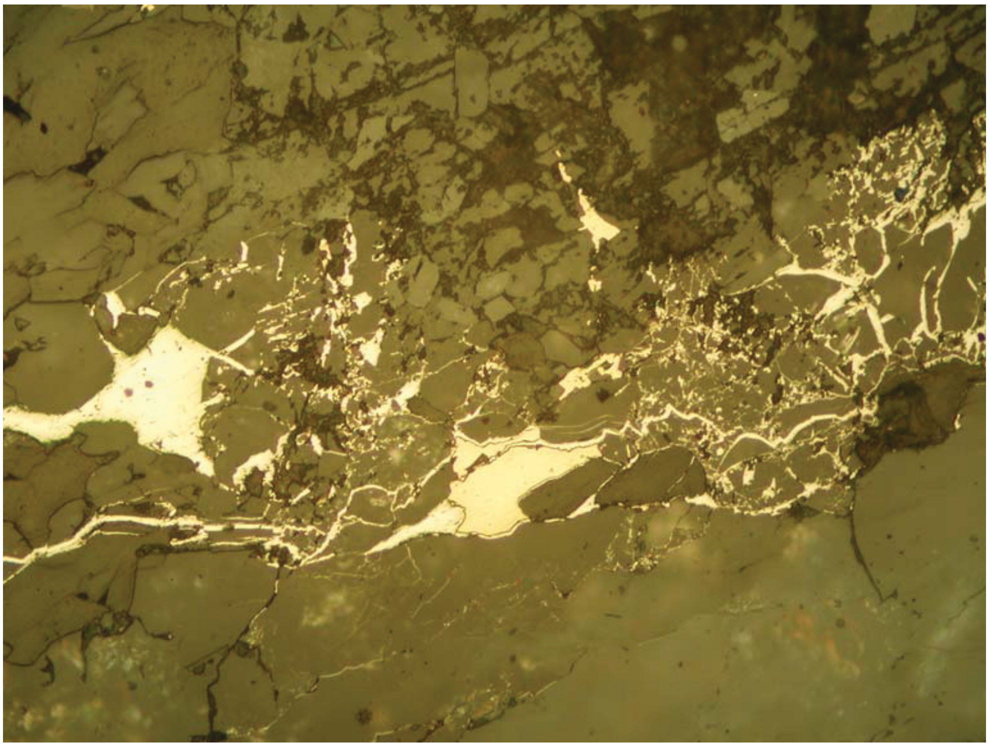

Figure 7-15: Pyrrhotite and chalcopyrite
CPLx20 (Picture size 75%).

Pyrrhotite is the sulphide phase which occurs most often. It appears in large, xenomorphic, partly longish grains as well as in shape of thin, oriented veins, also transverse to the cleavage (figure 7-15). In affected minerals, the pyrrhotite furthermore constitutes a ‘network’ of veins and large areas with irregular boundaries, in which it follows the fault zones in the rock. In some places of the microsections, corroded and affected aggregates can be identified. Pyrrhotite shows intergrowth with all sulphide phases classifiable in the microsections. The borders are mainly irregular, but also sharply-defined borders can be recognized, mainly to the basic matrix.

Chalcopyrite only occurs with xenomorphic grains in the samples. It appears (in about the same frequency) in singular grains and in connection with pyrrhotite (figure 7-15).

Pyrite is with exception of sample AG 6 the sulphide mineral with the least frequency in the samples. The sample AG 6 is characterized by higher amounts of pyrite and the occurrence of pyrite veins. In the other samples pyrite appears only in the form of small xenomorphic to hypidiomorphic grains in connection with pyrrhotite (figure 7-16).

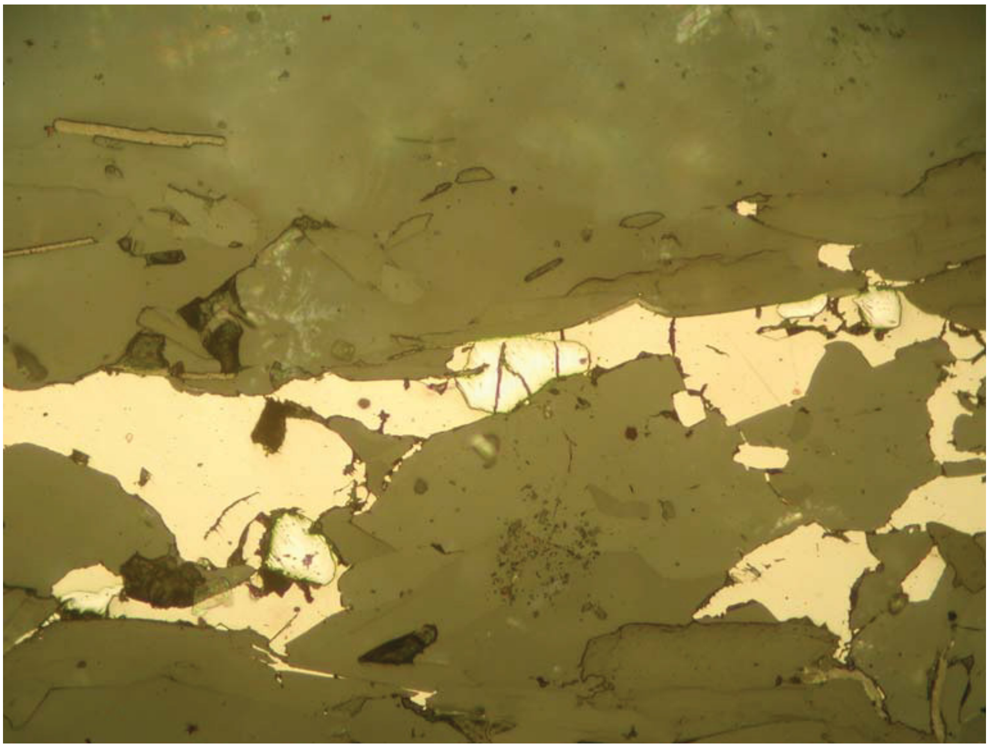

Figure 7-16: Hypidiomorphic pyrite in pyrrhotite
CPLx20 (Picture size 75%).

The four polished thin sections contain different amounts of rutile, titanite and graphite. Rutile constitutes small, roundish grains which are spread across the whole section as well as larger, xenomorphic aggregates. In pyrrhotite, titanite and pyrite rutile grains in different sizes can frequently be observed as inclusions. Particularly striking are the small rutile inclusions in the pyrite. Titanite is characterized through its xenomorphic development as well as through its idiomorphic aggregates in envelope shape (figure 7-17). The longish, lamellar graphite phases occur singularly as well as in connection with other mineral phases. The graphite appears corrugated and partly folded, but it shows a good orientation in all the samples.

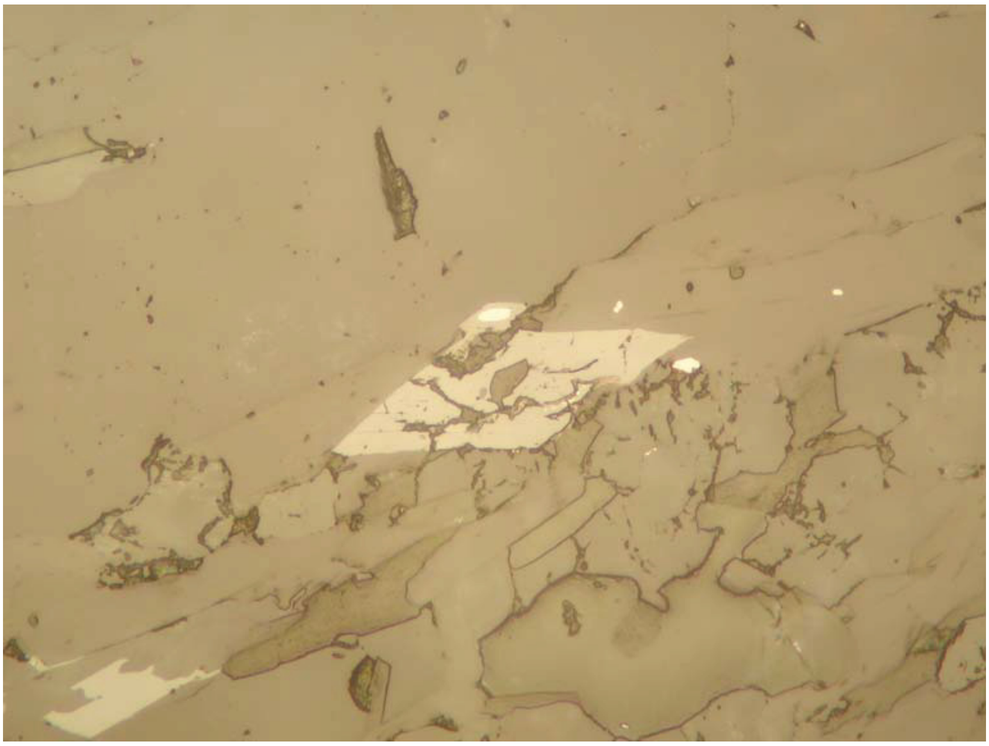

Figure 7-17: Idiomorphic titanite showing envelope shape CPLx20 (Picture size 75%).

7.2 Leco analysis

The rocks occurring at Rabenwald can be divided into two groups according to sulphur content. The rocks of the first group only contain approximately 0.01% sulphur (average of all examined samples of a rock type). To this group belong talc, leucophyllite, marble, granite gneiss, paragneiss and hornblende gneiss. The average sulphur content of these rocks can be seen in table 7-1 as well as in the APPENDIX.

Rock type	Number of samples	% S
Talc	6	0.007
Leucophyllite	5	0.009
Marble	5	0.003
Granite gneiss	4	0.010
Paragneiss	4	0.009
Hornblende gneiss	5	0.009

Table 7-1: The number of samples as well as the average sulphur content of the Rabenwald's low sulphur formations.

In contrast to the rocks of the above group, actinolite gneisses and calcsilicates are those formations of the surface mining, which contain distinctively more sulphur. However, the examined samples show a great variety in sulphur concentrations. The samples of the calcsilicate varied between 0.192% S (CS 5) and 1.462% S (CS 1); the sulphur content of the actinolite gneiss varied between 0.026% S (AG 10) and 2.801% S (AG 2). The actinolite gneiss thus presents the Rabenwald rock type richest in sulphur. The eleven analysed samples show an average content of 1.33% S. The nine samples of the calcsilicate contain on average 0.59% S. The sulphur contents of all examined calcsilicate and actinolite gneiss samples can be found in the APPENDIX.

Table 7-2 shows the results of the Leco analysis for the waste rock sample (WRS). In total six measurements were carried out. In the process an average sulphur content of 0.231% S could be determined for the waste rock sample.

Sample name	% S
WRS 1.1	0.202
WRS 1.2	0.201
WRS 1.3	0.288

Table 7-2: Sulphur concentrations of the waste rock sample.

The average sulphur contents of the examined 'artificial' bulk samples are summed up in table 7-3 and, in greater detail, in the APPENDIX. The measured values vary between 0.041% S (ABS 2_2) and 0.120% S (ABS 1_2, ABS 1_3).

Sample name	% S
ABS 1	0,111
ABS 2	0,051
ABS 3	0,080

Table 7-3: The average concentration of sulphur of the 'artificial' bulk samples.

According to the data in the above chart, an average sulphur content of 0.081% S can be assumed for the whole surface mining, using the bulk rock composition from drillhole data.

7.3 Leaching tests

In general it can be said that the leaching tests conducted as part of this thesis yielded similar trends as regards the dispersion of the pH values measured. Nearly all sample solutions that were analyzed showed an increase in pH values within a very short time (max. 100 hrs) reaching a maximum value that is not reached again during the course of the entire test period. Subsequently the pH curves form a basin, followed by a further increase and then finally a slight decrease before constant values are finally reached. The definitive constant values are reached by nearly all leaching tests within a short time span (max. 800 hrs). The results of the individual leaching tests will now be described.

7.3.1 Leaching tests of the waste rock sample (LT 1)

In order to simplify things, the individual samples and their basis fertiliser solutions will be described as follows in this chapter: <5 mm + H_2O = Sample 1; 5-20 mm + H_2O = Sample 2; >20 mm + H_2O = Sample 3; <5 mm+ H_2O + HCl = Sample 4; 5-20 mm + H_2O + HCl = Sample 5; >20 mm + H_2O + HCl = Sample 6.

The pH curves of LT 1 in general follow the trend described above. It is worth noting in particular, the synchronicity in the course of the curves 2, 3, 4 and 5, 6 (see figure 7-18). Also striking are the small differences between the values of the two basis fertiliser solutions with different initial pH values. These differences become insignificant as the testing progresses.

The first measurement after 48 hrs showed an increase in pH values in distinctly alkaline ranges with values of between 7.33 (sample 6) and 8.99 (sample 1). Graphical interpolation of the values leads us to believe that the maximum values of the initial increase lie between 60 and 80 hrs for all samples (figure 7-18). This value is also the highest pH value measured for curves 1 to 4 during the duration of the tests. The extent of the increases is dependant on the grain size of the samples as well as the basis fertiliser solution used.

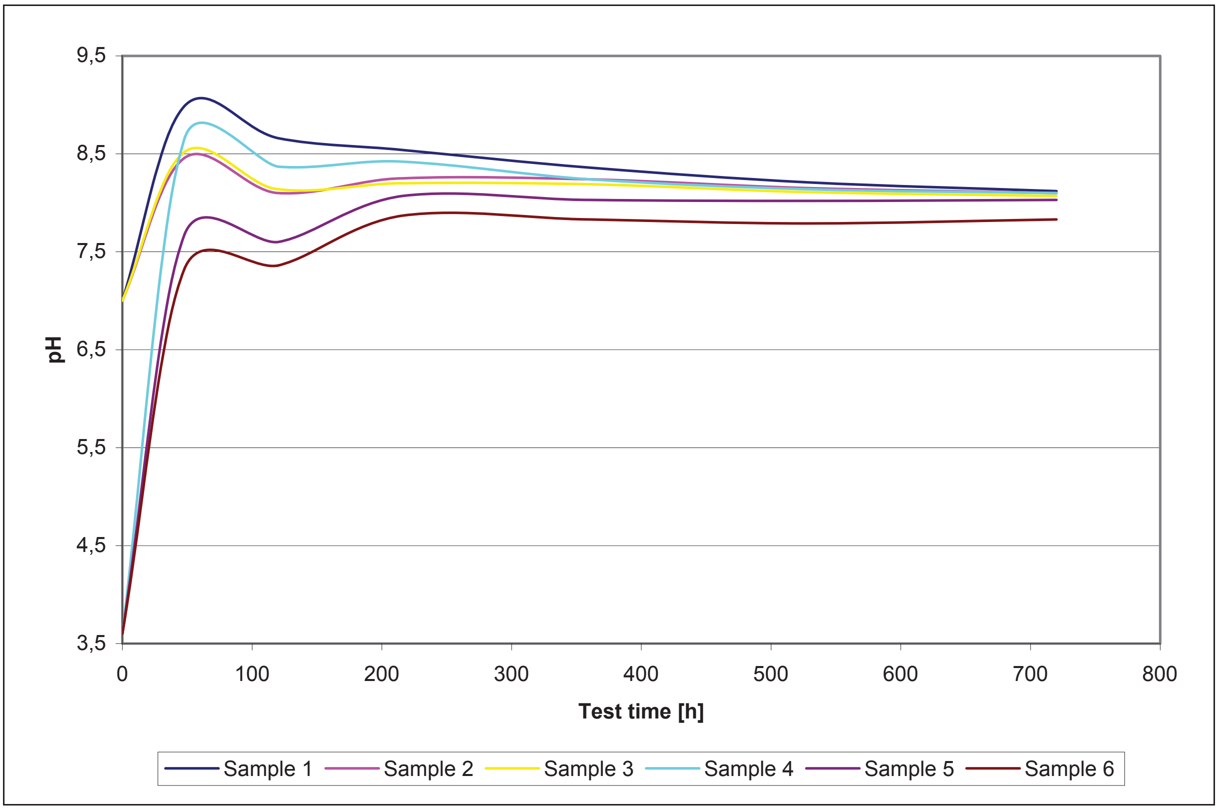

Figure 7-18: Interpolated pH curves of the leaching tests of the waste rock sample (LT 1) with an experiment time of 0 to 720 hrs.

The highest pH values were measured in solutions 1 and 4, which were also the samples with the finest grains (<5 mm). The significant change in the pH value of sample 4 is of particular note. Sample 4 starts with a value of 3.5 and reaches a value of 8.66 within 48 hrs. This puts the pH value of sample 4 only slightly behind that of sample 1 which incidentally displayed an initial pH value of 7. The pH curves of samples 2 and 3 proceed synchronously and have a value of approx. 8.5 at the first measurement. Whilst samples 5 and 6 display the lowest pH values they actually exhibit the highest proportional increase after sample 4.

After this initial increase the pH values of all samples fall. At a test time of approx. 120 hrs the curves form distinct indentations of differing intensities. The run of the curves at this point seem to be dependant on the pH of the sample solutions rather than the grain sizes of the samples. Samples 2 and 3 as well as 5 and 6 respectively display symmetric pH curves independent of the different grain sizes. Most striking, is the indentation displayed in samples 5 and 6, whilst the indentation in sample 1 on the other hand, is hardly noticeable. Subsequently, with the exception of sample 1, all samples display a second increase in pH values at a test time of between 200 and 250 hrs. Samples 5 and 6 reach values of 7.9 and 8.1 respectively, and are the highest measured pH values during the course of the whole experiment for those samples.

The pH of sample 1 already begins a steady decrease after a test time of approx. 120 hrs. This trend can only be seen after a time of 200 to 250 hrs in the other samples. The fine-grained samples 1 and 4 show a more significant decrease in pH values.

After a test time of between 700 and 800 hrs the samples level off to their final values and then only display small anomalies and fluctuations between this point and the end of the experiment at 3840 hrs (figure 7-19). What is particularly noticeable is the approximation of the six samples’ curves despite their different grain sizes as well as their different initial pH values. The last measurements are all within a pH range of between 7.8 (sample 6) and 8.02 (samples 4, 5).

The times at which the pH measurements were taken as well as the values that were noted at these times can be found in the APPENDIX.

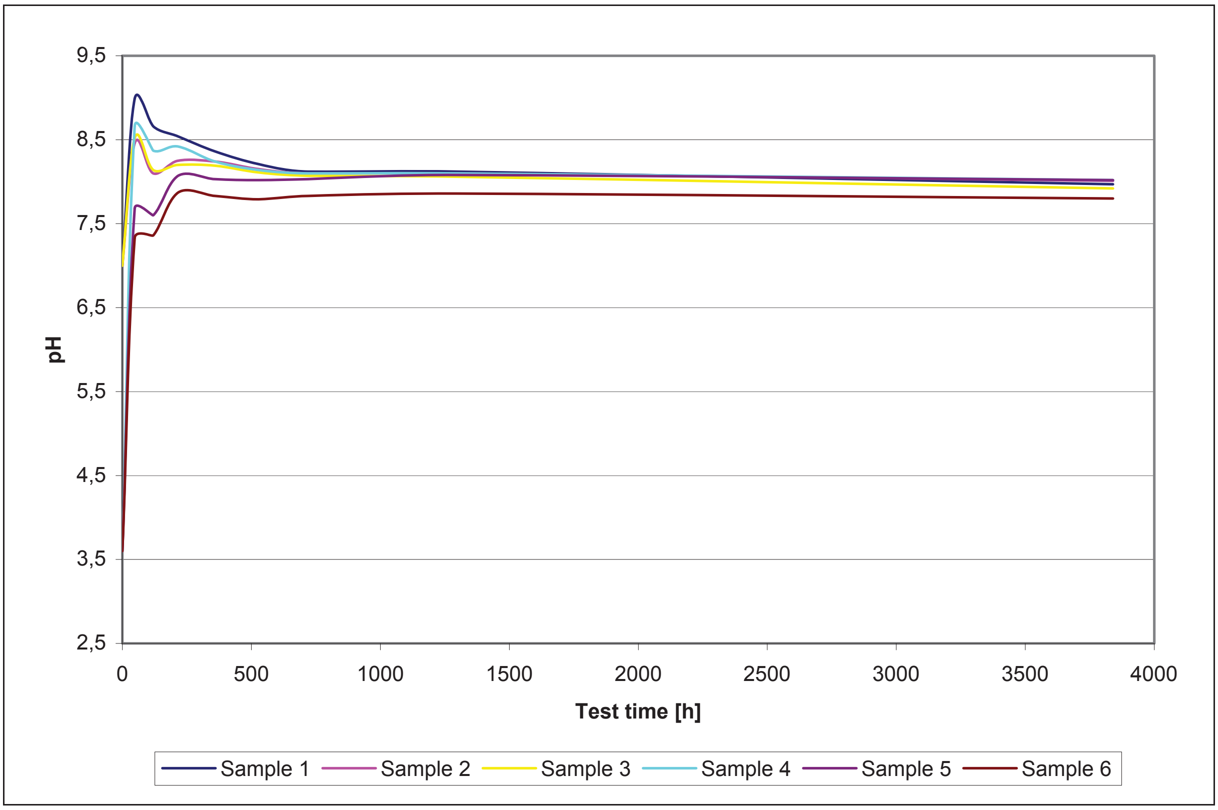

Figure 7-19: Interpolated pH curves of the leaching tests of the waste rock sample (LT 1) over the whole test time of 3840 hrs.

7.3.2 Leaching tests of the 'artificial' bulk samples (LT 2)

Just as in the leaching test LT 1 described above, the leaching test of the 'artificial' bulk samples (LT 2) also shows a sudden increase in pH values even during the first measurement after 71 hrs. The sample BS 3 consisting of rock powder measuring 8.93, displays the highest pH value. BS 1 on the other hand, with a pH of 7.44 displays the lowest value. BS 2 which has the same composition as BS 1 plus 2 weight.% marble is defined as having a pH of 8.06. During the remainder of the experiment the pH curves of the three samples differ (see figure 7-20) and will thus be described separately.

BS 1 is the only one of the three samples which shows a further increase in pH value up to a test time of between 140 to 150 hrs with an interpolated maximum value of approx. 7.66. After a small drop to 7.62 the continuing trend shows an increase in the pH values up to an experiment time of approx. 1150 hrs. At this point, BS 1 gives its highest pH value measurement of 7.82. At the end of the test time after 1870 hrs, the sample has a pH value of 7.78.

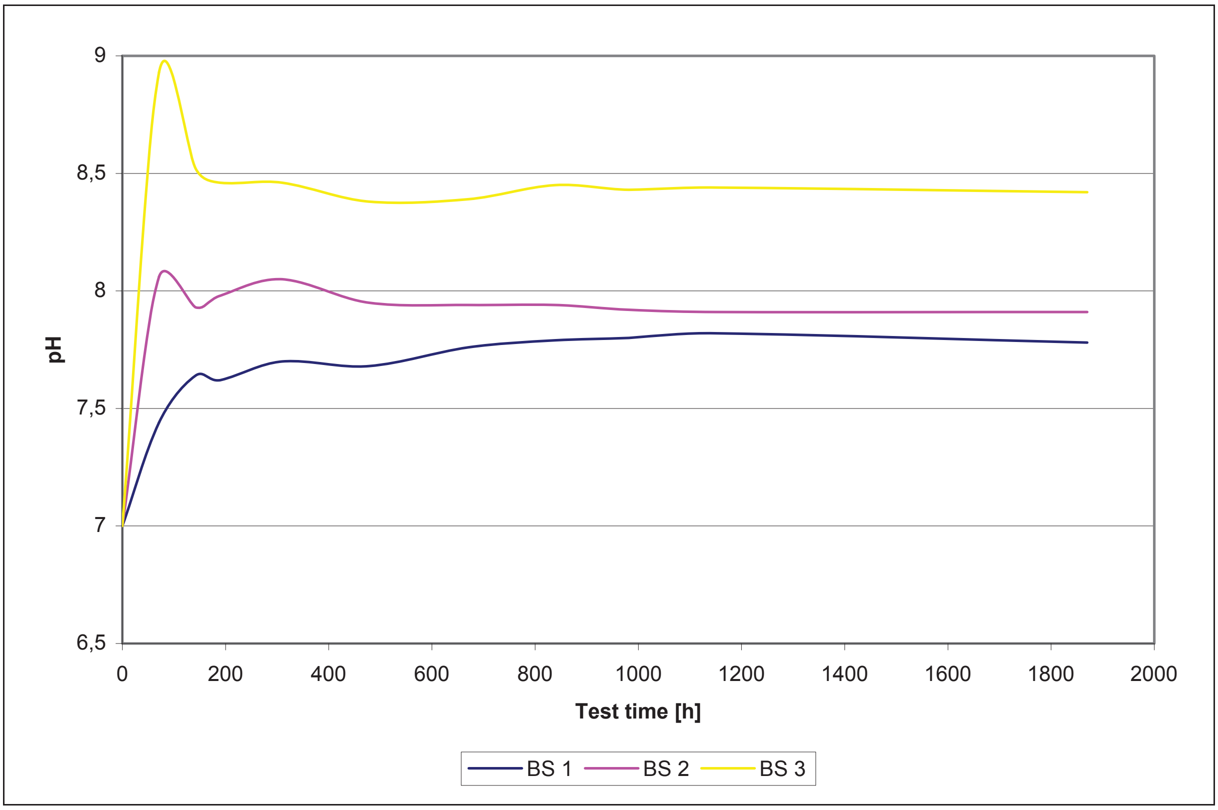

Figure 7-20: Interpolated pH curves of the leaching tests of the ‘artificial’ bulk samples (LT 2) over the whole test time of 1870 hrs.

The first increase in the pH of sample BS 2 ends at between 70 and 80 hrs with a maximum value of approx. 8.08. The ensuing indentation of the pH curve is defined by a minimum of 7.93 at 142 hrs. After a second increase in the pH value reaching 8.05 (310 hrs), the pH begins a continuous fall which until 478 hrs is steep before finally levelling out. After a test time of only approx. 500 hrs there appear to be only small fluctuations in the measured values. The last measurement after 1870 hrs gave a pH reading of 7.91. This measurement is also the lowest pH measured for this sample during the whole course of the experiment.

Sample BS 3 just like BS 2 also reaches a maximum pH value of approx. 8.97 in the initial increase at a test time of between 70 and 80 hrs. There then follows a steep fall in the pH reaching a minimum value of 8.46 at 190 hrs. This value remains constant over a time period of approx. 120 hrs. Finally the pH curve forms an indentation with the lowest value measured during the course of the experiment (8.38 at 478 hrs) followed by a further increase between 670 and 838 hrs. From this point, the values remain constant until the end of the test. The last measurement after 1870 hrs yields a pH of 8.42.

To summarise, it can be maintained that the pH curves of the three samples examined differ from each other but at the same time display some similarities and consistent trends. All samples show an initial increase in pH values; however this initial increase drops out to different degrees and lasts a different amount of time in each sample. BS 2 and BS 3 both reach their experiment maximum values within the time framework of the first increase, between 70 and 80 hrs. The increase in BS 1 lasts longer (approx. 140 to 150 hrs) and does not reach the maximum value reached in the entire experiment. The subsequent indentation is again time-delayed in its formation; this is particularly true of BS 2 and to a lesser extent also true of BS 1. BS 2 reaches the minimal pH values at approx. 50 hrs before BS 1 (142 hrs and 190 hrs respectively). A further similarity in the samples BS 1 and BS 2 is the second increase in pH values occurring between 190 and 310 hrs (in BS 1) and between 142 and 310 hrs (in BS 2). A drop in the values of both samples can also be identified in the subsequent reading taken after 478 hrs.

The times at which the pH measurements were taken as well as the values that were noted at these times can be found in the APPENDIX.

Figure 7-21 shows the pH curves of the LT 2 samples as well as the curves of the three LT 1 samples which used distilled water as basis fertiliser solution. A comparison of the individual curves allows us to recognise similarities between the fine-grained sample 1 and BS 3 as well as between 2, 3, BS 2 and with certain qualifications BS 1. This is all the more remarkable due to the fact that (as already described in the chapter ANALYTICAL METHODS) the waste rock sample taken for LT 1 exhibit an unrealistic stone composition and is therefore only representative to a limited extent. The higher pH values of samples 1, 2 and 3 when compared to those of BS 1 and BS 2 are particularly striking, as is the distinctive levelling off of the values of LT 1 with a fluctuation range of only 0.05 after only 720 hrs. The former is presumably caused by a higher than average concentration of calcite in the waste rock sample.

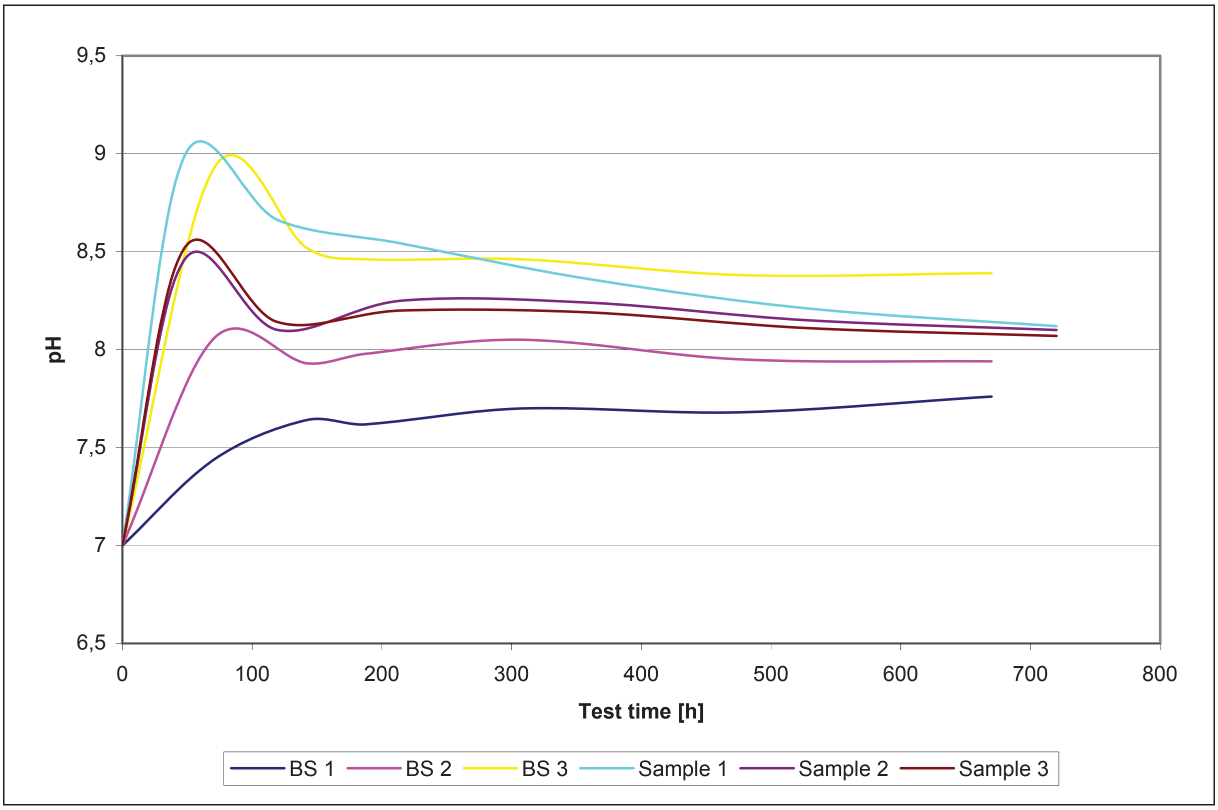

Figure 7-21: Interpolated pH curves of the samples 1, 2, 3 from LT 1 and the samples of LT 2 with a test time of between 0 to 720 hrs (LT 1) and 0 to 670 hrs (LT 2) respectively.

7.3.3 Leaching test of actinolite gneiss (LT 3)

In figure 7-22 the pH curve of LT 3 over a test time of between 0 and 569 hrs is displayed. The curve of the actinolite gneiss, in the same way as those of the previous samples, already displays an increase in pH from an initial 7 up to 7.28 at the first reading taken after 16 hrs. This increase, however, falls to a considerably weaker extent as compared to the other examined samples taken from the Rabenwald surface mining. The actinolite gneiss reaches its maximum pH value of approx. 7.35 after a test time of between 50 and 60 hrs. After reaching this peak, the pH value falls uninterruptedly until it settles at a value of just under 5.6 after a test time of approx. 550 hrs. This value remains constant over 1500 hrs. Up until the end of the experiment the pH curve subsequently displays a trend of falling values (figure 7-23). The indentation in the curve at between 2000 and 3000 hrs does not fit in with this trend and can be attributed to a measurement error. The last measurement taken after 4386 hrs gave a final pH reading of 5.36. The results of LT 3 are displayed in the APPENDIX.

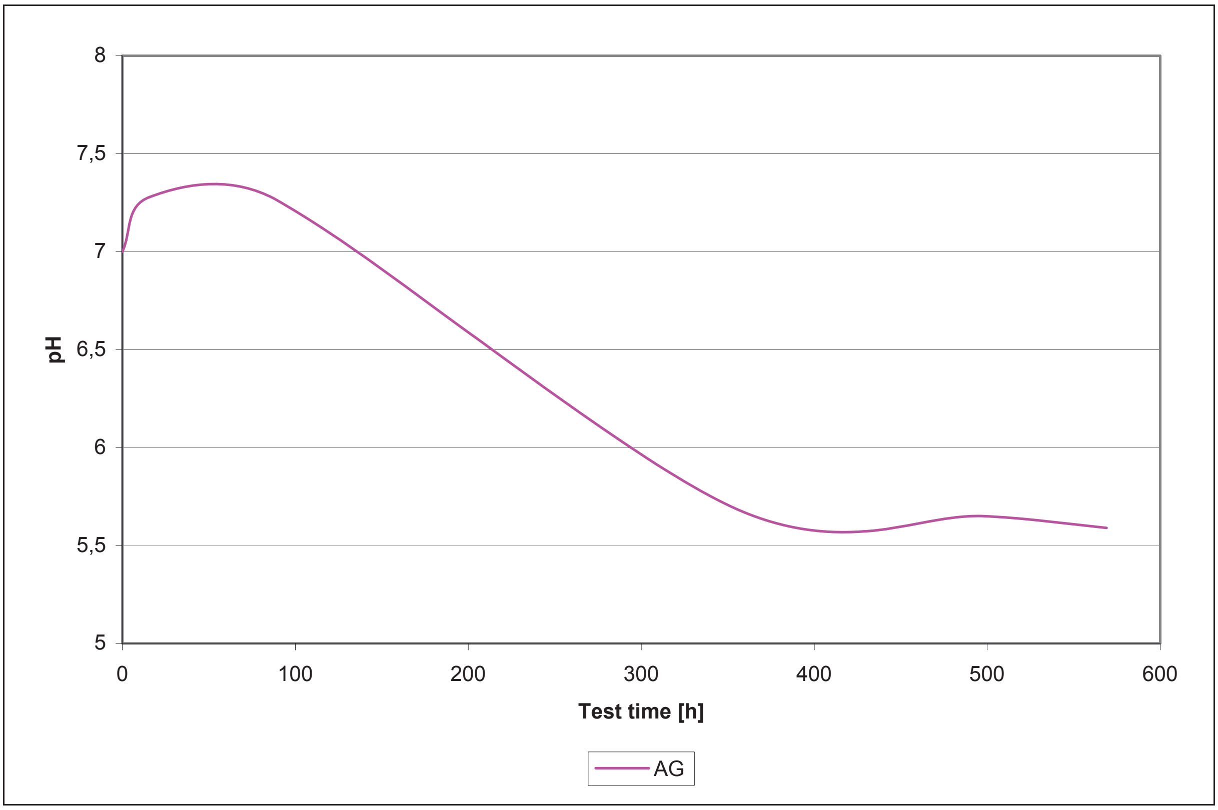

Figure 7-22: Interpolated pH curve of the actinolite gneiss of the LT 3 with a test time of 0 to 569 hrs.

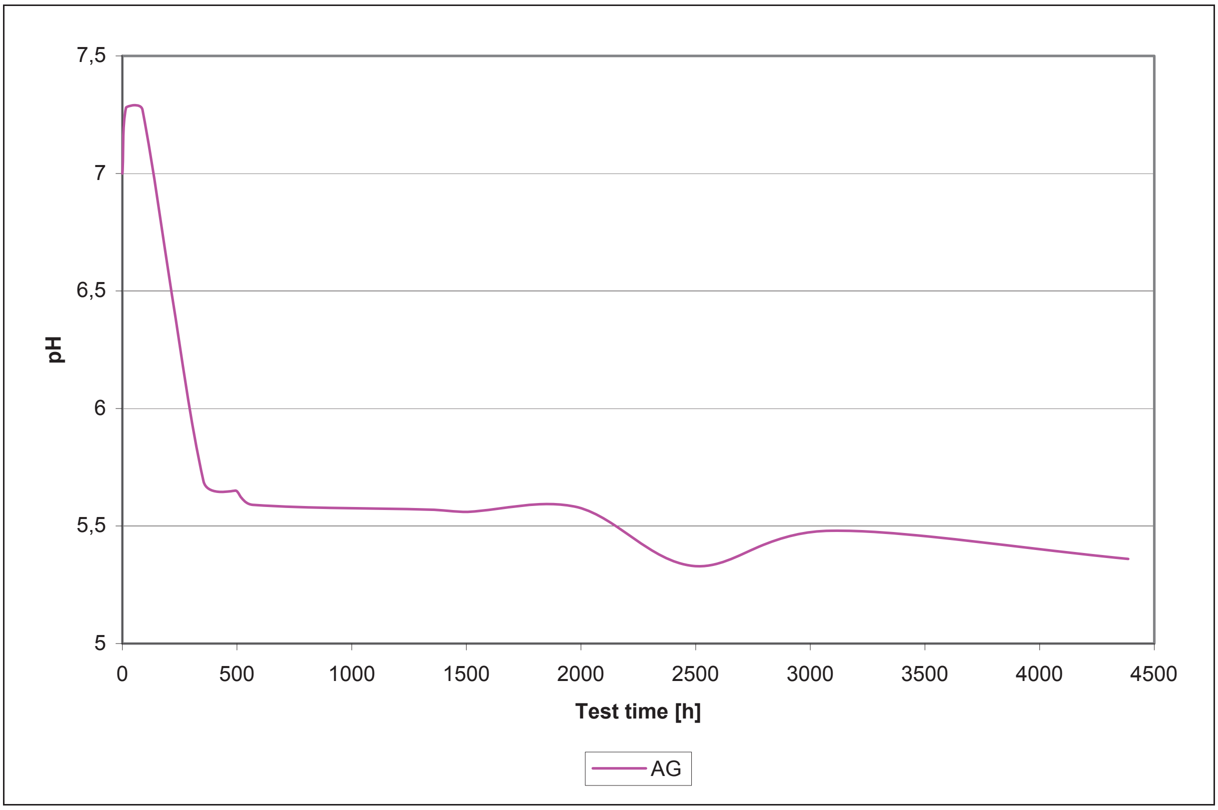

Figure 7-23: Interpolated pH curve of the actinolite gneiss of the LT 3 over the whole test time of 4386 hrs.

7.3.4 Leaching test acid rock drainage (LT 4)

The pH curves of LT 4 show two distinctly different trends (figure 7-24 and figure 7-25). The curve of the copper schist from Poland (CuS) and that of the sample from Pine Point, Australia (PP) run similarly over a longer period of time. The run of the pH curve for the sample from Waldenstein, Carinthia (Wa) is however significantly different. The pH values of both CuS and PP after a test time of only 4 hrs lie clearly within alkaline ranges (8.74 and 8.67 respectively). The subsequent decrease in pH ensues in both samples but to different degrees.

CuS falls down to a pH of 8.43 at the second measurement taken after 26 hrs. The minimum pH of 8.35 is reached after an experiment time of 142 hrs and this value remains for approx. 190 hrs. After the fifth measurement (332 hrs) there ensues a further increase in pH to values of between 8.7 and 8.8 after approx. 1100 hrs. From this point in time the values remain virtually constant until the end of the experiment. After 3691 hrs a pH of 8.6 was measured for this sample.

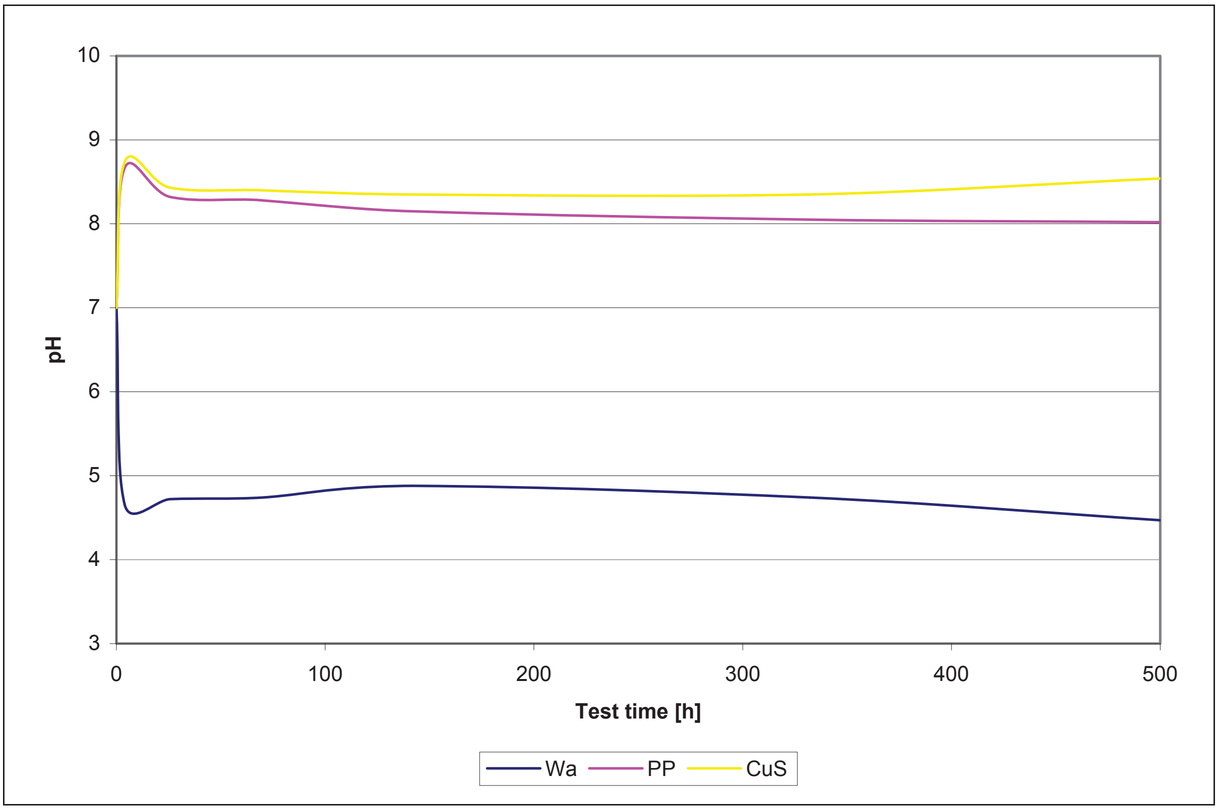

Figure 7-24: Interpolated pH curves of the LT 4 samples with a test time of 0 to 500 hrs.

After the first measurement, the sample PP shows a clearer decrease in pH down to a minimum value of 8.02 after 500 hrs. Subsequently, over a time span of approx. 2500 hrs, the pH value is only affected by small fluctuations within a range of two tenths (maximum: 8.2 at 2515 hrs; minimum: 8 at 3019 hrs). The last measurement, with a pH of 7.83, points to there being a downward trend in pH after an experiment time of approx. 2500 hrs.

In contrast to CuS and PP, the sample Wa lies clearly in acidic pH ranges (4.65 after 4 hrs) after the first measurement. After an increase in the curve reaching a highest pH value of 4.88 (142 hrs), the pH value then drops at a continuous rate until the end of the test. The last measurement of the sample, giving a pH measurement of 3.9 (3691 hrs) is at the same time the lowest measured value during the course of the whole experiment (figure 7-25).

The times at which the pH measurements were taken as well as the values that were noted at these times can be found in the APPENDIX.

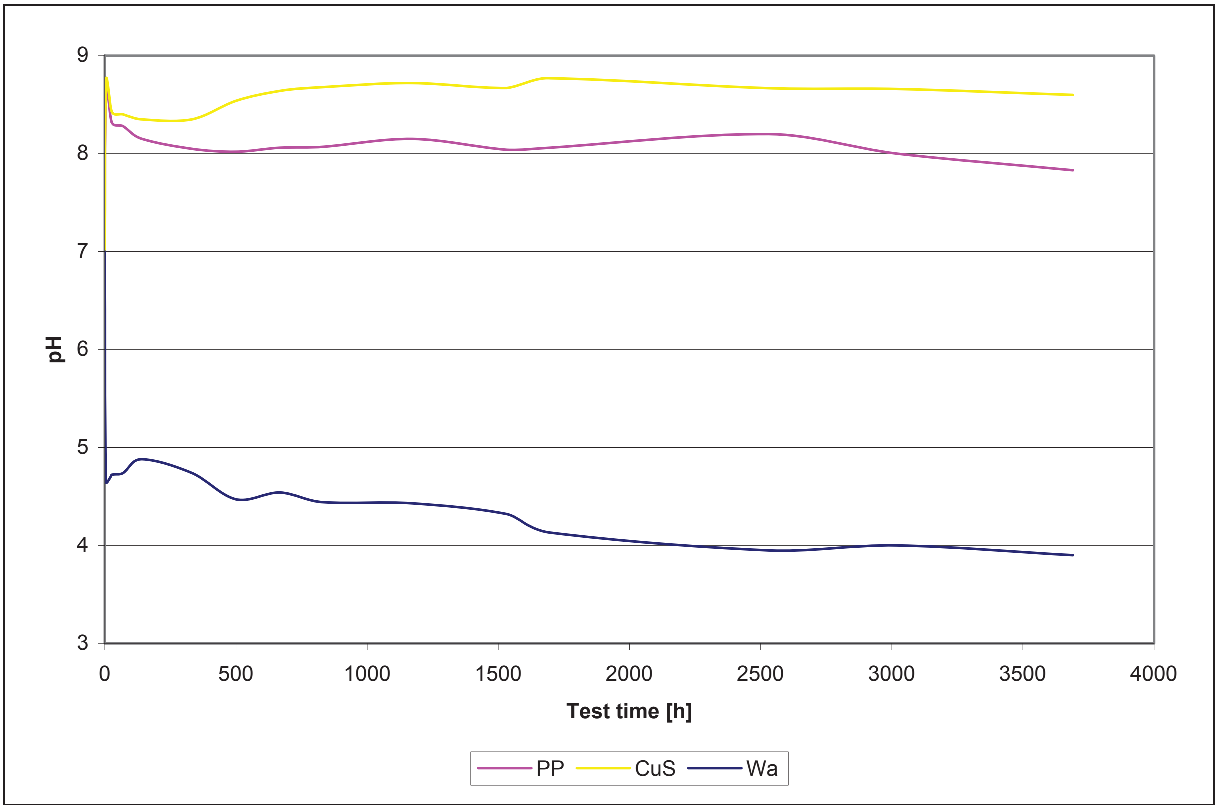

Figure 7-25: Interpolated pH curves of LT 4 over the whole test time of 3691 hrs.

7.4 Ion chromatography

The basis fertiliser solutions of LT 1 were investigated for their content of the anions F, Cl, SO_4, Br, NO_3, PO_4 as well as the cations Li, Na, K, Mg and Ca by means of ion chromatography. The results of the tests for the anions Br, NO_3, PO_4 are however irregular and inconclusive and therefore not usable. Due to the scope and definition of this investigation a special focus was placed on the SO_4 values of the samples. For this reason, these results will be discussed in detail in this chapter. The results of the ions are not mentioned here but are instead displayed in the APPENDIX.

On closer examination of the SO_4 curves of the individual samples one can identify correlations between the samples with the same grain sizes (figure 7-26). The samples with acidic basis fertiliser solutions tend, in principle, to have marginally higher values. Up to an experiment time of 120 hrs, all samples show a significant increase in SO_4 values. From this point the curves begin to level out. This effect is not as pronounced in the fine-grained samples 1 and 4. The curves of these samples run synchronously and show a continuous increase up to the end of the test. After 2640 hrs they exhibit double the concentration as compared with the other samples, with values of 110.3 mg/l (sample 1) and 113.1 mg/l (sample 4).

The curves of samples 2, 3, 5 and 6 already show a clearly weaker increase after a test time of 120 hrs but do not level off to constant values at the end of the experiment. The measured values at the end of the test are higher for the samples 2 and 5 (41.2 mg/l and 46.9 mg/l respectively) than for the samples 3 and 6 (25.1 mg/l and 27.9 mg/l respectively).

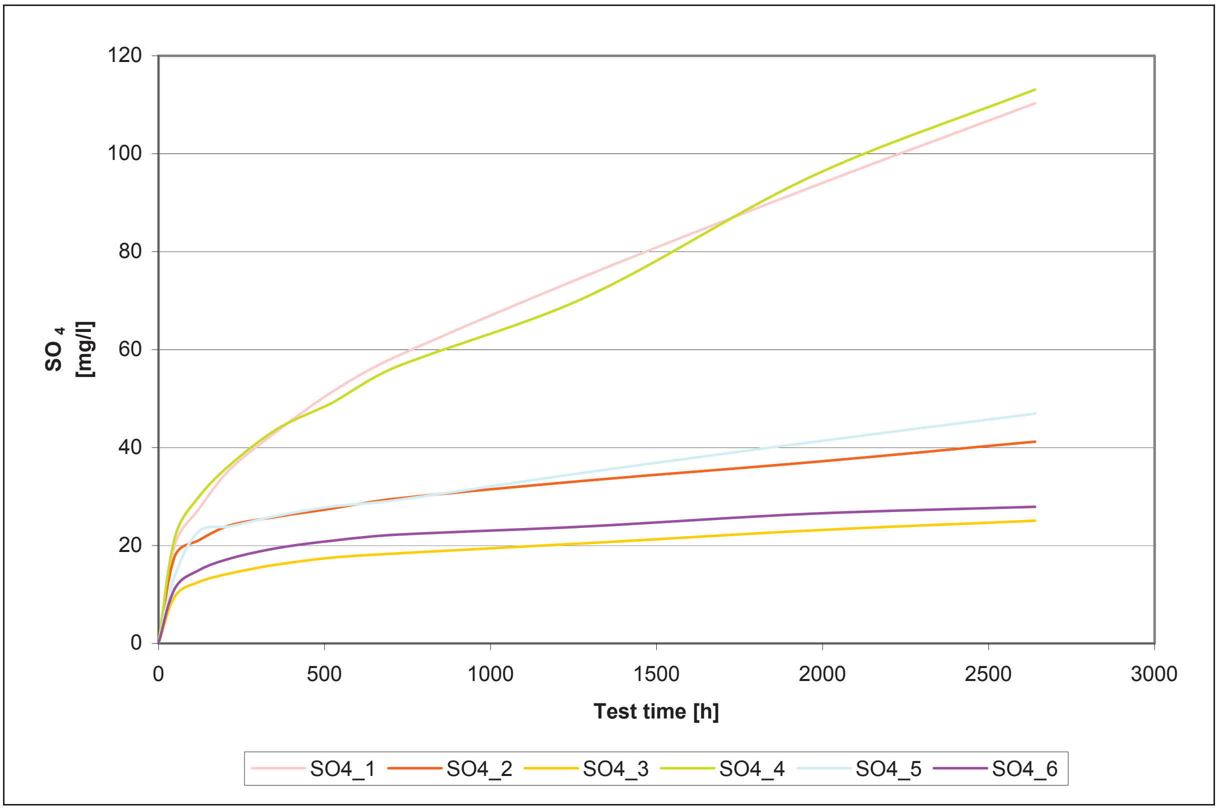

Figure 7-26: Interpolated SO_4 curves of the LT 1 with an experiment time of 2640 hrs.

The ion chromatography of the LT 2 basis fertiliser solutions also only produced reliable results for the anions F, Cl, SO_4 and the cations Li, Na, K, Mg and Ca. The results of the examined samples can be seen in the APPENDIX. All curves, with the exception of those of the calcium show similar trends. The highest values belong to the fine-grained sample BS 3 whilst the curves of samples BS 1 and BS 2 display an almost synchronous course.

The results of SO_4 follow this trend (see figure 7-27) and moreover, they display similarities to the SO_4 curves of LT 1. The first measurements taken for samples BS 1 and BS 2 show a significant increase in values up to 33.8 mg/l and 28.6 mg/l respectively. Both curves level out after a test time of between 160 and 180 hrs but continue to rise consistently up to the end of the measurements after 1870 hrs. The last measurement gave the following SO_4 values: 53.4 mg/l (BS 1) and 58.4 mg/l (BS 2).

The course of the BS 3 curve displays a few anomalies. Due to the fact that the measurement taken after an experiment time of 142 hrs was inconclusive, the first

SO_4 value (71.2 mg/l) could only be taken after 478 hrs. From this point in time, the course of the BS 3 curve, when followed to a test time of approx. 1000 hrs, only shows a very steady increase (an SO_4 maximum value of 87.1 mg/l can be seen at 982 hrs). Subsequently the SO_4 value falls slightly to 81.0 mg/l at the last measurement taken after 1870 hrs.

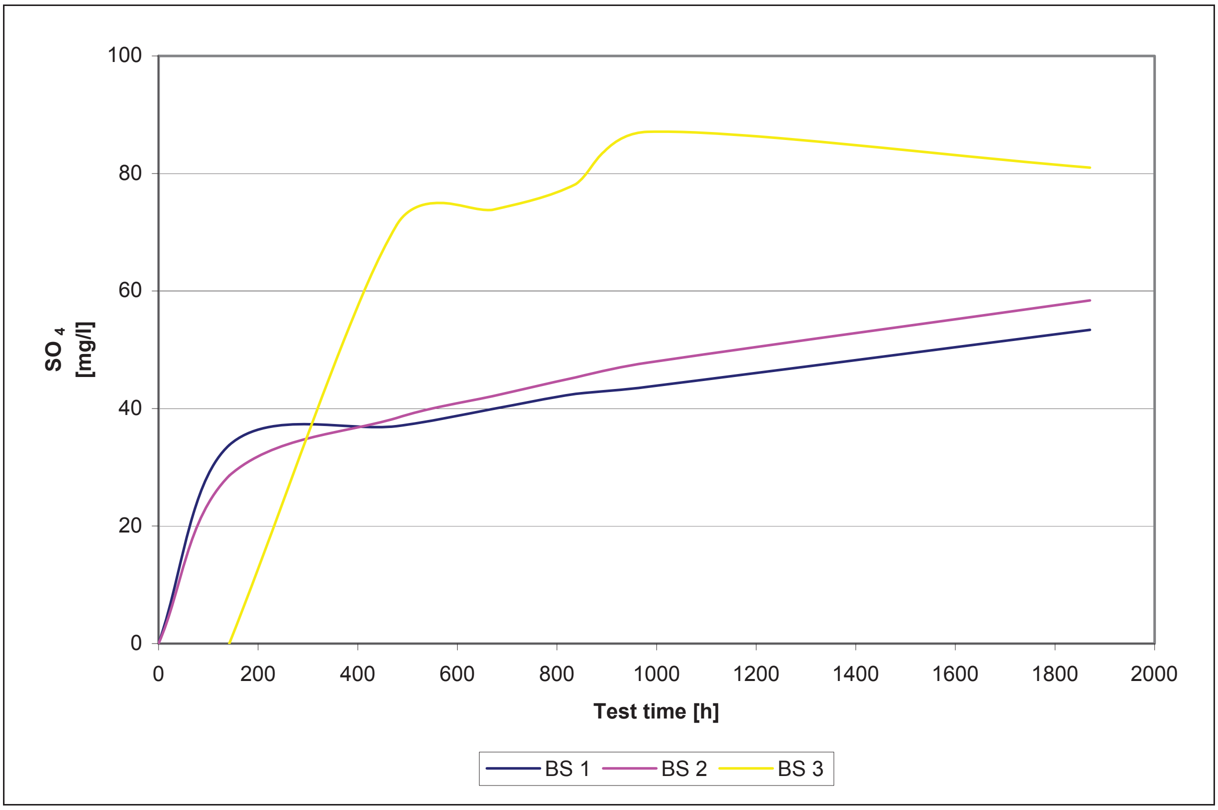

Figure 7-27: Interpolated SO_4 curves of the LT 2 samples with a test time of 1870 hrs.

7.5 Open pit mine water quality

The data and diagrams presented in this chapter were made available by Rabenwald open pit mining. The quality of the open pit mine water is measured in Rabenwald at the following 17 locations: Quelle Heil, NMW Hochbehälter, Schlagbauerquelle, retention basin “Wasserspiegel Brunnen”, Brunnen Krughof, Lilly Stollen, Lilly Quelle, Almer, Hummelbrunnen, Ernst August Stollen, Floingbach Sailer, Floingbach Brücke, Quelle Rossegger, Rohrquelle Feldhofer, Tagbau 1070, Tagbau Sickerwasser TAR, Pumpensumpf. The position of the locations can be seen in figure 7-28.

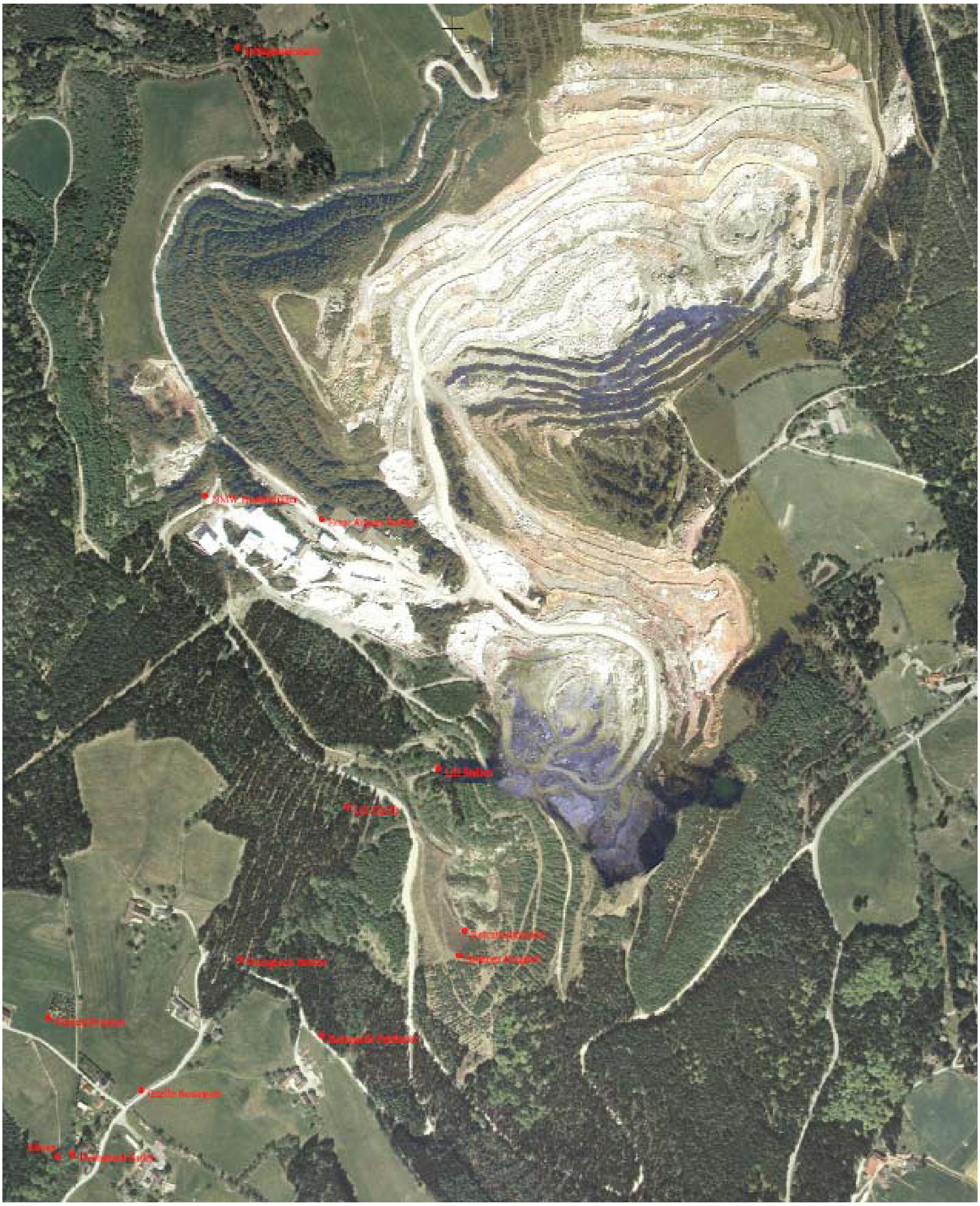

Figure 7-28: Location of the water control points at Rabenwald open pit mining.
Quelle Heil is beyond the frame.

Particular attention must be given to the water quality at the Ernst August Stollen and the Lilly Stollen, as practically the entire open pit mining is drained through these systems (see chapter GEOLOGY AND HYDROGEOLOGY). Also important here is the data from the retention basin "Wasserspiegel Brunnen", into which the Ernst

August Stollen drains, as well as the measurement points of the Floingbach, into which the waters of the Lilly Stollen flow. Of the parameters measured in Rabenwald, particularly the SO_4 and pH values of the open pit mine waters are of significance due to the scope of this thesis.

Between 2000 and 2004 the waters of the Ernst August Stollen showed a fivefold increase of the SO_4 values from 159 mg/l to 1007.8 mg/l (figure 7-29). The upper limit of 250 mg/l SO_4 in accordance with the regulatory guidelines for drinking water was first exceeded in 2001 and has been above the limit ever since (see APPENDIX). From 2004 onwards the waters of the Ernst August Stollen witnessed a decelerated increase of the SO_4 values.

Simultaneously between 2001 and 2004 the pH values fell from 8.1 to 6.4 (figure 7-30). Following this period, the values developed a more consistent range, fluctuating between 6.5 and 7 from 2004 to 2007, although in mid-2007 the range increased to higher values, from 7 to 7.8.

When directly comparing the results for the SO_4 and pH levels, a partially inverse trend is recognisable, meaning that the pH level falls when the SO_4 value rises and vice versa. This can be observed, for example, between 2001 and the beginning of 2006, and from mid-2007 to the end of 2008. All results from measurements of the open pit mine waters at the Ernst August Stollen can be found in the APPENDIX.

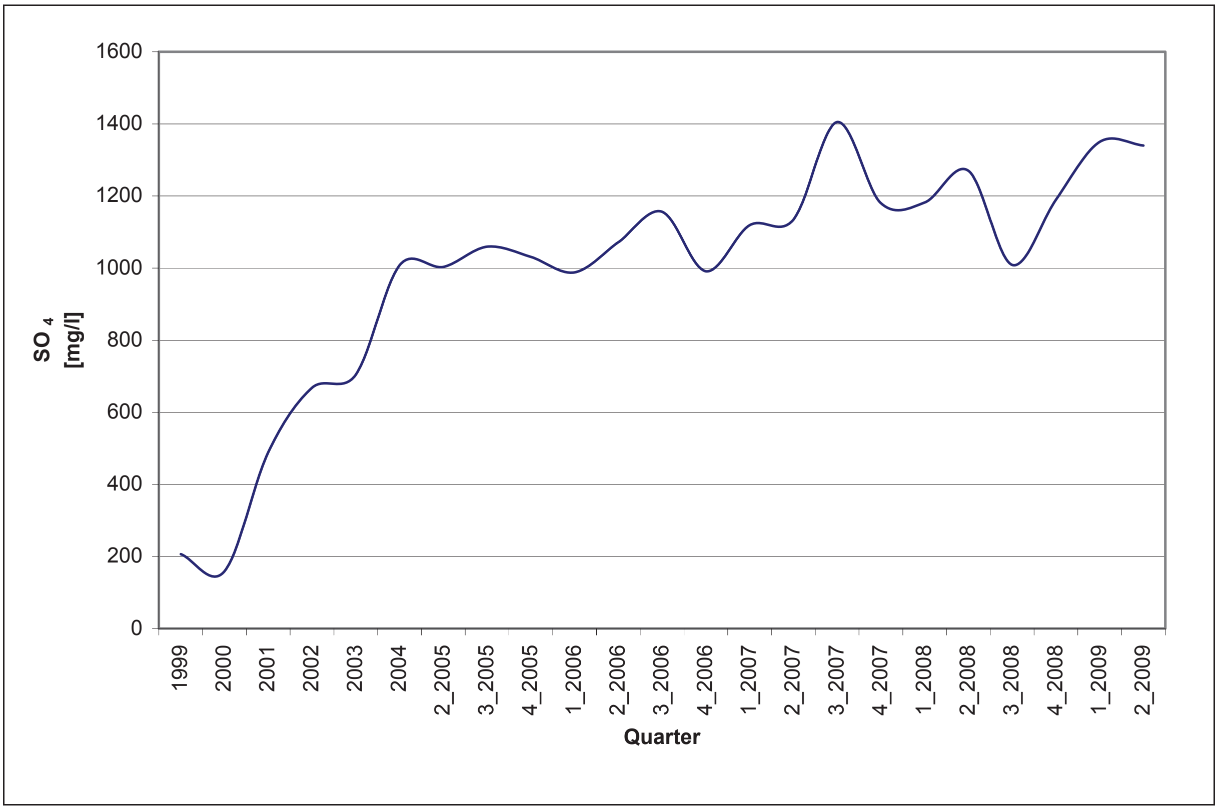

Figure 7-29: SO_4 curve of the waters at the Ernst August Stollen between 1999 and 2009.

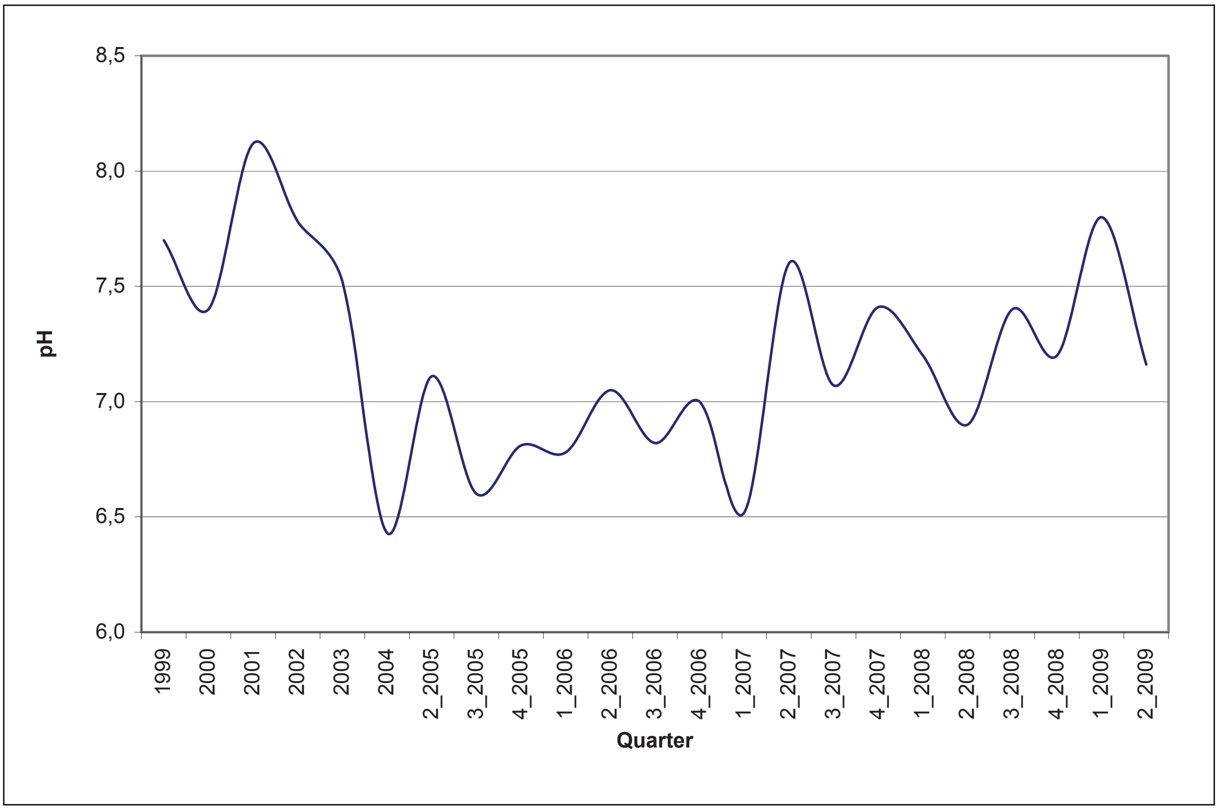

Figure 7-30: pH curve of the waters at the Ernst August Stollen between 1999 and 2009.

The retention basin “Wasserspiegel Brunnen”, into which the Ernst August Stollen drains via a channel, shows an increase in the SO_4 value between 2000 and 2005 of over 800%, from 112.9 mg/l (2000) to 967.3 mg/l (2005). As at the Ernst August Stollen, the SO_4 upper limit was first exceeded in 2001 (figure 7-31). In the following 17 measurements up until mid-2009, only twice the results were below the limit (2_2008 and 1_2009). The high results recorded in 2004 and 2005, which were almost four times the allowed value, were not reached again, yet the values did remain high.

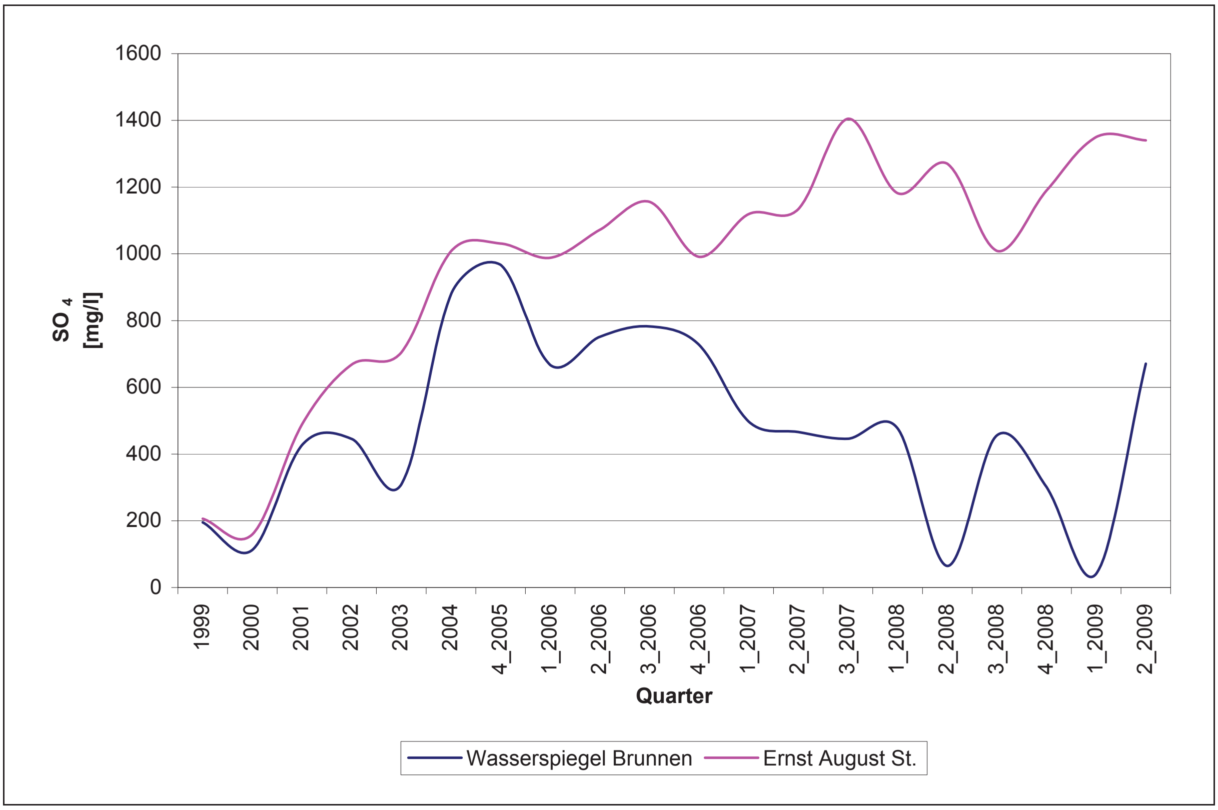

Figure 7-31: SO_4 curves of the waters at the Ernst August Stollen and the retention basin between 1999 and 2009.

In figure 7-31 one can see that the course of the SO_4 curves for the open pit mine waters at the Ernst August Stollen and at the retention basin Wasserspiegel Brunnen from 1999 until mid-2006 are very similar. Furthermore, a tendency of recently increased values at both of the measuring points is identifiable. For most of the observation period, the waters at the retention basin had notably lower SO_4 concentrations.

The pH values measured at the retention basin are inhomogeneous and therefore not particularly meaningful. Over the entire period of measurement from 1999 to mid-2009 they fluctuated between a pH of 7.2 and 8.3. The only trend that can be defined is seen with an increase of the pH level from the end of 2006 to mid-2008 when the maximum value was reached.

The pH curves of the waters at the Ernst August Stollen and the retention basin also highlight similarities (see figure 7-32). The correlation of the results is particularly clearly seen from 1999 to 2002 and from mid-2008. In contrast to the SO_4 curves, here the Ernst August Stollen produces lower pH values. Therefore a direct connection between SO_4 concentrations and the pH value can be drawn.

The measurement results of the retention basin Wasserspiegel Brunnen are included in the APPENDIX

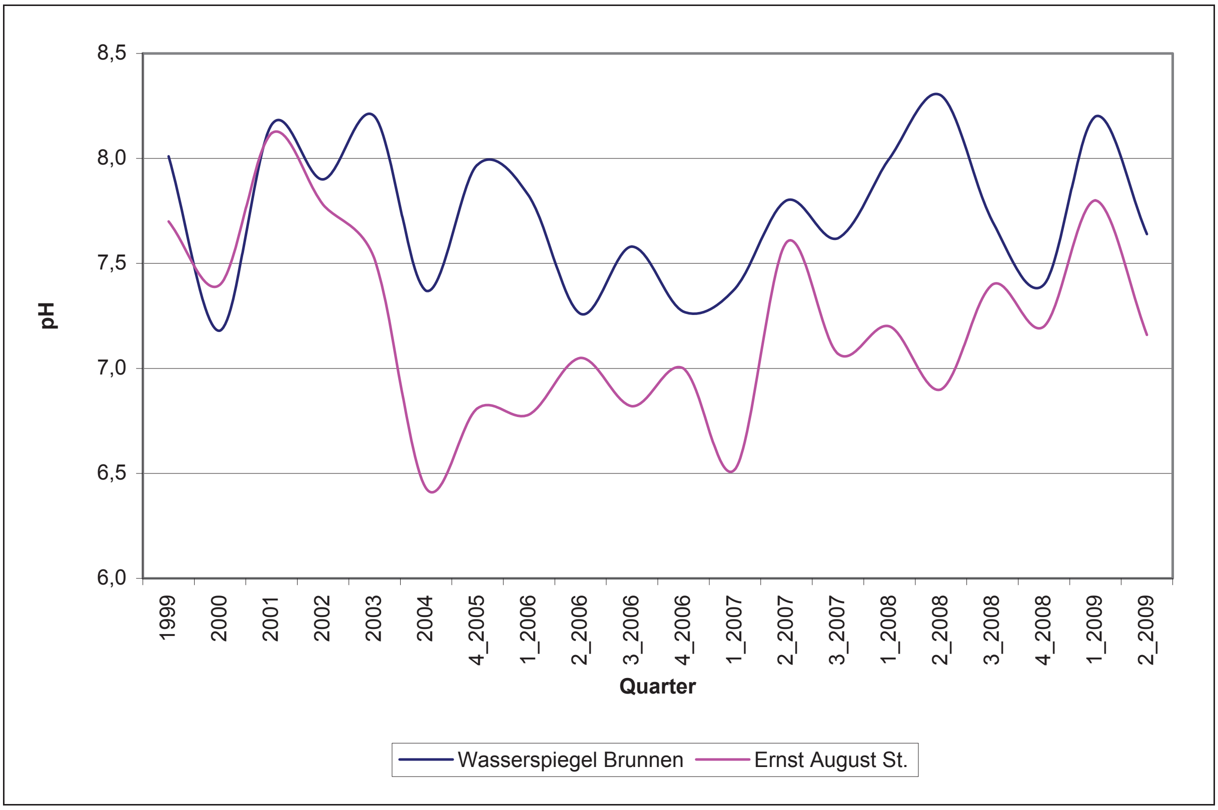

Figure 7-32: pH curves of the waters at the Ernst August Stollen and the retention basin between 1999 and 2009.

The open pit mine waters at the Lilly Stollen (figure 7-33) display, with few exceptions, a constant increase in SO_4 values between 2000 (70.5 mg/l SO_4) and the end of 2007 (1003 mg/l SO_4). Particularly striking is the change between the third and fourth quarters of 2007, during which the value practically doubled. The high value at the end of 2007, however, represents a one-off outlier in this range. The permitted SO_4 limit was first exceeded in 2004, and with the exception of two interruptions (2_2005, 2_2008) the value remained above the limit until 2009, in part by a significant margin. From 2008 the measured values remain at a high level, yet significant fluctuations in the values are clearly visible. A clear exception to the high levels is the result from the measurement in the second quarter of 2008 (112 mg/l), which dropped below the allowed limit for the first time since 2005. In 2009 an increase in the measured values is apparent, which continues to today (Oct. 2009).

The pH values of the waters at the Lilly Stollen (figure 7-34) constantly fluctuated in the period 2000 to mid-2005 between 7.9 and 8.3. From mid-2005, however, the pH values fell significantly, which correlates well to the strong increase of the SO_4 concentrations at that time. From this point onwards, the pH level remained below a value of 7.5 with only four exceptions.

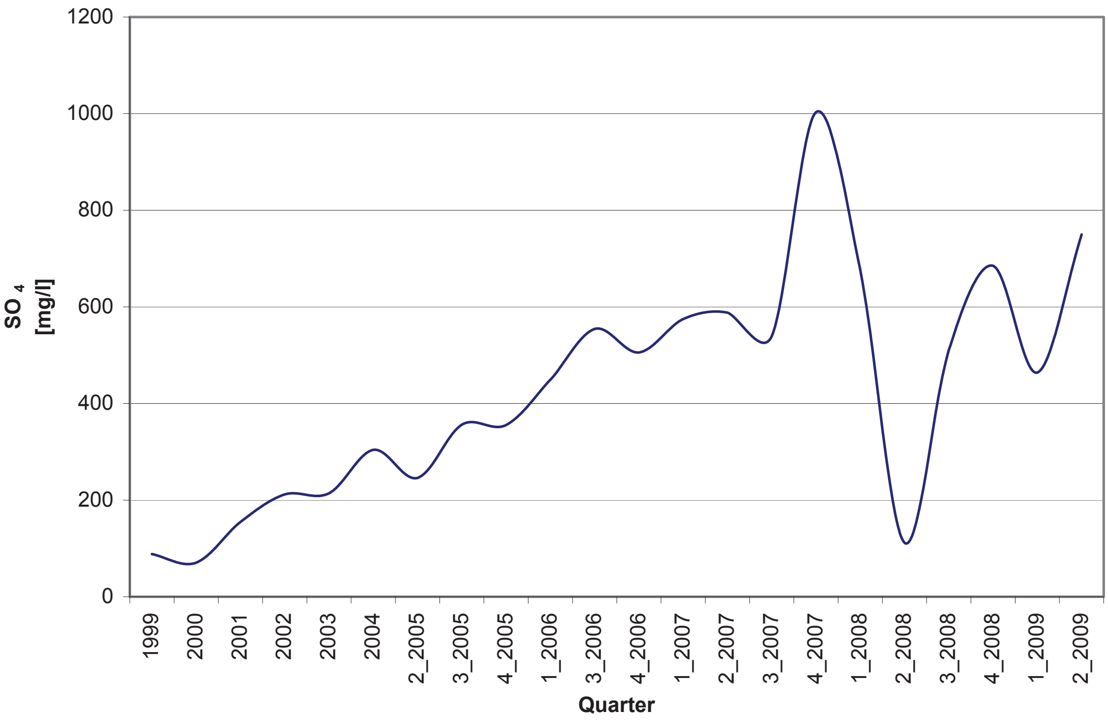

Figure 7-33: SO_4 curve of the waters at the Lilly Stollen between 1999 and 2009.

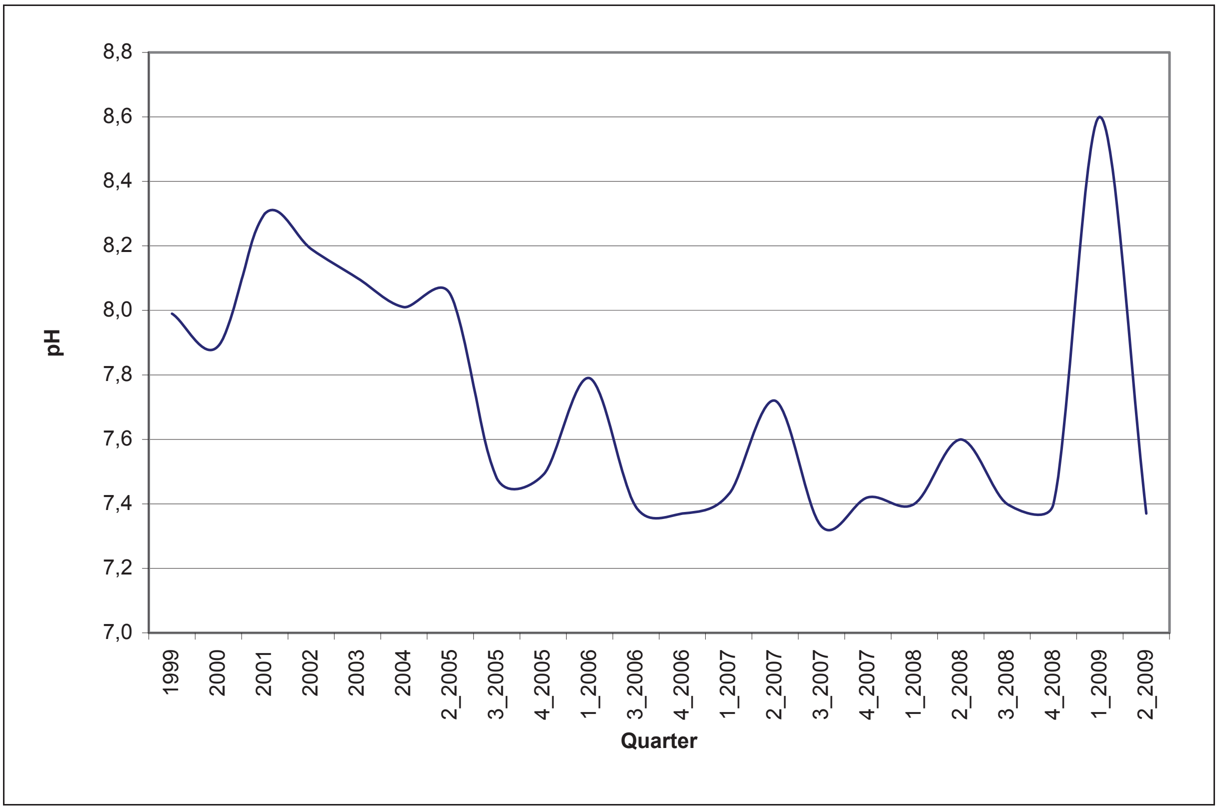

Figure 7-34: pH curve of the waters at the Lilly Stollen between 1999 and 2009.

The Lilly Stollen drains entirely into the Floingbach. Figure 7-35 and figure 7-36 show the SO_4 and pH curves of the two measurement points of the Floingbach (Floingbach Brücke, Floingbach Sailer) as well as those of the Lilly Stollen in direct comparison.

The SO_4 curves of the three measurement points run only partially concordantly, for example in 2005, 2006 and 2009. The SO_4 values from the Floingbach additionally show good correlation between 2007 and mid-2008. The first measurements in 2005 resulted in values between 246 mg/l (Lilly Stollen) and 271.1 mg/l (Floingbach Sailer). All in all, an increase in the values at all of the measurement points from 2005 to 2007, followed by irregular fluctuations in the values until the end of 2008 and then a unified increase can be seen. The last measurements in the second quarter of 2009 yielded SO_4 values between 614 mg/l (Floingbach Sailer) and 750 mg/l (Lilly Stollen). The results of all three measurement points also regularly exceed the SO_4 limit of 250 mg/l. Of a total of 44 measurements at the three points, only six results, just over 13.5%, fall below this level. Also noticeable is that in only five of the fifteen measurement rounds did the SO_4 results at the three points differ by a margin of more than 140 mg/l SO_4.

The pH curves of the waters at the Floingbach follow to a large extent a very similar path over the course of the observation period. At times, the waters at the Lilly Stollen also show similar tendencies in terms of the values over this period. The initial pH values (in 2005) measured at the three locations ranged between 8.1 (Lilly Stollen) and 8.3 (Floingbach Sailer). In principle, one can see a trend of falling pH values from 2005 to mid-2008 despite substantial fluctuations. Following the sharp increase of the pH level in 2008, the drops seen in both of 2009's measurements are clearly visible. The pH values recorded in the second quarter of 2009 range between 7.37 (Lilly Stollen) and 7.72 (Floingbach Brücke).

All results of the SO_4 and pH measurements at the three points can be seen in the APPENDIX.

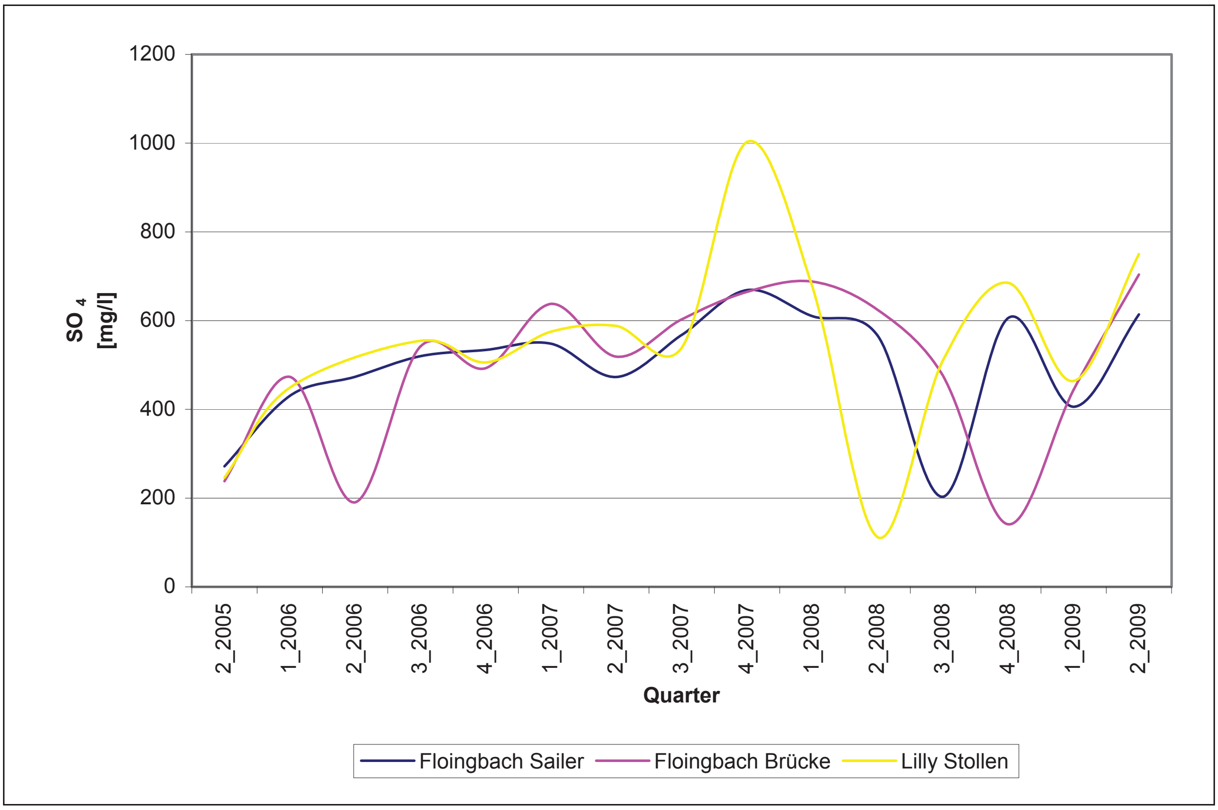

Figure 7-35: SO_4 curves of the waters at the measurement points Floingbach Sailer, Floingbach Brücke and Lilly Stollen between 2005 and 2009.

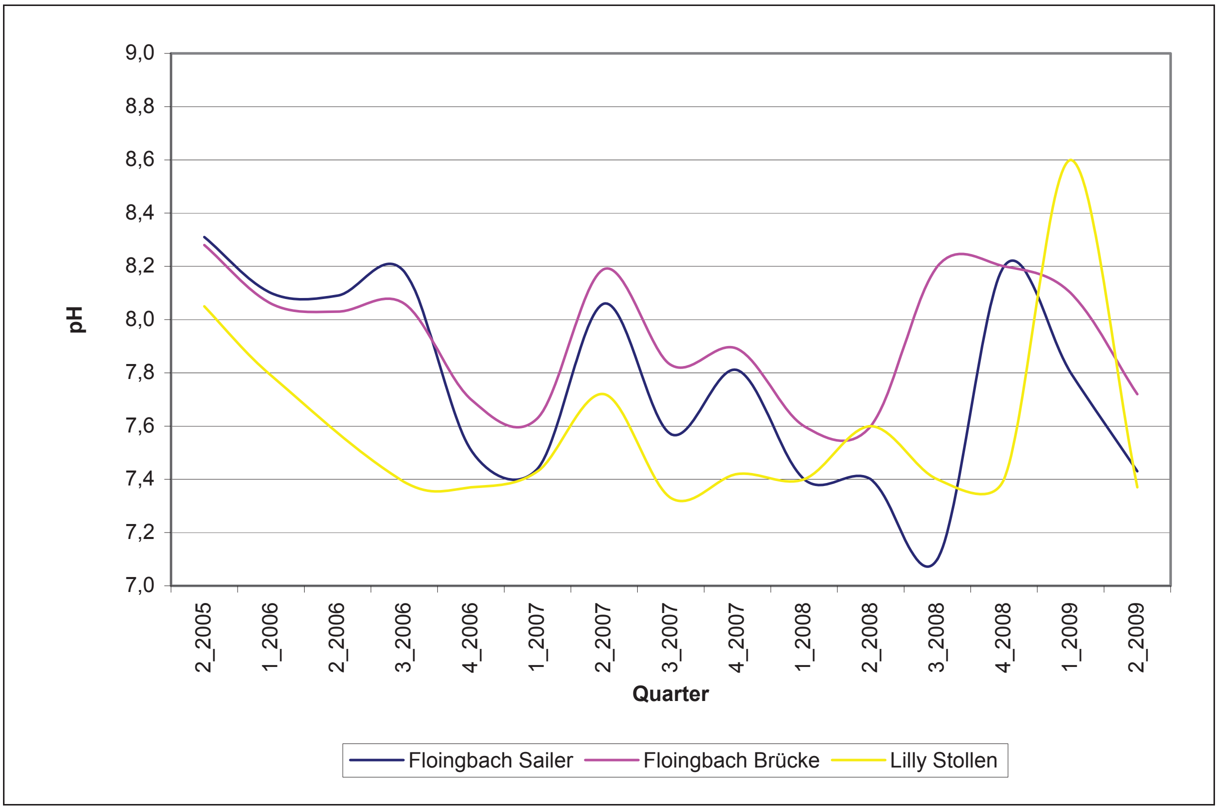

Figure 7-36: pH curves of the waters at the measurement points Floingbach Sailer, Floingbach Brücke and Lilly Stollen between 2005 and 2009.

Of the total of 17 water measurement locations in Rabenwald, 13 exceeded the SO_4 limit at least once. The locations where this happened are as follows (number of measurements above limit shown in brackets): Ernst August Stollen (21), retention basin “Wasserspiegel Brunnen” (16), Lilly Stollen (15), Floingbach Brücke (14), Floingbach Sailer (14), Rohrquelle Feldhofer (13), Hummelbrunnen (12), Brunnen Krughof (6), Tagbau Sickerwasser TAR (2), Schlagbauerquelle (1), Quelle Rossegger (1), Tagbau 1070 (1), Pumpensumpf (1).

From 2000/2001 a significant increase in SO_4 concentrations was measured at all of the locations (figure 7-37). The strong increase in the values between 2000 and 2004 was somewhat diminished by later results, yet the trend of increasing values applies until the last measurement in the second quarter of 2009. As early as 2001, the permitted level of SO_4 had already been exceeded at six of the measurement points. Over the entire duration of observation the waters at the measurement points Ernst August Stollen, Wasserspiegel Brunnen, Lilly Stollen, Floingbach Brücke, Floingbach Sailer, Hummelbrunnen, Brunnen Krughof and Rohrquelle Feldhofer in particular showed significantly increased concentrations. SO_4 peak values were reached in the individual curves between mid-2007 and the end of the year, as well as in 2009. Nevertheless, results in 2008 showed a decrease at many of the locations. As mentioned above, the highest values by far were recorded in the waters at the Ernst August Stollen.

In contrast to the SO_4 values, the pH values of the measured open pit mine waters generally remained within the permitted pH range of 6.5 to 9.5 (figure 7-38). All in all, pH levels below 6.5 were measured at the following locations (number of measurements below limit shown in brackets): Lilly Quelle (19), Almer (13), Hummelbrunnen (13), Quelle Heil (6), Ernst August Stollen (1), Schlagbauerquelle (1). The pH values of practically all of the measurement points are irregular and fluctuate significantly. A general increase in the pH levels between 2000 and 2001 can be seen, which was then followed by a trend of falling values until the start of 2007. At this point the results were inconsistent, generally leading to large fluctuations from quarter to quarter. The results of the last measurements saw the pH values come together, ranging between 6.93 (Hummelbrunnen) and 7.99 (Brunnen Krughof). The course of the measurement result curves display good correlation with the formation of peaks particularly in 2001/2002 and 2007 (figure 7-38). The tendencies which can be drawn from the results of the last measurements taken in the second quarter of 2009 are clearly noticeable. The measurement points at the Ernst August Stollen and the Lilly Stollen, which drain the surface mining, as well as the measurement points at Wasserspiegel Brunnen and the Floingbach, into which these systems drain, showed significantly lower pH values. In contrast to this the measurement points at Hummelbrunnen, Quelle Heil, Lilly Quelle and Almer showed noticeably increased pH values.

The SO_4 and pH results from the waters of all 17 measurement points can be found in the APPENDIX.

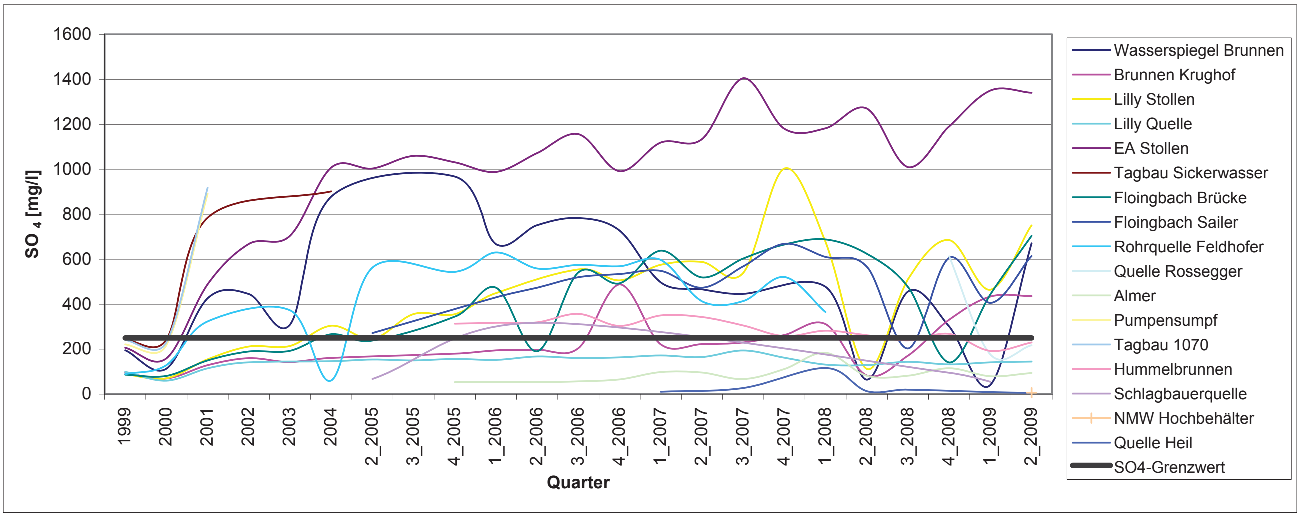

Figure 7-37: SO_4 curves of the open pit mine waters at Rabenwald between 1999 and 2009.

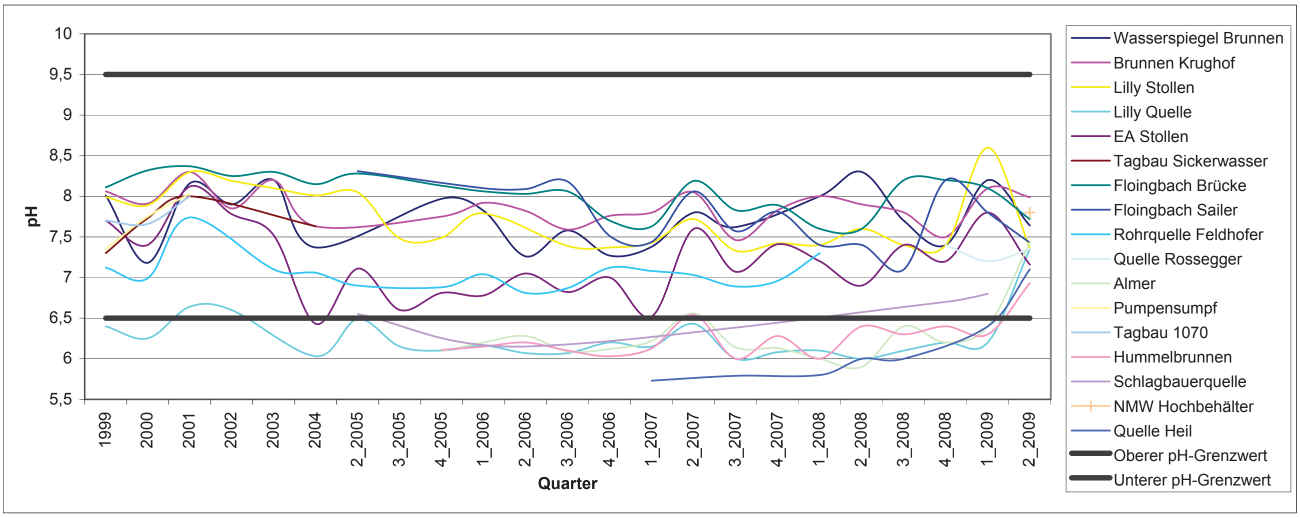

Figure 7-38: pH curves of the open pit mine waters at Rabenwald between 1999 and 2009.

8. DISCUSSION

The formation of acid rock drainage (ARD) is mainly described for coal mines and sulphide ore deposits in technical literature. Those deposits are characterised by high concentrations of sulphur and sulphide minerals. The leachate of those can cause large scale contamination and pollution of the surrounding environments. This is true for the pollution of soil by precipitation of heavy metals as well as for the decrease of water quality of receiving waters and ground water tables through acidity.

The rocks of the Rabenwald talc deposit show very low sulphur concentrations on average and hence at first sight no significant potential for the formation of ARD either. There are rocks with considerable amounts of sulphides, however, that could cause problems regarding the generation of acidic leachate. Furthermore the high SO_4 concentrations of the analysed open pit mine waters are an indicator of the possible formation of ARD. In this chapter, it should be clarified whether there is a danger by ARD for Rabenwald surface mining, what parameters play a role and thereby have to be considered, and which possibilities of preventive measures and countermeasures arise to prevent or minimise acid generation.

Of all examined rock types of Rabenwald surface mining only actinolite gneisses and calcsilicates show elevated sulphur concentrations. These rocks feature a potential for the formation of ARD by single consideration of this parameter. The classification of actinolite gneisses and calcsilicates per hand rock sample can lead to problems. Some samples can be allocated to both rock types when examined with the naked eye. Such uncertain samples can only be classified accurately by means of a microscope. Actinolite gneisses and calcsilicates offer similarities and constitute smooth transitions to some extent. The polished thin sections of the actinolite gneiss samples examined show - ordered by frequency in the rock - pyrrhotite, chalcopyrite, and pyrite as contained sulphides. Because of the similarities between actinolite gneisses and calcsilicates the occurrence of the same sulphide minerals for the latter is assumed. Pyrrhotite appears with big xenomorphic grains as well as in the shape of veins following the fault zones of the rocks in the polished thin sections. Furthermore, affected, decomposed pyrrhotite constitutes a ‘network’ of finely branched veins in the rock. Chalcopyrite constitutes xenomorphic aggregates with distinct borders to the basic matrix in the polished thin sections. No affected grains or decomposition phenomenons can be identified. In comparison to the other sulphides pyrite tends to better crystallinity and occurs in the shape of xenomorphic to hypidiomorphic grains only subordinate in the rock. Next to the borders of the aggregates to the basic matrix scattered affected areas can be observed.

The type of sulphide minerals appearing has great influence on the formation of ARD because their oxidation runs at different rates. Pyrrhotite tends towards rapid oxidation; the reaction of crystalline pyrite takes a longer period. These differences are reflected in magnitude and point of acid generation. The form of the sulphides in the rock is important as well. Grains that are well crystallised have smaller exposed surface areas and hence less reaction potential than disseminated mineral grains.

Due to the great abundance of pyrrhotite compared to chalcopyrite and pyrite in the samples it can already be assumed that this one must be seen as the main cause of acid generation at the Rabenwald deposit. The mass rate of produced acid is directly related to the exposed surface area of sulphide minerals (CARUCCIO and GEIDEL, 1981). The observed decomposition phenomena of the aggregates and veins of pyrrhotite are an indicator of the oxidation process already begun with decomposition of the pyrrhotite and potential acid generation. The occurrence of the mineral in shape of ‘networks’ of thin veins with diffuse borders additionally accelerates this reaction.

Kinetic tests like the conducted leaching tests provide information on the rate of sulphide oxidation and hence acid generation over longer periods and on the quality of the produced solutions. They therefore provide data about timing and rate of acid generation and acid consumption respectively. For the interpretation of data regarding the generation of acid the interaction of acidity, loss of alkalinity, pH, and sulphate is important. A change in the pH value of the basis fertiliser solution is an indication of the predominance of either the process of acid generation or acid consumption at this moment. During acid generation and acidic conditions that accompany that, the pH value drops; during acid consumption by means of alkalinity, the pH value increases. Sulphate is a product of the oxidation process. An increase in sulphate concentrations in the solutions is evidence of the activity of the reaction. These two starting points represent the basis for conclusions about the ARD potential of the examined rocks. The results of the leaching tests and of the ion chromatography are discussed below.

The first increase in the pH values of the waste rock samples of the first leaching test (LT 1) up to neutral and basic areas is a distinct signal of the predominance of acid buffering to acid generation in the solutions. Carbonate minerals contained in the rocks tend towards quick neutralisation of produced acid. Furthermore the rocks of the Rabenwald deposit show considerable amounts of silicates and alumosilicates. According to WILLIAMS et al. (1982) minerals like mica, pyroxene, feldspar and amphibole show a tendency towards alkaline pH values and can neutralise produced acid when the oxidation process runs slowly. This process is not as effective as the neutralisation process of carbonates, but is not to be underestimated because of the huge amount of silicates and alumosilicates in the rocks. Thus the high pH values of the first increase are probably the result of reactions of the generated acid with calcite contained in marble and calcsilicates and with silicate minerals. This causes a release of alkalinity which has the potential to neutralise acid in situ. There is enough alkalinity in the samples to neutralise even the basis fertiliser solutions with an initial pH of 3.5. This effect can be confirmed by comparison of the Ca^{2+} concentrations of the solutions. Calcium ions are a product of the neutralisation reaction of calcite. The samples with acidulated basis fertiliser solutions exhibit noticeably higher concentrations of Ca^{2+} ions than the samples with solutions of distilled water (see APPENDIX). This is an indication of the activity of the neutralisation reaction in the samples with acidulated basis fertiliser solutions to a major extent. Moreover this can be seen as an argument for high alkalinity and acid buffering capacity of the samples to such an extent that even significant acid solutions can be neutralised.

The following drop of the pH values of all solutions cannot be explained absolutely certainly. In principle there are several possibilities. The first possible explanation is a distinct loss of the alkalinity contained in the samples. This is consequently accompanied by an attenuation of the neutralisation reaction. The second conceivable scenario is the initiation of an additional oxidation process of a sulphide mineral. The slow diffusion of oxygen in water has to be considered as well. Since the leaching tests were carried out under water saturated conditions the lack of oxygen can lead to detraction or deceleration of the oxidation process. The pH values of the samples fall the relatively small amount of maximal ca. -0.4 (sample 2). In this period the process of acid generation definitely becomes predominant compared to the neutralisation of acid. This circumstance isn’t supported by the measured ion concentrations however. The concentrations in per cent of calcium increase to a higher amount than those of sulphate. Hence a different course of the pH curves would be expected. Due to its tendency towards rapid oxidation, pyrrhotite will be responsible for the first stage of acid generation. By reason that already the first ion chromatography measurement reveals significant sulphide concentrations the assumption seems to be likely that a loss of alkalinity produced by calcite minerals happened and consequently the pH values dropped. In addition to that a delayed onset or an attenuation of the oxidation process, because of the deficit of oxygen or the initiating oxidation of chalcopyrite or pyrite, can be jointly responsible for the decay of the pH curves.

The subsequent rise in the pH values of all solutions - except for sample 1 - is the result of further neutralisation of produced acidity. Because of the great amount of alumosilicates and silicates in the rock samples these minerals can consume acid over longer time periods. The second increase in the pH values of the samples 2 to 6 is probably bound to an interaction between the neutralisation reactions of calcite, alumosilicates and silicates. The solution of the fine-grained sample 1 doesn’t show a second rise of the pH, yet a flattening of the curve. The produced acid isn’t neutralised here but rather the process is attenuated.

The second drop in the pH values starting at a test duration of about 220 hrs is more distinct for the fine-grained samples; the pH values of the coarse grained samples only show little changes. Pyrite with its well crystallised aggregates when compared to pyrrhotite tends towards slower oxidation on the one hand, but is a better acid producer on the other. By means of milling the samples 1 and 4 the exposed pyrite minerals can react better and hence lead to further oxidation and decreasing pH. This assumption is confirmed by the continuing increase of the sulphate concentrations of samples 1 and 4 in contrast to the other samples.

The nearly constant pH values from a test time of 720 hrs are an indicator of the adjustment of equilibrium between acid generation and acid consumption. The further increasing sulphate concentrations mainly of the samples 1 and 4 are a clear hint of the continuing activity of the oxidation process though no change of pH is occurring anymore.

Regarding the significance of the waste rock sample and the LT 1 it has to be mentioned however that it is very difficult to carry out a reliable and representative sampling of a waste rock management unit. As the waste rocks are brought by trucks to the dump, the composition of the waste rock management units is heterogeneous and variable dependent on the abolished rock type, the location of deposition, the distribution of the minerals, the grain sizes and the hydrologic conditions. Hence different microenvironments with varying characteristics can develop. The investigated waste rock sample represents only one of various possible compositions and therefore cannot be taken as a basis for an estimation of the ARD potential of the total waste rock management unit. The composition of the waste rock sample (see chapter ANALYTICAL METHODS) features disproportionately high proportions of calcsilicates, actinolite gneisses and paragneisses. Above all the high content of calcsilicates leads to high pH values and thus to an overestimation of the neutralisation potential of the samples. The ratio between mass of rock sample to basis fertiliser solution represents another problem of the samples 3 and 6. These samples contain twice the amount of rock material but the same amount of basis fertiliser solution than the other four samples. The ion concentrations can easily be correlated by division; the significance of the measured pH values is questionable indeed.

The samples of the second leaching test (LT 2) permit a prediction of the ARD potential of the entire Rabenwald deposit. Besides the pH curves show similarities with those of the samples of LT 1 described above. The comparison of the pH values of the basis fertiliser solutions of BS 1 (bulk sample 1) and BS 2 (bulk sample 2) is of particular interest, because these samples demonstrate the differing effects of acid consumption by marble and calcsilicates.

The sample BS 1, which contains no marble, shows a slow increase in the pH value. This is based on the lower calcite concentrations of calcsilicates compared to marble. In addition the better encapsulation of calcite by other minerals in calcsilicates may play a role. Thus less mineral surface for neutralisation reactions is available. Particularly interesting is the fact that this slow release of alkalinity is keeping the pH values at a constant level although acid generation is initiated. The pH value only drops twice temporary. This is probably caused by the different points of initiation of the oxidation processes of pyrrhotite, chalcopyrite and pyrite. As already discussed, the following trend of higher pH values is most likely determined by reactions of calcite, alumosilicates and other silicates.

The intense first increase in the pH of sample BS 2 is caused by reactions of calcite minerals in marble and calcsilicates secondarily. Furthermore the impact of acid consuming reactions of silicates may not be underestimated. The almost simultaneous release of great amounts of alkalinity by the quick reaction of mainly calcite leads to high initial pH values. The adjacent progress of the pH curve is most likely controlled by acid consuming processes of calcite, alumosilicates and other silicates and acid generating reactions by pyrite similar to the pH curves of the waste rock samples.

The ion concentrations of calcium and sulphate of both solutions (BS 1, BS 2) are not affected by the different compositions and pH values. Both samples show nearly equal curve trends of both ions during the whole test time; furthermore the concentrations of both ions increase until the end of the tests. Sulphate is a product of the ARD process. The equal development of the sulphate curves of samples BS 1 and BS 2 provides the indication that in both samples the oxidation process runs to the same extent. Acid, which is produced in the process, is neutralised depending on the different buffering capacities of the two samples. The differences in neutralisation capacity are reflected in differing pH values. Thus oxidation reactions take place in the sample solutions; the produced acid is neutralised in situ; sulphate as a product of the oxidation of sulphides remains. Moreover it can be stated that the addition of carbonates influences the pH value and acid consumption; the activity of the oxidation process with production of sulphate is not affected however.

The high increase in the pH value of sample BS 3 (bulk sample 3) at the beginning of the test is caused by calcite minerals similar to the other samples. The large exposed surface areas of the minerals due to the milled grains reinforce this effect. The following drop in the pH curve is probably based on the oxidation of pyrrhotite and chalcopyrite. The further course of the pH curve is nearly constant. The acid generating processes (reaction with pyrrhotite and pyrite) and the acid consuming reactions (reaction with calcite, alumosilicates and other silicates) that influence the other samples only play a subordinate role for BS 3. The fineness of the sample favours a quick progress of the reactions. This assumption is confirmed by the trend of the calcium and sulphate curves. Those stay nearly equal from a test time of ca. 500 hrs. Together with the constant pH value this fact indicates the completion of the oxidation processes of pyrrhotite, chalcopyrite and pyrite.

No calcite is contained in actinolite gneisses. The low increase in the pH value right at the beginning of the third leaching test (LT 3) is therefore due to silicates. From a test time of ca. 100 hrs the acid produced by oxidation processes cannot be neutralised anymore. Consequently the pH value drops. The values even out at a pH level between 5.5 and 5.6. Further processes as seen with the other samples cannot be determined here by means of the pH curve. The tendency towards falling pH values from a test time of ca. 2000 hrs is an indicator of the ARD process and accompanied acid generation still running at that time. Thus a complete termination of acid production cannot be assumed for this test.

It has to be mentioned restrictively that the results of LT 3 have to be seen as a worst case scenario for the Rabenwald surface mining. In reality other existing rocks will attenuate the acid generation of actinolite gneiss.

The aim of the fourth leaching test (LT 4) was to demonstrate to what extent pH values of samples from sulphide deposits vary with time. The copper schist from Poland as well as the sample from Pine Point, Australia show explicit basic pH values during the whole test time. Thus the samples offer enough alkalinity to buffer the produced acid. The sample from Waldenstein, Carinthia shows considerable acid generation and subsequent low pH values. This sample has a significant potential for the formation of ARD. The first measurement results after a test time of 4 hrs are of particular interest. Even after this short time period on the one hand a release of alkalinity (PP, CuS) and on the other hand acid generation (Wa) occurred and the pH of all samples changed significantly. This is an indication of the very quick initiation and progression of the oxidation processes and the neutralisation reactions of the ARD process.

According to the B.C. AMD TASK FORCE (1989) samples from leaching tests can be classified into three groups due to their pH values. Samples with pH values <3 can be seen as strong acid generators; with pH values between 3 and 5 acid is generated but there is a certain neutralisation capacity; samples with pH values >5 have either no potential for acid generation or enough alkalinity to buffer the produced acid in situ. According to this classification the rocks of the Rabenwald can be divided into the following two groups:

- Rock type with uncertain potential for the formation of ARD: actinolite gneiss.

- Rock types without potential for the formation of ARD: talc, leucophyllite, marble, granite gneiss, paragneiss, hornblende gneiss, calcsilicates.

Because of the measured sulphur concentrations only actinolite gneisses and calcsilicates have a potential for the formation of ARD in theory. Calcsilicates have considerable amounts of calcite however and therefore enough alkalinity to buffer the produced acid. The leaching test of the actinolite gneiss produced no pH values <5. Yet the pH curve tends towards falling values at the end of the test. A drop in the pH to values <5 cannot be excluded definitely. Even for that case the actinolite gneiss has only low potential for the formation of ARD according to the classification of the B.C. AMD TASK FORCE (1989) since there is a certain neutralisation capacity counteracting the acid generation.

It has to be mentioned however that sulphide minerals contained in the rocks are oxidised to some extent producing acid and sulphate. The continuous increase of sulphate concentrations support the assumption that the oxidation process is running until the end of the test. This instance indicates that there is no danger in terms of acidic solutions but instead in terms of higher sulphate concentrations for the Rabenwald open pit mine waters.

The minerals pyrrhotite, chalcopyrite and pyrite are responsible for acid generation in the solutions. Pyrrhotite is of particular importance as it is most probably the main causer of acid generation at Rabenwald deposit because of its great abundance in the rocks. The mineral probably causes the first decrease in the pH values because of its tendency towards quick oxidation. The influence of chalcopyrite on acid generation is unclear. The mineral is most probably jointly responsible for the first drop in the pH as well. Pyrite features better crystallinity in the samples and has an affinity to later oxidation on the one hand, but can generate more acid than other sulphide minerals on the other hand. So the influence of the pyrite becomes supposedly evident with the second, smooth drop in the pH curves.

The neutralisation capacity of the rocks of Rabenwald surface mining is mainly determined by calcite contained in marble and calcsilicates. In addition to that, alumosilicates and other silicates like mica, feldspar, amphibole and pyroxene have to be considered because of their huge quantities in the rocks. These minerals tend to produce an alkaline pH and have the capacity to neutralise acid. Calcite in marble offers very high neutralisation potential and is therefore primarily responsible for the quick and high increase in the pH at the beginning of the tests. Calcsilicates and silicate minerals have lower neutralisation potential on the one hand, but tend towards more constant and long-term consumption of acid on the other hand. The differences in the neutralisation capacities of the rocks can be seen in the leaching tests of the samples BS 1, BS 2, and AG. BS 1 comprises calcsilicates and silicate minerals; BS 2 contains 2 weight per cent marble in addition; the neutralisation capacity of AG is limited to silicate minerals.

The results of LT 3 and LT 4 argue for a very quick progress of acid generation and acid consumption respectively. The pH values of the Rabenwald samples increase to neutral and alkaline values immediately after the beginning of the tests. In the initial phase of the tests the release of alkalinity and the process of acid consumption proceed to a bigger extent than acid generation which seemingly starts somewhat delayed.

The trend of the pH curves is an indicator of the predominance of acid generation and acid consumption in the solutions respectively. The first rise is mainly due to marble, which reacts quickly. The oxidation reactions of pyrrhotite and chalcopyrite start at the same time or perhaps a little time-delayed. The samples offer enough alkalinity to buffer the generated acid and to keep the pH at a constant level over a certain time period. Acid generation caused by pyrrhotite and chalcopyrite become predominant afterwards; the alkalinity of marble is consumed. The acid generation potential of the rocks is too low to push the pH to acidic areas however. Further neutralisation reactions induced by calcsilicates and silicate minerals and the beginning of the oxidation of pyrite subsequently balance each other out and lead to a constant pH. The pH values of all samples except for the actinolite gneiss even out at alkaline values at the end of the tests and hence show no potential for acid generation. The approach of the pH values of all samples of LT 1 and the samples BS 1 and BS 2 of LT 2 at alkaline levels even though their different compositions and basis fertiliser solutions is of particular interest. This fact argues for minor influence of sulphide minerals and thus low potential for acid generation and formation of ARD for the rocks of Rabenwald open pit mining. A theory according to WILLIAMS et al. (1982) supports this assumption. WILLIAMS et al. (1982) relates the rate of initial acid production at a pH >4.5 directly to the quantity of acid that is produced after the complete run-off of the oxidation process. According to that a high initial rate of acid generation signifies a great amount of acid ultimately produced and vice versa. The rocks of the Rabenwald, with exception of actinolite gneisses, only show low initial acid generation rates and have therefore no significant potential for acid generation. In contrast to that the sample of actinolite gneiss generates considerable amounts of acid and hence an acidic pH is produced. According to that a low potential for the formation of ARD has to be assumed for actinolite gneisses.

The results of the leaching tests can only be allocated with reservations to the real conditions in the Rabenwald surface mining.

The problems regarding the representative sampling of a waste rock management unit because of the variable composition of the dump have already been discussed above. Not only the dump but also the rocks themselves vary concerning their mineral compositions. Thus the measurements of the sulphur concentrations of actinolite gneisses yield a variation limit between 0.026 and 2.801% S. As the quantities of the individual rock types are quite small in the samples, some minerals may be over-represented or under-represented and therefore lead to incorrect results.

The leaching tests in the laboratory were carried out with grain sizes <3 cm. Therefore larger areas of exposed mineral surfaces and hence better reaction potential are offered. Furthermore water saturated conditions were chosen for the leaching tests. This can lead to limitations of the oxidation processes due to the slow diffusion of oxygen in water and the following lack of oxygen for the reaction. The rock material in the dump will be coarse grained on average in reality. This causes an encapsulation of minerals in unweathered rocks and thus their reaction potential is limited. In addition to that large areas of the dump will be located above the ground water table under unsaturated conditions and hence only be accessible for seepage water.

There is no change or mixture of the basis fertiliser solutions happening in the laboratory; the sample material is not modified during the whole test time as well and can consequently lose its reaction potential. Water will not stay in the same place for long time periods under real conditions but will flow at different velocities and hence consistently reach ‘fresh’ rock material for further reactions. ‘Fresh’ minerals for reactions can either be provided by deposition of waste rocks or by natural weathering and decomposition of big rock components.

The factor time can cause some uncertainties and problems regarding the interpretation of kinetic tests too. The measurements of course reflect the pH values at that specific time. The development of the pH in the partly very long periods between two measurements can only be interpolated. This leads to problems especially during the initial phases of the tests with rapid changes in the pH values regarding point in time and duration of acid generation and acid consumption. Above all, the duration of the laboratory tests is necessarily very short compared with the long-term potential for ARD formation of an active or closed deposit. For these reasons the obtained results from laboratory tests make a reliable estimation of the development of the potential for ARD formation over several years or decades nearly impossible.

The sulphate concentrations that are partly increasing until the end of the tests are a distinct indicator of the continued activity of the oxidation process even though the pH is not changing anymore. That is why only an estimation of the final potential for acid generation and formation of ARD is possible.

The fact that laboratory test only demonstrate a strong simplification of the natural conditions causes further problems. Thus it can happen that parameters, which have an important influence on the formation of ARD in the deposit, are not considered and are underestimated and therefore yield incorrect results and conclusions.

Sources of potential acid generation at Rabenwald open pit mine are present at pit walls and at dumps on the surface and subsurface because of previous underground mining. These possible sources are briefly described below.

Mining in open pits usually results in large areas of exposed rocks and a drawdown of the ground water table. Hence seepage water and oxygen that reach sulphides in pit walls or in joints can initiate the ARD process. Furthermore ‘fresh’ material for reaction with water and oxygen is consistently exposed by continuing mining activity. Most of the exposed rocks of Rabenwald surface mining have no potential for the formation of ARD however. Calcsilicates produce acid, yet this acid is neutralised in situ. Actinolite gneisses appear in the footwall of the ore body and in the shape of tectonically formed ‘Walzen’ within the vein in the Rabenwald open pit mine (WURM, 2005). So, to win talc, a continued exposure of ‘fresh’ actinolite gneiss is caused, whose sulphides can react with rainwater and oxygen. The generated acid is buffered by actinolite gneisses themselves and by other rocks in their surroundings. The seepage water flowing through the actinolite gneisses and the calcsilicates show elevated sulphur concentrations due to the activity of the oxidation process however.

There was an underground mining operating at Rabenwald in the past. Subsurface mining usually leads to a significant drawdown of the ground water table. Sulphide minerals that used to be located below the water surface before the mining operations started can become available for water and oxygen and therefore produce acid. Furthermore fracturing and jointing of the rocks will happen by underground mining operations. This leads to better flow paths for water and oxygen. The underground mining at Rabenwald was closed in 1990. As the rocks only have low sulphide concentrations it can be assumed that formation of ARD in galleries does not play an important role anymore.

Besides the pit walls the dumps are the second necessary source of potential ARD formation at Rabenwald surface mining. The physical properties of the dumps that influence the acid generation potential are grain size, permeability and weathering as well as the nature of deposition of acid generating and acid consuming rocks. The grain sizes of the deposited rock material is of particular interest as it defines the area of rock surfaces that are exposed for oxidation and weathering. The smaller the grains the larger are the exposed surfaces and the better is the reaction potential. In addition to that, the grain sizes of the particles define the permeability of the dump body. Waste rock management units often consist of coarse-grained rock material and offer therefore good flow paths for water and oxygen. It is possible that dumps with good permeability and coarse-grained rock material feature higher potential for the formation of ARD than those with fine-grained with larger reaction surfaces but low permeability. The weathering of the waste rocks mainly plays an important role regarding the prediction of the ARD potential over longer time periods. Weathering causes a reduction in grain sizes. This in turn leads to larger reaction surfaces of the particles, continued exposure of sulphides and carbonates minerals and changes concerning the permeability of the dump.

It cannot be resolved absolutely certainly which source is responsible for the maximum of acid generation at Rabenwald open pit mining. At pit walls ‘fresh’ material for reaction is exposed again and again. Moreover the conducted leaching tests indicate that maximal acid generation is happening already after short test time. The rocks may not be brought to the dump at that time yet and acid generation can happen with rocks in situ.
The rocks of the dump offer larger exposed surface areas which can react because of their higher grade of decomposition. Weathering results in smaller grain sizes and exposure of further ‘fresh’ sulphide minerals. As the rocks are located in the dump for decades and decades, and are completely decomposed with time, each existing sulphide mineral can react. Thus the oxidation process with production of acid and sulphates can run completely. In addition to this the rocks of the pit walls are jointed

but compared to the rocks of the dumps still unweathered, compact, and massy. So water and oxygen will find better flow paths in the pile.
There is probably a predominance of acid generation in the dumps at the Rabenwald talc deposit.

The water which is needed for the oxidation process is provided by precipitation and seepage at Rabenwald. The open pit mining is located in the humid climate area which is characterised by means of constant rainfall. Acid generation proceeds steadily in humid areas since constant water flow assures equilibrium between production and migration of acid. Due to the permanent availability of water no periodic cycles of accumulation and leachate of contaminants appear. Besides, the amount of sulphides in the rocks at Rabenwald is too low and thus the oxidation process to weak to generate enough acid and sulphate so that trends in the variations could be detected.

The formation of ARD and the migration of the acidic solutions are codetermined by the hydrologic conditions of the deposit and its surroundings. The permeability and the mechanical defects define the flow velocities of water and oxygen in dumps and in bedrock. The sizes of catchment areas and of receiving streams characterise the magnitude of potential polluting damage of the deposit and its surrounding environments.

Waste rock management units usually offer good permeability because of their coarse-grained components on average. According to POLTNIG and REINSDORFF (2000), rainfall water which infiltrates the dump body will find good flow paths for that reason. Thus the flow will happen with relatively high velocities. In the footwall of the mined talc the seepage water encounters a formation with low water permeability. The seepage follows this formation southwards to the Floingbach (POLTNIG & REINSDORFF, 2000).

According to SUETTE (2004), the outcrops of springs at Rabenwald are responsive to metrological conditions. This fact is seen as evidence of low water-holding capacity of the bedrock. The retention time of water in the rock is thus relatively short.

In bedrock of the deposit seepage occurs along joints and fault zones and can lead to reactions there.

The fact stated above indicates an important difference to the test arrangements in the laboratory. There was no change of either the basis fertiliser solutions or the rock samples carried out during the test times of the leaching tests. Water in bedrock and dumps of the deposit is flowing with high velocities and reaches new reactive rock material consistently. Sulphates, which are produced by the oxidation processes, migrate immediately after formation with seepage water. The differences between the retention times of water as well as the amounts of ‘fresh’ rock available for reaction can be seen as possible reasons for the varying sulphate concentrations of the laboratory tests and the open pit mine waters.

It was expected that the solutions of the leaching tests would yield higher sulphate concentrations than the control points of the deposit, but the opposite happened. The sulphate concentrations of the examined samples do not reach the critical values subject to the drinking water directive in spite of their fineness of grains and their long reaction times. The concentrations measured in the laboratory under-run the highest measured value in the deposit (Ernst August gallery; 3rd quarter 2007) by a factor of more than 10. It has to be mentioned restrictively that the sulphate concentrations of the samples in the laboratory are still increasing at the end of the tests.

The assumption seems to be likely that the high sulphate concentrations of the measurement points at the deposit are based on the water flow over ‘fresh’ rock surfaces. The leaching tests indicate that the oxidation processes are at least attenuated even after short time periods. Thus the bigger part of the produced acid and sulphate in the deposit emanates from recently deposited rock material. In contrast to the laboratory the decomposition of big components of the dump results in continued exposure of sulphides and therefore in a more constant action of the oxidation processes over longer time periods.

Almost the whole surface mining is drained by the closed systems of the Ernst August gallery and the Lilly gallery. The surface waters of the open pit mining due to rainfall events are brought into the galleries using pumps as well (POLTNIG & REINSDORFF, 2000). The processes described above most likely lead to the elevated sulphate concentration of the control points of the Rabenwald open pit mining.

The entire water of the Ernst August gallery flows into the retention basin “Wasserspiegel Brunnen” and seeps away there. The high sulphate concentrations of the Ernst August gallery and the retention basin therefore can lead to problems in deeper ground water horizons as well. The high differences between the sulphate concentrations of the control points of the Ernst August gallery and of the retention basin starting from 2006 are particularly striking. This can be for several reasons. Dilution can occur or seepage of the open pit mine waters or precipitation of sulphate minerals on the way between the two control points. Furthermore measuring errors cannot be excluded.

The Lilly gallery drains into the Floingbach as large areas of the open pit mining as well (POLTNIG and REINSDORFF, 1998). The size of the drainage area of the Floingbach will still be expanded as a result of a future mining plan (POLTNIG and REINSDORFF, 1998). Hence the Floingbach will determine the discharge conditions of the deposit, in particular those of the dump waters, to a higher extent. According to SUETTE (2004) this fact will lead to qualitative and quantitative damage due to the modified supply of water.

The elevated sulphate concentrations of the open pit mine waters measured in the last few years are due to oxidation processes. The responsible rock types can be found with actinolite gneisses and calcsilicates. Those rocks hardly have potential to produce acid leachate as the produced acid is buffered in situ. Sulphate as a product of the oxidation of sulphides migrates with seepage and is reflected in the high concentrations measured of open pit mine waters however. Thus the elevated values are a direct consequence of the exposure and deposition of actinolite gneisses and calcsilicates and the oxidation processes of sulphides contained therein. Consecutively the reasons for the increase in sulphate concentrations of the open pit mine waters of the Ernst August gallery and the Lilly gallery are explained.

Sulphate concentrations of the open pit mine waters of the Ernst August gallery have shown a significant increase since 2000/2001. This is probably based on augmented deposition of actinolite gneisses and calcsilicates on the dump from that time.

At the Rabenwald deposit the mining of talc in the south of the Krughof Hill was initiated in the last few years. The control point of the Lilly gallery is located in the sphere of influence of this newly started mining activity. The SO_4 concentrations of the open pit mine waters of the Lilly gallery have strongly increased since 2005. At the same time the clearance of waste rock and first exposure of bedrock in the south pit happened. The measurement of the fourth quarter reveals almost a doubling of the sulphate concentrations. Thus the high sulphate concentrations of the Lilly gallery feature a certain dependency on the operation in the south pit. The Lilly gallery drains into the Floingbach which has a similar dependency of mining operations in the south consequently.

There are different methods to prevent or minimise the formation of ARD. The kind of deposition as well as the distribution of different rock types in the dump has to be considered with questions concerning potential acid generation and possible counteractions. Areas of the dump, where only sulphide-bearing rock material is deposited, have of course the tendency to produce acid and to form ARD

There are several possibilities to control the ARD process which can be realised relatively simple. By mixing sulphide-bearing material with alkaline rocks during deposition, generated acid can be neutralised in situ. A further possibility exists in the alternate layering of rocks with ARD potential and rocks with the capacity to buffer acid. Storing of carbonates above sulphide-bearing rocks leads to an increase of alkalinity of the seepage already before it reaches the formations of potential acid generation and therefore prevents the production of acidic solutions. The encapsulation of sulphide-bearing rock material from water and oxygen in order that the oxidation process cannot take place is another method to prevent the formation of ARD. The reliability of this method over longer time periods is doubtful however.
The hydrologic conditions of the deposit, the dump and the environment affect the acid generation as well as the migration of acidic solutions. The acid generation of

sulphide-bearing rock material, which is deposited under water saturated conditions, is limited by the slow diffusion of oxygen in water.

As already mentioned above, the rocks of the Rabenwald talc deposit contain only small amounts of sulphide minerals. Thus the generated acid by the oxidation processes can be neutralised in situ. So no acidic solutions are produced; the deposit has no potential for the formation of ARD. Problems occur by means of high sulphate concentrations of open pit mine waters. To reduce these concentrations the oxidation process with acid generation and sulphate production need not only be controlled but has to be prevented completely. From the methods described above just the encapsulation of sulphide minerals of water and oxygen is suitable for Rabenwald surface mining. The method of encapsulation prevents the oxidation process and thus the production of sulphate. As mentioned above, the long-term reliability of this method is questionable.

9. REFERENCES

BRITISH COLUMBIA AMD TASK FORCE (1989): *Acid Rock Drainage Draft Technical Guide, Volumes I and II*. Report 66002/2. Prepared for the British Columbia AMD Task Force by SRK, Inc.

BELOCKY, R. (1992): *Regional vergleichende Untersuchung lagerstättenbildender Fluide in den Ostalpen als Hinweis auf eine mögliche metamorphe Ableitung*. Bswg. geol.-paläont. Diss., **14**, 103S., TU Braunschweig.

BUCKMANN, H. O., BRADY, N. C. (1960): *The nature and Properties of Soils*. 6th Edition. New York: Macmillan. 567.

BUSENBERG, E., CLEMENCY, C. (1967): *The Dissolution Kinetics of Feldspars at 25° C and 1 Atmosphere CO_2 Partial Pressure*. Geochim. Cosmochim, Acta 40: 41-49.

CARUCCIO, F. T., GEIDEL, G. (1981): *Estimating the Minimum Acid Load That Can Be Expected From a Coal Strip Mine*. Symposium on Surface Mining Hydrology, Sedimentation and Reclamation, Lexington, N. Y.

DIONEX CORPORATION (1997): *DX-120 ION Chromatograph operator's manual*. U.S.A.

FANK, J., HARUM, T., HABSBURG-LOTHRINGEN, C., STICHLER, W. (1999): *Agri-Environmental Measures and Water Quality in Mountain Catchments - AGRENUALP, Final Report*. Unveröffentlicher Bericht, Institut für Hydrogeologie und Geothermie, Joanneum Research, 55S., Graz.

FERGUSON, K. D., ERICKSON, P. M. (1986): *Pre-Mine Prediction of Acid Mine Drainage*. Salomans W. Forstner U. (eds) Environmental Impact and Managment of Mine Tailings and Dredged Materials. Springer Verlag, Heidelburg.

FLÜGEL, H. W., NEUBAUER, F. (1984): *Erläuterungen zur Geologischen Karte der Stmk*. 1-127, GBA.

FRIEDRICH, O. M. (1947): *Die Talklagerstätte des Rabenwalds*. Ostmk.- Berg.- u. Hüttenmänn. Mh, 92, 66-85.

HERITSCH, H. (1967): *Über die Bildungstemperaturen der Talklagerstätte Rabenwald*. Miner. Mitt.bl. Joanneum 1/2, 40-45.

HUTCHINSON, I. P. G., ELLISON, R. D. (1992): *Mine waste managment*. Sponsored by the California Mining Association. Published by Lewis publishers.

KIESL, W., WIESENEDER, H., KLUGER, F. (1983): *Untersuchungen des Vorkommens der Seltenen Erden und von Thorium in Gesteinen des unterostalpinen Kristallins des Semmering-Wechselfensters*. Sitzber. Österr. Akad. Wiss., math.-naturwiss. Kl., Abt. I, **192**, 1-20, Wien.

KIESL, W., KLUGER, F., WIESENEDER, H. (1986): *Petrology and geochemistry of selected talc deposits in the eastern Alps*. Mineral Paragenesis, Athen, 649-664.

KLEINMANN, R., CRERAR, P., PACELLI, R. (1981): *Biogeochemistry of Acid Mine Drainage and a Method to Control Acid Formation*. Mining Engineering.

KNAPP, R. (1987): *Bio/Geochemistry of Acid Generation in Sulphides Tailings and Waste Rock*. Proceedings of the ARD Seminar/Workshop. Environment Canada, Halifax, Nova Scotia.

LAPAKKO, L. (1987): *Prediction of ARD from Duluth Complex Mine Waste in North Eastern Minnesota*. Acid Mine Drainage Workshop. DSS Cat. No. En. 40-11-7 11987E. 187-221.

LUZENAC NAINTSCH RABENWALD: *Data about infiltration, climate, quality of the open pit mine waters and locations of the measuring points*.

MOINE, B., FORTUNE, J. P., MOREAU, P., VIGUIER, F. (1989): *Comparative mineralogy, geochemistry and conditions of formation of two metasomatic talc and chlorite deposits : Trimouns (Pyrenees, France) and Rabenwald (Eastern Alps, Austria)*. Economic Geology, Vol. 84, 1398-1416.

MOREAU, Ph. (1981): *Le Massif du Rabenwald et ses Minéralisations (Talc, Chlorite, Disthène, Leuchophyllite)*. Diss. Univ. Besancon, p. 327.

POLTNIG, W., REINSDORFF, S. (1998): *Tagebau Rabenwald - Hydrogeologische Bewertung der Abbauführung auf die Einzugsgebiete der Vorfluter des Bergbaues Rabenwald: Dunstbach, Floingbach, Schmidbach und Lambach, Tätigkeitsbericht 1. Arbeitsjahr 1998*. Unveröff. Bericht, Institut für Hydrogeologie und Geothermie, Joanneum Research, 13S., Graz.

POLTNIG, W., REINSDORFF, S. (2000): *Tagbau Rabenwald - Hydrogeologische Bewertung der Abbauführung auf die Einzugsgebiete der Vorfluter des Bergbaues Rabenwald: Dunstbach, Floingbach, Schmidbach und Lambach, Endbericht 2. Arbeitsjahr 1999*. Unveröff. Bericht, Institut für Hydrogeologie und Geothermie, Joanneum Research, 24S., Graz.

PROCHASKA, W. (1988): *Geologie, Geochemie, und Genese der ostalpinen Talklagerstätten*. Habilitationsschrift MU-Leoben, 1-253.

SCHROLL, E., PAPESCH, W., DOLEZAL, P. (1986): *Beitrag der C- und O-Isotopenanalyse zur Genese ostalpiner Sideritvorkommen*. Mitt. österr. geol. Ges., Wien, 78, 181-191.

SCHUSTER, K., BERKA, R., DRAGANITS, W., SCHUSTER, R. (2001): *Lithologien, Metamorphosegeschichte und tektonischer Bau der kristallinen Einheiten am Alpenostrand*. Arbeitstagung der Geologischen Bundesanstalt 2001, 29-56.

SILVER, M. (1987): *Aquatic Plants and BOG Covers to Prevent Acid Generation - Base Metal Tailings*. Proceedings of the ARD Seminar/Workshop. Halifax.

SMITH, E. E., SHUMATE, K. S. (1971): *Sulfide to Sulphate Reaction Mechanism. A Study of the Sulfide to Sulphate Reaction Mechanism as It Relates to the Formation of Acid Mine Waters*. Report by Ohio State University Research Foundation to the United States Federal Water Pollution Control Administration, PB-213 876.

SUETTE, G. (2004): *Wiedergewältigung Krughof*. Unveröffentlichtes Gutachten. Technisches Büro für Geologie, Dörfla.

TOLLMANN, A. (1976): *Neue Fenster des Wechselsystems am Ostrand der Zentralalpen*. Jber. 1975 Geol. Tiefbau Ostalpen, **3**, 58-64, Wien (Zentralanst. Meteor. Geodyn.).

TOLLMANN, A. (1977): *Geologie von Österreich*. Deuticke Verlag Wien, p. 1-766.

WALSH, F. M., MITCHELL, R. (1975): *Mine Drainage Pollution Reduction by Inhibition of Iron Bacteria*. Water Research, IX: 525-528.

WIESENEDER, H. (1971): *The eastern end of the central Alps*. Guide Excurs. Int. Geol. Congr. 23 Sess. Prague, 32 C, Geol. B.-A., Wien, 25-42.

WILLIAMS, E. G., et al. (1982): *Factors Controlling the Generation of Acid Mine Drainage*. Report to the United States Bureau of Mines, Research Grant No. G5105086. 256.

WURM, M. (2005): *Geological assessment and computer based three dimensional deposit modelling of the Rabenwald talc deposit taken as a basis for a forward-looking product development*. Dissertation MU-Leoben.

10. APPENDIX

10.1 Results of the Leco analyses

10.1.1 Table 1: Results of the Leco analysis for talc (TA), leucophyllite (LP), marble (MA), granite gneiss (GG), paragneiss (PG), and hornblende gneiss (HG)

Sample name	Sulphur content [%]			
	Measurement 1	Measurement 2	Mean of the measurements	Mean of the rock type
TA 1	0.009	0.008	0.009	0.007
TA 2	0.009	0.007	0.008	
TA 3	0.008	0.008	0.008	
TA 4	0.007	0.005	0.006	
TA 5	0.006	0.007	0.007	
TA 6	0.007	0.006	0.006	
LP 1	0.008	0.008	0.008	0.009
LP 2	0.012	0.009	0.010	
LP 3	0.009	0.008	0.008	
LP 4	0.008	0.008	0.008	
LP 5	0.010	0.008	0.009	
MA 1	0.005	0.004	0.004	0.003
MA 2	0.004	0.003	0.003	
MA 3	0.003	0.002	0.003	
MA 4	0.004	0.003	0.004	
MA 5	0.003	0.002	0.003	
GG 1	0.011	0.012	0.011	0.010
GG 2	0.012	0.010	0.011	
GG 3	0.011	0.009	0.010	
GG 4	0.010	0.008	0.099	
PG 1	n.a.	0.010	0.010	0.009
PG 2	0.007	0.009	0.008	
PG 3	0.009	0.010	0.009	
PG 4	0.009	0.009	0.009	
HG 1	0.011	0.010	0.010	0.009
HG 2	0.009	0.010	0.009	
HG 3	0.009	0.008	0.009	
HG 4	0.009	0.009	0.009	

Sample name	Sulphur content [%]			
	Measurement 1	Measurement 2	Mean of the two measurements	Mean of the rock type
HG 5	0.008	0.009	0.009	0.009

10.1.2 Table 2: Results of the Leco analysis for calcsilicate (CS) and actinolite gneiss (AG)

Sample name	Sulphur content [%]				
	Measurem. 1	Measurem. 2	Measurem. 3	Mean of the measurements	Mean of the rock type
CS 1	1.469	1.455	n.a.	1.462	0.593
CS 2	0.970	0.988	n.a.	0.979	
CS 3	0.367	0.379	n.a.	0.373	
CS 4	0.500	0.483	0.479	0.487	
CS 5	0.193	0.192	n.a.	0.192	
CS 6	0.222	0.201	0.184	0.202	
CS 7	0.866	0.875	0.855	0.865	
CS 8	0.544	0.533	0.516	0.531	
CS 9	0.245	0.254	0.236	0.245	
AG 1	0.815	0.812	n.a.	0.814	1.321
AG 2	2.756	2.845	n.a.	2.801	
AG 3	0.875	0.937	n.a.	0.906	
AG 4	1.114	1.120	n.a.	1.117	
AG 5	2.711	2.679	n.a.	2.695	
AG 6	0.720	0.785	n.a.	0.752	
AG 7	1.677	1.856	1.678	1.737	
AG 8	0.201	0.177	0.222	0.200	
AG 9	0.820	0.900	0.791	0.837	
AG 10	0.024	0.026	0.028	0.026	
AG 11	2.602	2.699	2.628	2.643	

10.1.3 Table 3: Results of the Leco analysis for the waste rock sample (WRS)

Sample name	Sulphur content [%]			
	Measurement 1	Measurement 2	Mean of the measurements	Mean of the WRS
WRS 1.1	0.202	0.203	0.202	0.231
WRS 1.2	0.197	0.206	0.201	
WRS 1.3	0.213	0.362	0.288	

10.1.4 Table 4: Results of the Leco analysis for the 'artificial' bulk samples (ABS)

Sample name	Sulphur content [%]				
	Measurem. 1	Measurem. 2	Mean of the measurements	Mean of the single ABS	Mean of the three ABS
ABS 1.1	0.083	0.106	0.095	0.111	0.081
ABS 1.2	n.a	0.120	0.120		
ABS 1.3	0.112	0.128	0.120		
ABS 2.1	n.a.	0.058	0.058	0.051	
ABS 2.2	0.040	0.042	0.041		
ABS 2.3	0.051	0.057	0.054		
ABS 3.1	0.066	0.073	0.070	0.080	
ABS 3.2	0.066	0.099	0.083		
ABS 3.3	0.091	0.082	0.086		

10.2 Results of the leaching tests (LT)

10.2.1 Table 5: Measured pH values of the solutions of LT 1 (waste rock sample)

Sample name	Test time [h]	pH value	Sample name	Test time [h]	pH value	Sample name	Test time [h]	pH value
1.1	0	7	1.4	216	8.54	1.7	720	8.12
2.1	0	7	2.4	216	8.25	2.7	720	8.10
3.1	0	7	3.4	216	8.2	3.7	720	8.07
4.1	0	3.6	4.4	216	8.42	4.7	720	8.10
5.1	0	3.6	5.4	216	8.07	5.7	720	8.03
6.1	0	3.6	6.4	216	7.87	6.7	720	7.83
1.2	48	8.99	1.5	360	8.36	1.8	1296	8.12
2.2	48	8.46	2.5	360	8.24	2.8	1296	8.10
3.2	48	8.52	3.5	360	8.19	3.8	1296	8.06
4.2	48	8.66	4.5	360	8.24	4.8	1296	8.10
5.2	48	7.68	5.5	360	8.03	5.8	1296	8.08
6.2	48	7.33	6.5	360	7.83	6.8	1296	7.86
1.3	120	8.66	1.6	528	8.21	1.9	3840	7.97
2.3	120	8.1	2.6	528	8.15	2.9	3840	8.01
3.3	120	8.14	3.6	528	8.11	3.9	3840	7.92
4.3	120	8.37	4.6	528	8.14	4.9	3840	8.02
5.3	120	7.6	5.6	528	8.02	5.9	3840	8.02
6.3	120	7.36	6.6	528	7.79	6.9	3840	7.80

10.2.2 Table 6: Measured pH values of the solutions of LT 2 (bulk samples)

Sample name	Test time [h]	pH value	Sample name	Test time [h]	pH value	Sample name	Test time [h]	pH value
BS 1.1	0	7	BS 1.5	310	7.70	BS 1.9	982	7.80
BS 2.1	0	7	BS 2.5	310	8.05	BS 2.9	982	7.92
BS 3.1	0	7	BS 3.5	310	8.46	BS 3.9	982	8.43
BS 1.2	71	7.44	BS 1.6	478	7.68	BS 1.10	1150	7.82
BS 2.2	71	8.06	BS 2.6	478	7.95	BS 2.10	1150	7.91
BS 3.2	71	8.93	BS 3.6	478	8.38	BS 3.10	1150	8.44
BS 1.3	142	7.64	BS 1.7	670	7.76	BS 1.11	1870	7.78
BS 2.3	142	7.93	BS 2.7	670	7.94	BS 2.11	1870	7.91
BS 3.3	142	8.52	BS 3.7	670	8.39	BS 3.11	1870	8.42
BS 1.4	190	7.62	BS 1.8	838	7.79			
BS 2.4	190	7.98	BS 2.8	838	7.94			
BS 3.4	190	8.46	BS 3.8	838	8.45			

10.2.3 Table 7: Measured pH values of the solutions of LT 3 (actinolite gneiss)

Test time [h]	pH value	Test time [h]	pH value	Test time [h]	pH value
0	7	497	5.65	1986	5.58
16	7.28	569	5.59	2490	5.33
88	7.27	1339	5.57	3090	5.48
354	5.69	1506	5.56	4386	5.36

10.2.4 Table 8: Measured pH values of the LT 4 (acid rock drainage)

Sample name	Test time [h]	pH value	Sample name	Test time [h]	pH value	Sample name	Test time [h]	pH value
Wa	0	7	PP	70	8.28	CuS	500	8.54
PP	0	7	CuS	70	8.40	Wa	668	4.54
CuS	0	7	Wa	142	4.88	PP	668	8.06
Wa	4	4.65	PP	142	8.15	CuS	668	8.64
PP	4	8.67	CuS	142	8.35	Wa	835	4.44
CuS	4	8.74	Wa	332	4.74	PP	835	8.07
Wa	26	4.72	PP	332	8.05	CuS	835	8.68
PP	26	8.32	CuS	332	8.35	Wa	1173	4.43
CuS	26	8.43	Wa	500	4.47	PP	1173	8.15
Wa	70	4.74	PP	500	8.02	CuS	1173	8.72

Sample name	Test time [h]	pH value	Sample name	Test time [h]	pH value	Sample name	Test time [h]	pH value
Wa	1533	4.32	CuS	1698	8.77	PP	3019	8.00
PP	1533	8.04	Wa	2515	3.95	CuS	3019	8.66
CuS	1533	8.67	PP	2515	8.20	Wa	3691	3.90
Wa	1698	4.13	CuS	2515	8.67	PP	3691	7.83
PP	1698	8.06	Wa	3019	4.00	CuS	3691	8.60

10.3 Results of the ion chromatography

10.3.1 Table 9: Results of the ion chromatography for the basis fertiliser solutions of the LT 1

Sample name	Test time [h]	SO_4 [mg/l]	F [µg/l]	Cl [µg/l]	Li [ppb]	Na [ppb]	K [ppb]	Mg [ppb]	Ca [ppb]
1.1	0	0	0	0	0	0	0	0	0
2.1	0	0	0	0	0	0	0	0	0
3.1	0	0	0	0	0	0	0	0	0
4.1	0	0.2	8.3	7229.9	n.a	25.4	115.3	28.9	212.5
5.1	0	0.2	8.3	7229.9	n.a	25.4	115.3	28.9	212.5
6.1	0	0.2	8.3	7229.9	n.a	25.4	115.3	28.9	212.5
1.2	48	19.9	97.5	2573.0	1.6	2995.4	5604.5	1420.4	6756.5
2.2	48	17.7	30.7	1070.7	n.a	853.2	1979.4	1986.8	7224.4
3.2	48	19.0	38.8	1416.7	n.a	1213.0	2264.6	2127.8	3554.1
4.2	48	21.4	98.5	9637.4	1.9	3069.2	6410.4	2706.9	11940.5
5.2	48	13.7	40.5	7982.5	n.a	823.0	2127.4	2361.8	10592.7
6.2	48	22.2	34.5	8324.1	n.a	1159.5	2703.5	3215.3	4781.0
1.3	120	27.3	130.7	3531.6	1.7	4201.3	7513.4	2501.7	12416.9
2.3	120	21.0	41.1	1154.0	n.a	984.9	2549.5	2982.4	10488.6
3.3	120	25.0	42.0	1590.6	n.a	1305.4	3018.2	3596.6	5640.9
4.3	120	30.0	128.2	10748.9	2.3	4167.7	8247.0	3873.4	16500.3
5.3	120	22.8	47.0	8964.5	n.a	965.9	3071.3	3602.9	14426.3
6.3	120	29.8	43.1	8497.2	1.6	1305.4	3484.6	4506.5	7355.2
1.4	216	35.6	164.5	4279.8	2.3	5331.1	9112.4	3627.0	15773.6
2.4	216	24.1	59.4	1171.8	n.a	1036.7	3012.1	3750.6	12622.5
3.4	216	28.6	51.8	1668.4	n.a	1401.7	3374.5	4432.2	6642.1
4.4	216	36.5	160.6	11122.9	2.8	5107.3	9626.2	4696.8	18831.1
5.4	216	23.9	43.0	8073.4	n.a	900.5	3226.9	3958.8	15211.5
6.4	216	34.8	47.8	8377.4	1.9	1390.5	3929.8	5447.7	8465.1
1.5	360	43.5	197.5	4705.2	2.8	6196.9	10400.2	4557.0	18699.1
2.5	360	25.9	55.1	1146.5	n.a	1035.8	3358.2	4295.6	14432.2

Sample name	Test time [h]	SO_4 [mg/l]	F [µg/l]	Cl [µg/l]	Li [ppb]	Na [ppb]	K [ppb]	Mg [ppb]	Ca [ppb]
3.5	360	32.3	55.4	1651.8	1.6	1510.1	3995.2	5332.5	7504.2
4.5	360	43.9	186.7	11409.1	3.4	5843.5	10759.3	5433.6	21272.7
5.5	360	26.1	49.0	7970.9	n.a	909.9	3548.8	4447.7	16981.8
6.5	360	39.0	54.0	8383.2	2.4	1545.2	4584.6	6477.0	9684.8
1.6	528	51.6	224.1	4892.4	3.5	6841.3	11466.5	5366.3	21511.6
2.6	528	27.6	61.3	1187.3	1.6	1052.7	3783.9	4734.0	16015.9
3.6	528	35.1	62.2	1711.4	1.7	1546.2	3962.5	6103.7	8370.7
4.6	528	49.3	204.8	11213.2	3.6	6314.5	11461.6	6071.5	23183.1
5.6	528	28.0	51.2	7938.0	1.6	939.8	4019.2	4855.7	18457.6
6.6	528	42.1	57.3	8380.1	2.6	1538.3	5155.4	7191.2	10573.9
1.7	720	58.7	244.1	5038.8	3.8	7351.1	12380.6	6058.0	23970.0
2.7	720	29.6	66.3	1341.4	1.8	1126.0	4318.6	5180.4	17506.1
3.7	720	36.7	67.5	1743.8	2.0	1578.5	4126.7	6563.4	8901.9
4.7	720	56.6	223.9	11345.5	4.3	6713.5	12169.8	6621.0	24921.2
5.7	720	29.3	54.9	7739.4	1.7	938.3	4365.8	5061.6	19336.6
6.7	720	55.4	65.4	8368.8	2.8	1585.5	5597.3	7819.8	11357.5
1.8	1296	75.3	284.9	5195.8	4.4	8074.1	13818.5	7358.8	28319.1
2.8	1296	33.3	75.7	1321.7	2.0	1112.9	5053.6	5670.8	19628.6
3.8	1296	41.1	76.7	1746.3	2.3	1590.2	4891.6	7459.1	9953.3
4.8	1296	71.4	259.6	11074.3	4.1	7171.6	13275.5	7624.2	28601.3
5.8	1296	35.0	55.8	7623.1	2.1	992.4	5257.0	5666.3	21935.5
6.8	1296	47.8	67.6	8095.9	3.4	1616.4	6143.5	8678.3	12098.6
1.9	1968	93.2	309.3	5136.4	n.a	n.a	n.a	n.a	n.a
2.9	1968	37.0	80.9	1340.2	n.a	n.a	n.a	n.a	n.a
3.9	1968	46.1	85.3	1757.8	n.a	n.a	n.a	n.a	n.a
4.9	1968	95.4	290.1	11424.1	n.a	n.a	n.a	n.a	n.a
5.9	1968	41.1	60.2	7830.4	n.a	n.a	n.a	n.a	n.a
6.9	1968	53.0	78.6	8416.6	n.a	n.a	n.a	n.a	n.a
1.10	2640	110.3	363.1	5277.9	5.9	8079.1	14475.8	6935.0	28855.9
2.10	2640	41.2	134.8	1688.2	3.5	1098.6	4774.2	3912.5	16118.6
3.10	2640	50.1	160.7	2413.4	2.2	1474.9	4817.3	6283.3	9738.5
4.10	2640	113.1	376.0	11449.1	3.3	7158.9	13860.0	7818.7	36582.1
5.10	2640	46.9	97.3	7893.6	2.0	950.9	5010.9	4534.9	22507.1
6.10	2640	55.8	112.9	8317.3	4.0	1490.5	5808.4	8172.5	11024.1

10.3.2 Table 10: Results of the ion chromatography for the basis fertiliser solutions of the LT 2

Sample name	Test time [h]	SO_4 [mg/l]	F [µg/l]	Cl [µg/l]	Li [ppb]	Na [ppb]	K [ppb]	Mg [ppb]	Ca [ppb]
BS 1.1	0	0	0	0	0	0	0	0	0
BS 2.1	0	0	0	0	0	0	0	0	0
BS 3.1	0	0	0	0	0	0	0	0	0
BS 1.2	142	33.8	108.8	1185.5	3.5	1563.7	9706.9	4969.4	16329.3
BS 2.2	142	28.6	87.4	926.7	2.4	1208.8	7972.7	4391.0	15451.2
BS 3.2	142	n.a.	n.a.	n.a.	n.a.	0.8	570.5	4.3	31.3
BS 1.3	478	37.0	126.3	1317.5	3.4	1367.1	12449.8	5973.7	21794.9
BS 2.3	478	38.5	80.2	812.6	3.0	1273.5	11713.1	6047.4	22095.9
BS 3.3	478	71.3	459.7	5526.7	22.6	12226.3	63314.4	22832.6	13082.6
BS 1.4	670	39.9	143.4	1377.7	3.1	1394.4	13819.9	6375.3	24191.1
BS 2.4	670	42.1	145.5	1006.5	2.9	1297.5	12777.6	6452.7	24264.2
BS 3.4	670	73.8	470.4	5366.8	23.6	12362.2	64234.0	25260.2	12858.4
BS 1.5	838	42.5	148.8	1481.1	3.3	1421.8	14806.1	6658.0	25702.2
BS 2.5	838	45.3	135.1	984.8	3.4	1329.6	13635.2	6768.9	25886.4
BS 3.5	838	78.2	536.1	5875.4	24.1	12789.0	65682.5	26354.4	12916.2
BS 1.6	982	43.7	130.4	1645.6	3.0	1437.6	15495.9	6748.6	26371.4
BS 2.6	982	47.8	141.7	955.6	3.3	1361.6	14483.3	7053.5	27620.9
BS 3.6	982	87.1	527.6	6111.5	27.7	14139.2	73308.9	30137.3	14675.6
BS 1.7	1870	53.4	161.5	1645.0	3.2	1425.2	17260.0	7139.9	29472.9
BS 2.7	1870	58.4	127.0	928.6	3.8	1381.2	16486.8	7599.6	31163.0
BS 3.7	1870	81.0	490.0	5254.8	23.3	12294.2	64421.9	27757.5	13026.8

10.4 Results of the measurements of open pit mine water quality

10.4.1 Table 11.1: SO_4 values [mg/l] of the open pit mine waters at Rabenwald deposit

Measuring point	Year						Quarter							
	1999	2000	2001	2002	2003	2004	2_2005	3_2005	4_2005	1_2006	2_2006	3_2006	4_2006	1_2007
Wasserspiegel Brunnen	195.0	112.9	426.5	445.2	308.5	878.2	n.a.	n.a.	967.3	667.0	751.3	782.7	727.3	497.0
Brunnen Krughof	94.6	69.5	128.7	160.1	142.3	160.8	n.a.	n.a.	179.8	194.6	197.6	204.8	488.2	221.0
Lilly Stollen	88.2	70.5	154.9	211.7	214.2	304.0	246.0	356.2	355.5	449.0	n.a.	554.4	505.7	575.0
Lilly Quelle	99.0	59.9	114.5	141.1	n.a.	146.6	154.1	150.0	156.3	152.2	167.5	160.2	163.6	172.0
Ernst August Stollen	206.3	159.0	488.5	668.0	703.8	1007.8	1003.2	1059.4	1010.9	988.0	1072.0	1156.0	991.1	1119.0
Tagbau Sickerwasser	242.2	241.9	784.5	n.a.	n.a.	901.4	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Floingbach Brücke	86.5	80.8	149.8	189.4	192.5	266.5	238.1	n.a.	345.1	473.4	190.4	541.5	493.0	638.0
Floingbach Sailer	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	271.7	n.a.	n.a.	430.4	473.4	519.9	534.0	548.0
Rohrquelle Feldhofer	89.1	128.8	323.5	n.a.	372.7	60.9	562.9	n.a.	543.6	630.0	558.8	574.6	568.7	597.0
Quelle Rossegger	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Almer	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	53.1	53.3	53.3	56.6	65.3	98.0
Pumpensumpf	221.4	217.8	891.3	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Tagbau 1070	240.8	230.3	918.3	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Hummelbrunnen	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	313.3	317.2	319.1	356.8	303.1	350.0
Schlagbauerquelle	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	67.3	n.a.	n.a.	n.a.	317.2	n.a.	n.a.	n.a.
NMW Hochbehälter	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Quelle Heil	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	10.6
SO_4 limit	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0

In this table, the red font colour signalises an exceeding of the SO_4 limits according to the regulatory guideline for drinking water.

10.4.1 Table 11.2: SO$_4$ values [mg/l] of the open pit mine waters at Rabenwald deposit

Measuring point	Quarter								
	2_2007	3_2007	4_2007	1_2008	2_2008	3_2008	4_2008	1_2009	2_2009
Wasserspiegel Brunnen	466.0	446.0	n.a.	477.0	64.6	456.0	301.0	41.1	671.0
Brunnen Krughof	222.0	230.0	263.0	311.0	82.8	171.0	331.0	434.0	436.0
Lilly Stollen	588.0	539.0	1003.0	677.0	112.0	511.0	685.0	464.0	750.0
Lilly Quelle	165.0	194.0	162.0	131.0	129.0	144.0	132.0	141.1	145.0
Ernst August Stollen	1134.0	1405.0	1180.0	1182.0	1270.0	1009.0	1190.0	1350.0	1340.0
Tagbau Sickerwasser	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Floingbach Brücke	519.0	603.0	665.0	688.0	624.0	476.0	141.0	444.0	704.0
Floingbach Sailer	473.0	568.0	669.0	610.0	566.0	203.0	606.0	406.0	614.0
Rohrquelle Feldhofer	413.0	414.0	521.0	365.0	n.a.	n.a.	n.a.	n.a.	n.a.
Quelle Rossegger	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	613.0	176.0	217.0
Almer	96.0	66.8	111.4	184.0	81.3	82.0	115.0	79.4	94.0
Pumpensumpf	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Tagbau 1070	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Hummelbrunnen	343.0	305.0	258.0	282.0	262.0	239.0	268.0	192.0	230.0
Schlagbauerquelle	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	94.9	54.7	n.a.
NMW Hochbehälter	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	6.6
Quelle Heil	n.a.	27.4	n.a.	116.0	13.2	19.5	n.a.	9.2	5.7
SO$_4$ limit	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0	250.0

In this table, the red font colour signalises an exceeding of the SO$_4$ limits according to the regulatory guideline for drinking water.

10.4.2 Table 12.1: pH values of the open pit mine waters at Rabenwald deposit

Measuring point	Year						Quarter							
	1999	2000	2001	2002	2003	2004	2_2005	3_2005	4_2005	1_2006	2_2006	3_2006	4_2006	1_2007
Wasserspiegel Brunnen	8.0	7.2	8.2	7.9	8.2	7.4	n.a.	n.a.	8.0	7.8	7.3	7.6	7.3	7.4
Brunnen Krughof	8.1	7.9	8.3	7.9	8.2	7.6	n.a.	n.a.	7.8	7.9	7.8	7.6	7.8	7.8
Lilly Stollen	8.0	7.9	8.3	8.2	8.1	8.0	8.1	7.5	7.5	7.8	n.a.	7.4	7.4	7.4
Lilly Quelle	6.4	6.3	6.6	6.6	n.a.	6.0	6.5	6.2	6.1	6.2	6.1	6.1	6.2	6.2
Ernst August Stollen	7.7	7.4	8.1	7.8	7.5	6.4	7.1	6.6	6.8	6.8	7.1	6.8	7.0	6.5
Tagbau Sickerwasser	7.3	7.7	8.0	n.a.	n.a.	7.6	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Floingbach Brücke	8.1	8.3	8.4	8.3	8.3	8.2	8.3	n.a.	8.1	8.1	8.0	8.1	7.7	7.6
Floingbach Sailer	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	8.3	n.a.	n.a.	8.1	8.1	8.2	7.5	7.4
Rohrquelle Feldhofer	7.1	7.0	7.7	n.a.	7.1	7.1	6.9	n.a.	6.9	7.0	6.8	6.9	7.1	7.1
Quelle Rossegger	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Almer	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	6.1	6.2	6.3	6.1	6.1	6.2
Pumpensumpf	7.4	7.8	8.0	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Tagbau 1070	7.7	7.7	8.0	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Hummelbrunnen	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	6.1	6.2	6.2	6.1	6.0	6.1
Schlagbauerquelle	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	6.6	n.a.	n.a.	n.a.	6.2	n.a.	n.a.	n.a.
NMW Hochbehälter	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Quelle Heil	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	5.7
Upper pH limit	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5
Lower pH limit	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5

In this table, the red font colour signalises an exceeding or an undercut of the pH limits according to the regulatory guideline for drinking water.

10.4.2 Table 12.2: pH values of the open pit mine waters at Rabenwald deposit

Measuring point	**Quarter**								
	2_2007	**3_2007**	**4_2007**	**1_2008**	**2_2008**	**3_2008**	**4_2008**	**1_2009**	**2_2009**
Wasserspiegel Brunnen	7.8	7.6	n.a.	8.0	8.3	7.7	7.4	8.2	7.6
Brunnen Krughof	8.1	7.5	7.8	8.0	7.9	7.8	7.5	8.1	8.0
Lilly Stollen	7.7	7.3	7.4	7.4	7.6	7.4	7.4	8.6	7.4
Lilly Quelle	6.4	6.0	6.1	6.1	6.0	6.1	6.2	6.2	7.4
Ernst August Stollen	7.6	7.1	7.4	7.2	6.9	7.4	7.2	7.8	7.2
Tagbau Sickerwasser	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Floingbach Brücke	8.2	7.8	7.9	7.6	7.6	8.2	8.3	8.1	7.7
Floingbach Sailer	8.1	7.6	7.8	7.4	7.4	7.1	8.2	7.8	7.4
Rohrquelle Feldhofer	7.0	6.9	7.0	7.3	n.a.	n.a.	n.a.	n.a.	n.a.
Quelle Rossegger	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	7.4	7.2	7.4
Almer	6.6	6.1	6.1	6.0	5.9	6.4	6.2	6.4	7.4
Pumpensumpf	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Tagbau 1070	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Hummelbrunnen	6.5	6.0	6.3	6.0	6.4	6.3	6.4	6.3	6.9
Schlagbauerquelle	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	6.7	6.8	n.a.
NMW Hochbehälter	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	7.8
Quelle Heil	n.a.	5.8	n.a.	5.8	6.0	6.0	n.a.	6.4	7.1
Upper pH limit	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5
Lower pH limit	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5	6.5

In this table, the red font colour signalises an exceeding or an undercut of the pH limits according to the regulatory guideline for drinking water.