Bakouma Uranium Project

Central African Republic

Conceptual Study

Underwater Dredge Mining

by

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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.

1st September 2009

Underwater Dredge Mining

Summary

The uranium deposits from Bakouma in the Central African Republic have been known since a long time and were already evaluated for the first time in the ‘70s and ‘80s. The technical and economical condition at that time was not suitable for their mining. The actual reformation of the electric nuclear sector -as well as the recent increase in the price of the uranium- drove Areva, the 3rd global producer, logically restart feasibility studies of these deposits, with the objective of their exploitation to satisfy the market demand.

The region of Bakouma is in a tropical rain forest in a very humid zone. The rainy season can last up to 9 months per year; the ground water table is at a maximum depth of 10 m and the future mining sites are regularly flooded during the rainy season. Pumping the water is a major difficulty for a traditional mining site in an open pit and it also represents an important cost factor. To be certain from a conceptual point of view Areva wanted to study an alternative mining technique: underwater dredging.

In cooperation with the principal global fabricant of underwater dredge systems, IHC Merwede in the Netherlands, we could evaluate the characteristics of the equipments as well as their advantages and disadvantages. For the first time, different technical solutions of dredging were examined, especially two very purposeful techniques: dredging with boats and dredging with underwater equipments that are operated from a distance. For these two techniques, a drill arm realizes the mining of the material, which cuts the rock before sucking the produced slurry, a mixture of water and solid, for further transport via pipeline. A technical and economical comparison of the different dredging systems and of the traditional mining method resulted in an elimination of the variant with underwater equipments and in an increase in studies for dredging with boats.

Such a mining dispositive imposes several difficulties. First, the boats are limited in their dredging depth, which demands a progressive adaption of the water level in the pit; the mining has to be done in successive horizontal levels. Further, the ore is produced in a slurry form with solid content of approximately 20 to 30%; this is no storable product; the mouth-to-mouth function of the mining and treatment is a very important point. The waste material of the overburden is also under water. Important volumes of overburden in slurry form must be treated and stored in accordance with environmental exigencies and conditions of Areva.

With our knowledge, a solution of dredging with a boat was never done for mining of deposits where the upper layers came quasi to light and were the deepest layers, go down to 145 m from the topographical surface. The installation of such a technique therefore needs a radical different mining organization and planning compared to a classical open pit mining. Models of the deposits were created with blocks of 25*25*10 m³. With different technical assumptions for the models, we could design the boats for ore and waste mining. It has to be taken into account that the mining

phasing includes the transfer of the boats from one pit to the next and also the variability of the mineralization.

For the belongings to handle the waste slurry decantation and sedimentation, tests were performed in the laboratory with samples coming from different depths of the deposits in Bakouma. That allowed for the determining of a flow-sheet of a treatment installation of slurry for separating water and solids, and therefore the designing of needed equipments. The question of impact of that mining technique on the ore treatment unit was also technically, environmentally, and economically handled.

The study showed that underwater dredging is a technically possible solution. It presents advantages for the climatic and hydrological conditions (quasi-continuous work). On the other side it presents also a worst case mining which does not allow stabilizing the ore production over the lifetime of the mine, due to the heterogeneity of ore in the horizontal layers that have to be mined successive. Further the handling of the slurry and water is an essential challenge for the success of the operation, which demands quite some time of examination and experimentation when taking into account the very innovative character of the project.

Economically is underwater dredge mining compared to a traditional open pit mining for the conditions in Bakouma no favourable solution because higher capital and operating costs of the total scenario (mining and treatment equipments). Dredging could be favourable under other geological situations. Sandy material which is easy to settle or an adopted geometry of the deposit could give a totally different solution.

1 Introduction and general information

1.1 Central African Republic

The Central African Republic (CAR) is located in Central Africa. The capital is Bangui. The CRA (Illustration 1) is bordered clockwise from the north by Chad, Sudan, the Democratic Republic of Congo, the Republic of Congo and Cameroon.

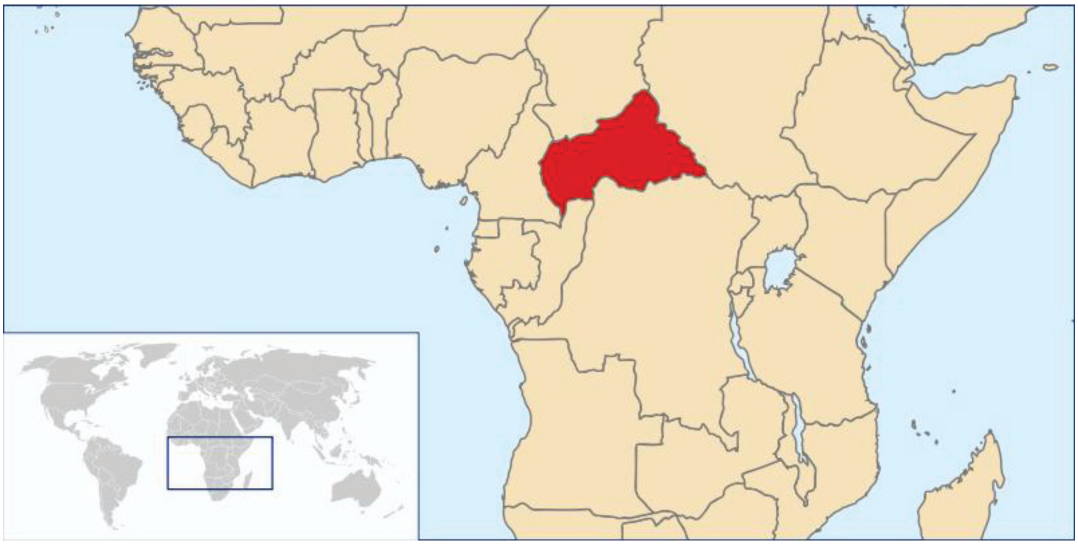


Illustration 1: Central African Republic

The climate is seasonally flooded. In the north, the rainy season lasts for four months, and in the south it can last eight to ten months. The bigger part of the country is tree savannah, which passes into the tropical rain forest in the south.

Less than one third of the land is used for agriculture, which serves for the greater part the local alimentation. Export products are cotton, peanuts, coffee, and palm fruits. The most important natural resources are diamonds, which are often smuggled (blood diamonds). In the west of the country there is also gold mining activity. For industry it is mainly the forestry and the agricultural industry that play an important role. Today, the Central African Republic is one of the poorest countries in the world.

The uranium resources in Bakouma would allow uranium to become the first export product of the Central African Republic. The first prospecting of the Bakouma uranium deposits was already done in 1959, and in 1969, a feasibility study was undertaken for the first time. In 1971, the deposits were evaluated to be non-profitable at that time during the existing economic situation.

1.2 General and local geology

Bakouma is situated in the south of the country in a Neo-Proterozoic sedimentary basin (Illustration 2). The area is flat, at an altitude of 500 to 550 m above sea level. The annual maximum mean temperature is between 27 to 33 °C, the minimum between 15 and 21 °C. The uranium deposits are situated in the rainforest approximately 5 km away from Bakouma. There is a rainy season that can last up to 9 months. The dry season is from January to March. The average rainfall is 1.6 m every year. The ground water table rises to 10 m under the surface.

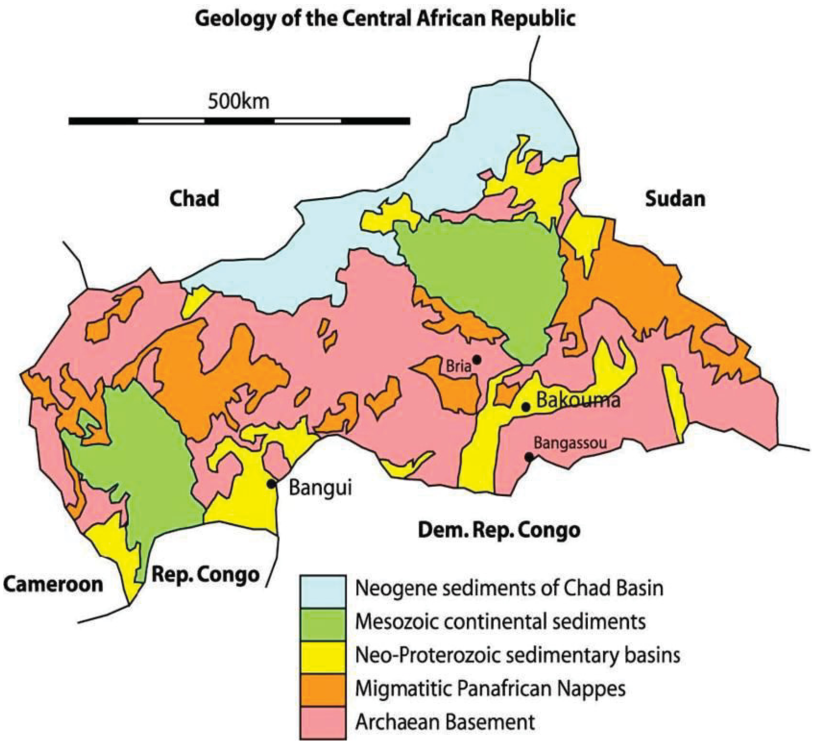


Illustration 2: Geology of the Central African Republic

Geologically, the Bakouma area is a syncline with a SW-NE axis. The edges of the syncline are marked by Precambrian quartzites forming a semicircle. On these quartzites is found the so-called Bakouma series which is probably Infracambrian. This series is composed, first of tillite and varved clay, than of dolomite and limestone. It is only known in bore-holes.

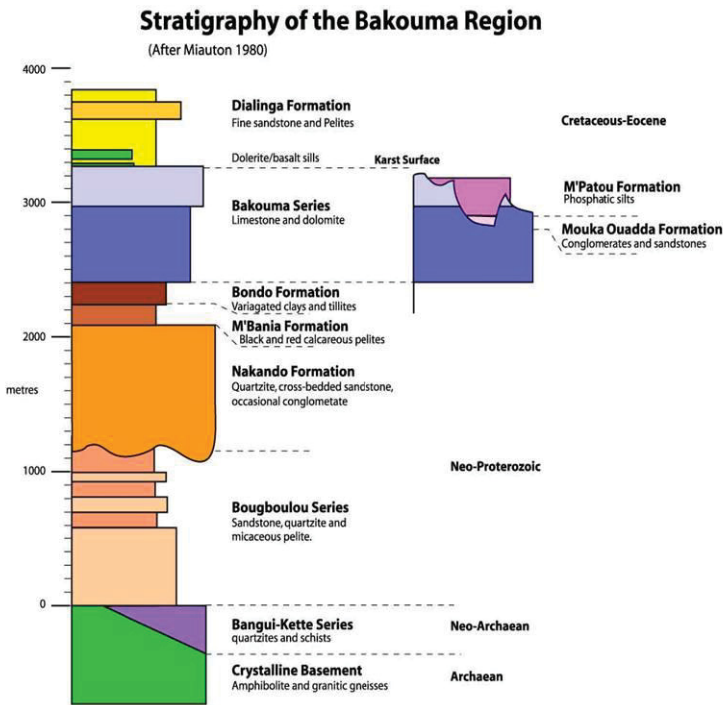


Illustration 3: Stratigraphy of the Bakouma region

The northern part of the syncline is cut off by the quartzites of the dialinga formation (upper Precambrian) (Illustration 3). The dolomite formation was eroded deeply until, in the Eocene, the Bakouma series (Illustration 5) was filled by clay silts. This aggradation is heterogeneous: straight above the karstic erosion funnels, which can be up to 80 m deep, the clay silts pass laterally to calcium and alumina phosphates. These phosphates are the host rock for the known uraniferous mineralization (Illustration 4).

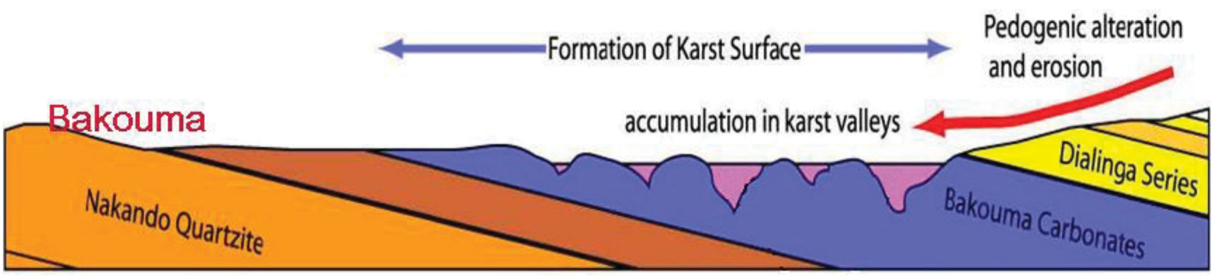


Illustration 4: Genesis of the deposits

Uranium occurs in its hexavalent form – autunite and torbernite – generally in the upper layers (light colour formations), or as tetravalent ions dispersed in the crystal lattice of the francolites (fluorapatites) in which case it only can be extracted by destroying the phosphate molecule.

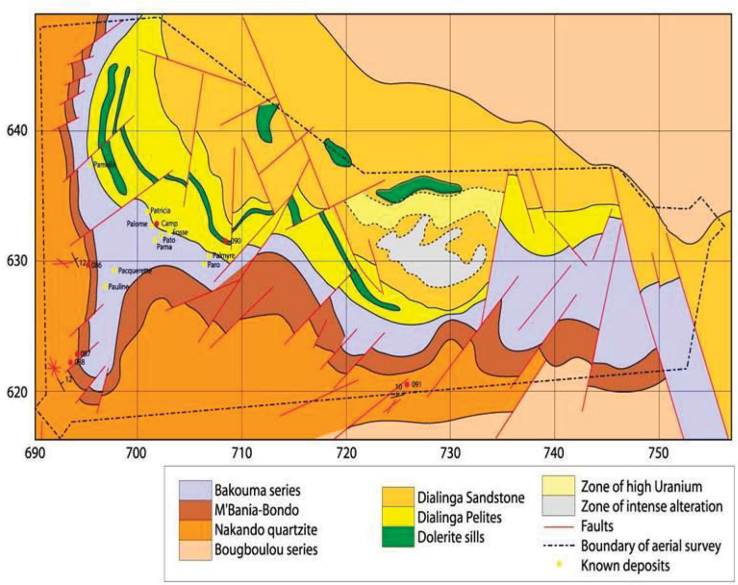


Illustration 5: Geological map and location of the uranium deposits

The uranium deposits are located in an area without any existing infrastructure. That means that for the mining of the deposits, everything has to be built up from the beginning (roads, energy systems, etc.) (Illustration 6).

Illustration 6: Impressions of the local situation

The 3 detected uranium deposits are: Patricia, Pato-Pama and Fosse (Illustration 7). Pato-Pama is seen as one deposit because its two ore bodies are very close together and will therefore be mined together. The uranium deposits have a maximum depth of 145 m. The tonnages of the ore bodies vary between $3.2*10^6$ and $5.0*10^6$ t. Their total volume is $12.3*10^6$ t.

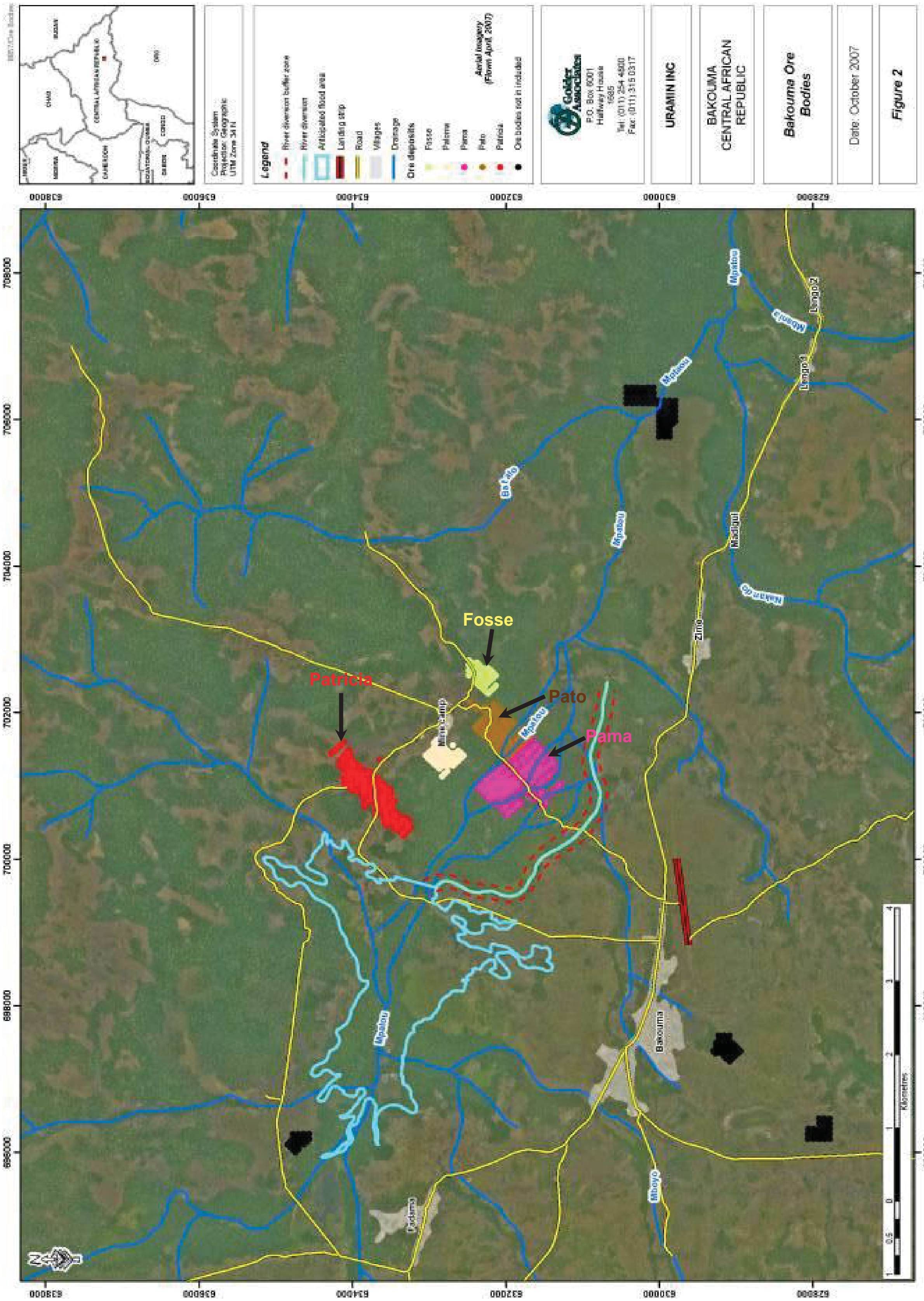


Illustration 7: Bakouma uranium deposits

1.3 Bakouma Uranium deposits

1.3.1 Location

The 3 uranium-containing ore bodies -Patricia, Pato-Pama and Fosse- are all situated approximately 5 kilometres away from Bakouma. Rivers are crossing the possible mining area (Illustration 7). During the rainy season, the area is partially flooded.

1.3.2 Geology of the deposits

A typical cross section of the Patricia deposit (Illustration 8) shows us that the first covering layer consists of laterite. The upper 10 to 15 m of overburden will mainly be mined with trucks and shovels.

The bigger part of the ore body is situated in saprolite rock. This saprolite has an average resistance of about 1 MPa. Partially, there will also be harder quartz inclusions. This part is the most interesting for the mining of the ore bodies.

Subsequently the saprolite is followed by a layer of saprock, and at the end, a layer of fresh rock. For open pits the overall slope angle was considered to be between 18° and 31°. The illustration was taken from the AMC Geotechnical Assessment of June 2008.

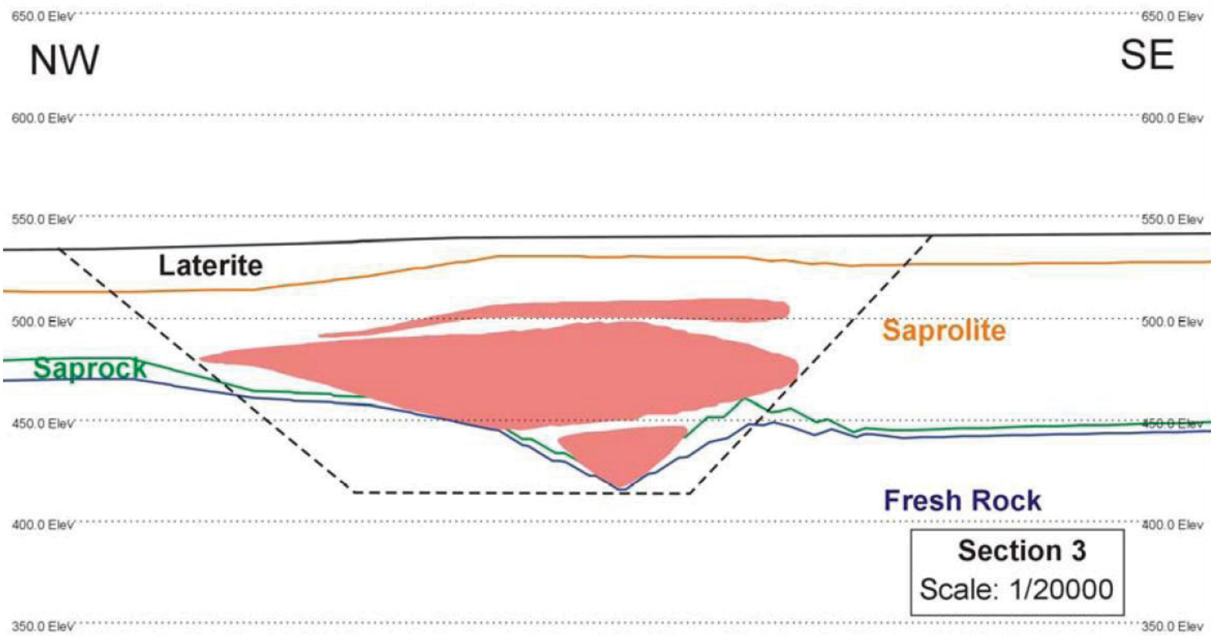


Illustration 8: Typical cross section of the Patricia deposit in Bakouma

1.3.3 Three deposits

The existing deposits were already illustrated as 3D models. The information was obtained through drill sounding.

1.3.3.1 Patricia

The Patricia ore body (Illustration 9) is the biggest and the richest of the three deposits. Its length extends to 1.25 km and its width to 350 m. Its depth goes down to 120 m (Illustration 10). At the thinner end the overburden reaches a sectional height of 45 m.

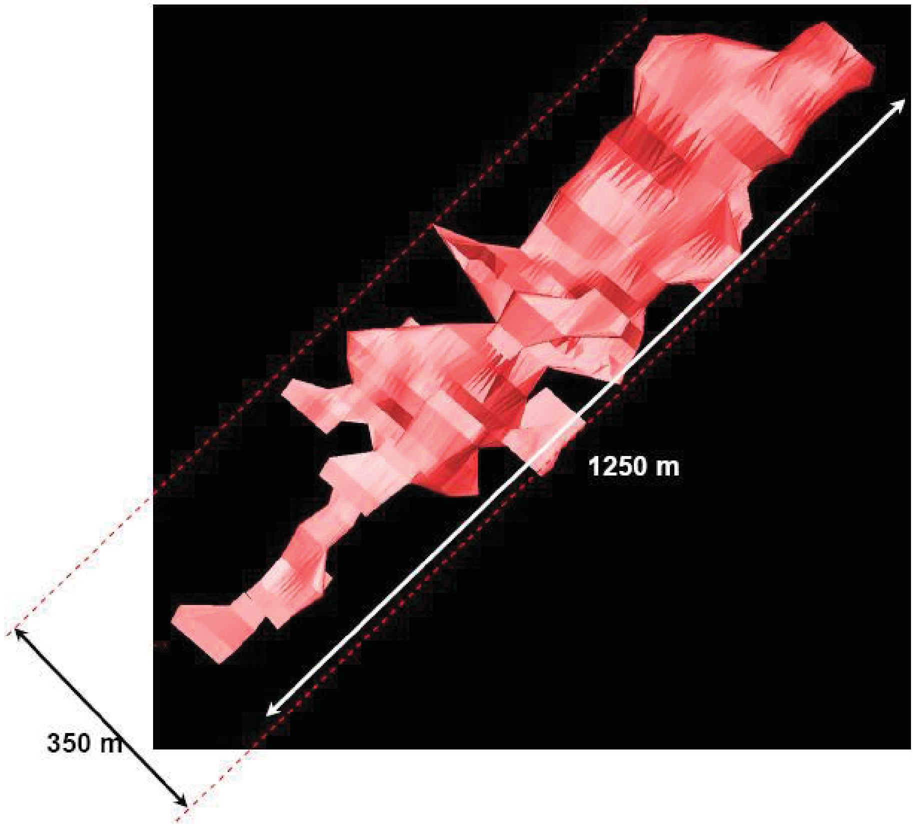


Illustration 9: Top view of the Patricia ore body

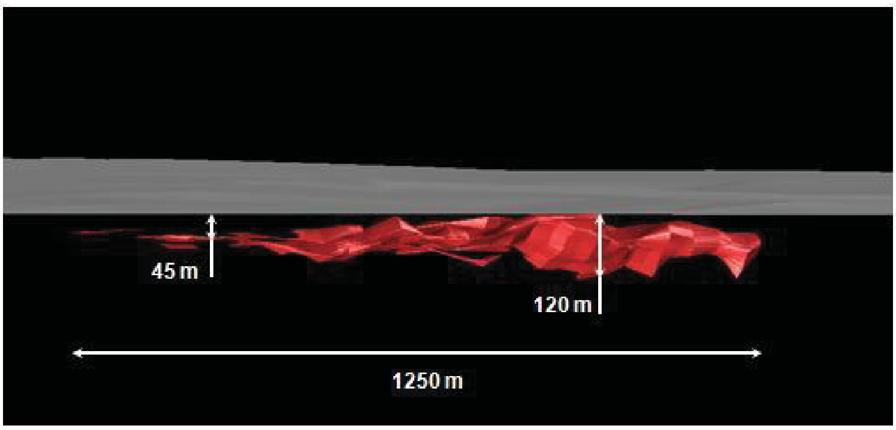


Illustration 10: Side view of the Patricia ore body

1.3.3.2 Pato-Pama

The Pato-Pama deposit (Illustration 11) covers two smaller uranium-containing ore bodies, which are very close together. The longer side is estimated to be 1325 km and its width on the thicker side reaches 785 m. The maximal depth of the deposits reaches 95 m. The biggest overburden is estimated to be 20 m (Illustration 12).

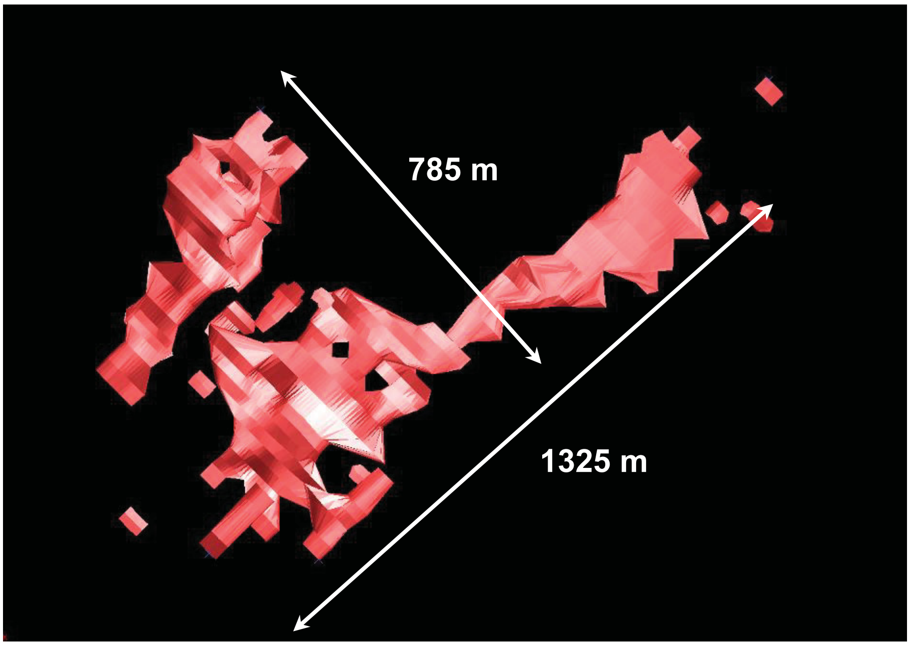


Illustration 11: Top view of the Pato-Pama ore bodies

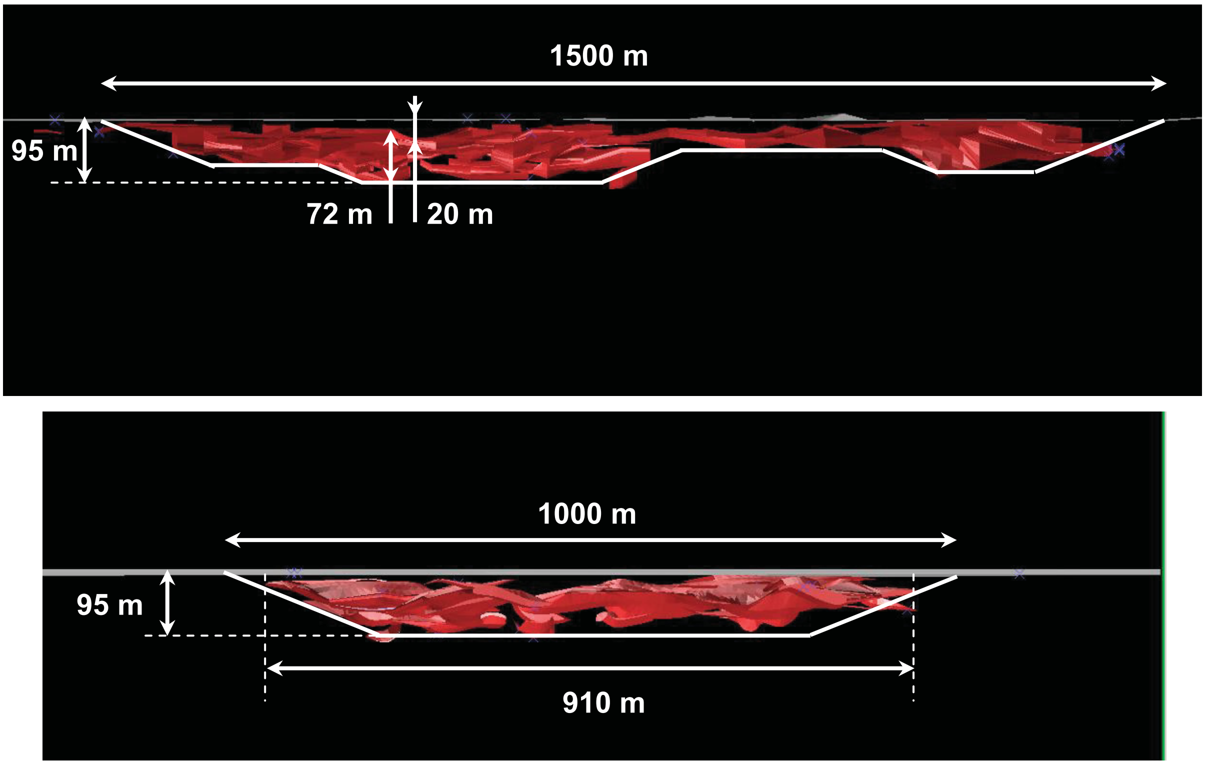


Illustration 12: Side view from Pato-Pama ore bodies

1.3.3.3 Fosse

The Fosse ore body (Illustration 13) is the smallest and deepest one. Its length is estimated to 348 m and its width 230 m. The overburden reaches heights from 28 to 72 m.

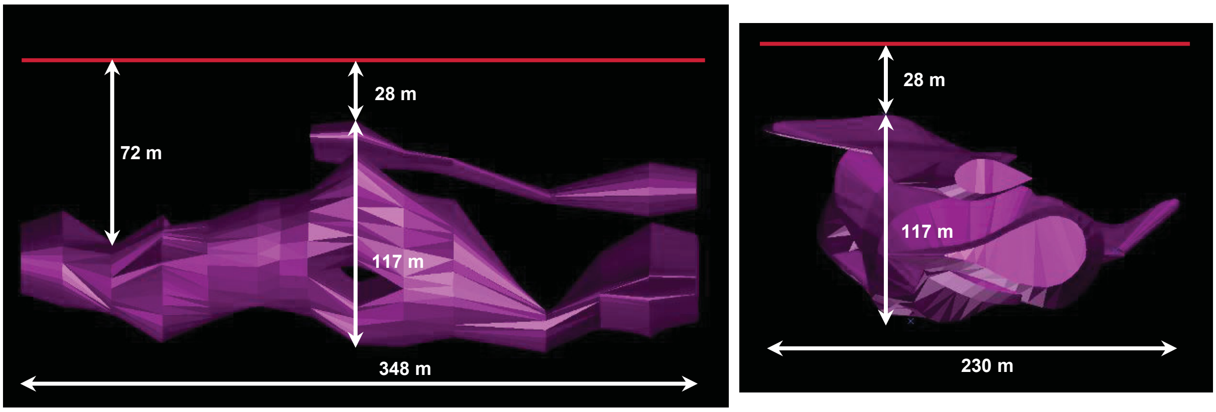


Illustration 13: Side view from Fosse ore body

1.4 Mining Project

The mining procedure will sometimes consist of a simultaneous mining of two mining sites. The starting point for a traditional mining site will probably be at the Patricia deposit, which is the highest located ore deposit and which will therefore not be flooded during the rainy season. The Pato-Pama and the Fosse deposits are at a lower altitude and may be flooded during the rainy season. Initiating mining at the Patricia deposit would allow for enough time to construct a dam. The diversion of the rivers would protect the Pato-Pama and Fosse mining sites from flood during the rainy season (Illustration 14).

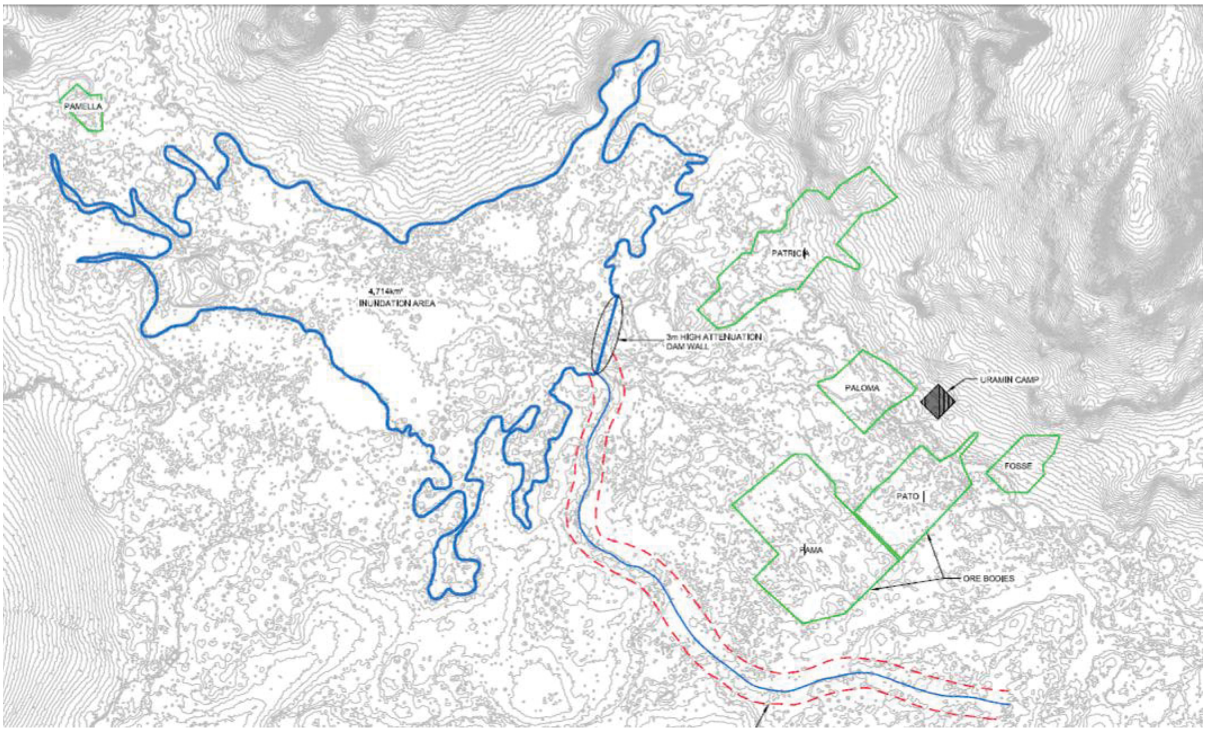


Illustration 14: 100 m wide stream diversion canal

A possible scheduling for a traditional mining site could be the following: starting point at the Patricia deposit and a stable mining output after 1 year of preparation; after the first 2 years of production, a simultaneous mining of the Pato-Pama deposits could start; after the fifth year of production the Fosse deposit could be mined simultaneously with Pato-Pama deposits. From the 11th year on, mining procedures could continue only at the Fosse deposit (Illustration 15).

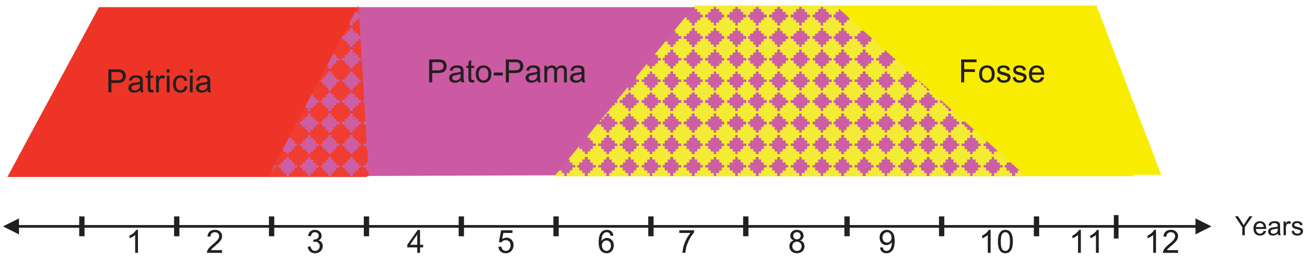


Illustration 15: Schematic mine planning

1.4.1 Production target

AMC gives in its report "Resources report for the Patricia, Pato, Pama and Fosse deposits – Bakouma" from November 2008 updated tonnages of the ore deposits and their proper grades. The total tonnage of material comes from the primary AMC assessment. These values give the baseline for following calculations (Table 1).

	Tonnage of ore deposits [Mt]	Grade of U_3O_8 [kg/t]	Metal U_3O_8 [kg/t]	Metal U (84,8%) [kg/t]	Ratio [waste/ore]	Total material [Mt]
Patricia	5,0	3,1	15,5	13,1	8,0	44,9
Pato-Pama	4,1	3,1	12,7	10,8	7,9	36,4
Fosse	3,2	3	9,6	8,1	14,4	49,2
Total	12,3	3,07	37,8	32,1	9,61	130,5

Table 1: AMC resource assessment

The estimated resources have a cut-off grade of 1 kg/t U_3O_8. The grade is 3 kg/t for the Fosse deposit and 3.1 kg/t for Patricia and Pato-Pama deposits. In total, $12.3*10^6$ t uranium ore are available.

The Taylor's formula from the MINING ENGINEERING HANDBOOK, SME; 1992, p. 406 allowed for the definition of mine life as:

$$P = 4.88\, T^{0.75}$$

P: Annual production rate in short tons
T: Short tons of diluted ore reserves T = $1.36*10^6$ st (=$12.3*10^6$ t)

The annual production rate results in $1.09*10^6$ st or $0.989*10^6$ t. The mine life with full production rate would therefore be 12.4 years. The metal recovery of the processing is 92.4%. This would result in an annual production of 2392 t metal U.

1.4.2 Long term planning

The criterion for a mining site in Bakouma is an annual production of 2000 t of uranium. The production rates have to be fixed for any further calculation or dimensioning for a mining scenario.

The average specific gravity of saprolite, which represents the main rock of the uranium deposits, was defined by AMC to be 2 t/m^3.

The efficient working hours per year were fixed to be 5040 h/y. It was assumed that mining is possible over the whole year in 3 shifts with one total production stop over 2 weeks for maintenance (Table 2).

ORGANISATION OF WORKING TIME AT THE MINE SITE		
Number of days per year	350	d/y
Number of shifts per day	3	p/d
Number of hours per post	8	h/p
Number of hours per year	8400	h/y
WORKING TIME FOR EQUIPMENTS		
Rate of availability of equipments	75%	
Total number of working hours per year	6300	h/y
Rate of production	80%	
Number of production hours per year	5040	h/y
Number of non productive hours per year	1260	h/y
Ratio non productive hours/productive hours	0,25	

Table 2: Working and production hours

Following the hourly needed production was defined.

Uranium grade in U_3O_8	0,848	
Average Ratio	9,6	
Density	2	t/m³
Objective of U production	2000	t/y
Metal recovery of processing	92,4	%
Needed production of U from the mine	2165	t/y
Needed production of U_3O_8	2552	t/y
Needed production of ore	830	kt/y
Waste to be mined	8,0	Mt/y
Efficient working hours	5040	h/y
Mining time	14,8	y
Total mining per year	8,81	Mt/y
Total mined ore per year	0,83	Mt/y
Total mined waste per year	7,98	Mt/y
Total needed production	1748	t/h
Mined ore	165	t/h
Mined waste	1583	t/h

Table 3: Production values for a dredging mining site

The objective of producing 2000 t U metal per year allowed calculating the hurly needed production of ore in the mine. Therefore a 92.4% metal recovery through processing was respected. The grade of U in U_3O_8 is 84.4%. The total needed ore production was calculated to be 165 t/h and the total waste production to 1583 t/h (Table 3).

1.4.3 Mining options

The local situation does not allow for a classical open pit in dry conditions without pumping great quantities of water. These quantities support values between 750 l/sec (at the beginning) and 3000 l/sec (at the end). For the purpose of comparison: the consumption of water in Paris is 6000 l/sec. Areva wished therefore to study an alternative mining option which avoids pumping too much water: dredging. At first, different options of dredging were consulted, especially two systems: dredging with boats and dredging with underwater mining vehicles. Both seemed to be feasible and quite interesting possibilities to mine the 3 uranium deposits in Bakouma.

2 Mining by dredging

2.1 Introduction to dredging techniques

Dredging is a mining technique, where the material is taken away by scraping. This can be done in dry and wet conditions. Two main possibilities of proceeding are available:

- the first and most common technique is **mechanical dredging**. This is usually done with draglines, which can be seen as big rope shovels. The difference is that they do not mine by pushing the bucket into the mineral, but they dredge the bucket over the material. Mechanical dredging can be realized in two different modes, which are the *classical mode* on one hand and the use of a *clamshell* on the other hand. The machines of this technique reach great working depths compared to machines used for classical hydraulic dredging. A positive point of this technique is the flexibility and accessibility of the machines, which work on the surface. The transport of the mined material must be done by trucks or band conveyors. Draglines can exploit the mineral in dry and wet conditions. In the case of underwater working, the wet material has to be drained in a silo. Following a second loading and hauling is demanded;

- the second technique is **hydraulic dredging**. This technique in the *classical* sense uses a vessel as main working side, which is equipped with a ladder. The maximum working depth of this technique, which goes down to approximately 30 m in standard conditions, is limited by the length of the ladder. A special technique of hydraulic dredging uses *underwater mining machines*, which work on the seafloor and transport the material to the surface through pipelines. These machines can work in depths down to 300 m. There are two main types available: the Tripod and the Crawler.
  Hydraulic dredging can be operated with a variety of drilling assemblies on the end of the ladder. These can be cutter heads, wheels or drag heads.

2.2 Dredging methodology for Bakouma deposits

The uranium deposits in Bakouma reach a maximal depth of 145 m. The maximal dredging depth for classical mechanical dredging is limited by the boom length of the draglines. There were no draglines that could perform in the required circumstances. Therefore this technique was eliminated.

Clamshell draglines which use cables instead of booms are not limited in working depth. Usually they are used for mining small volumes of sand in sites which are difficult to access. The capacity of the clamshells varies between 0.75 and 6 m^3. The productivity (30 to 500 m^3/h) decreases with working depth because of longer lifting phases. Also the correct position is harder to handle in great depth, which has a negative impact on the selectivity. This technique seems not to be the best choice for our mining site in Bakouma.

The richest parts of the deposits in Bakouma are in their final depths. Therefore there is a big interest to go down to it for avoiding losses. Hydraulic dredging with underwater mining vehicles, like the Tripod or the Crawler, is not limited to work in a depth of 145 m. Underwater mining vehicles allow also to work selective with a more or less constant productivity. Therefore, this seems to be an interesting solution for the mining.

Nevertheless underwater mining vehicles have higher capital costs and operating costs than other hydraulic mining devices. A good solution may be the combination of two hydraulic mining techniques, which combine the main goals in the best way: reach the needed depths and keep the costs to a minimum. This could be done by a Beaver Cutter Suction Dredger and a Crawler. A general organization of mining is described in chapter 2.3.5.

2.3 Equipment for hydraulic dredging

IHC Holland B.V. fabricates and supplies equipment for hydraulic dredging. Their website www.ihcholland.com/company/profile/ gives the following overview of the company:

"IHC Holland B.V. is the world's market leader in the design, fabrication and supply of equipment and services for the dredging and alluvial mining industries. The company has built up extensive know-how and experience through the fabrication of thousands of dredgers. The company serves 50% of the world market.

Design and fabrication take place either at one of the company's modern facilities in the Netherlands, or at local yards for cooperation contracts. Where local fabrication is required, the company provides both design and technical assistance, and delivers the components for the dredging installations and their control systems.

Design and fabrication of the dredger components and the control systems are undertaken at IHC Holland's own premises. The range of customer services also encompasses dredging consultancy, spare parts supply, service contracts, crew training, life cycle support and renovation of technically outdated dredgers. The company is unique in offering a complete service to the market. Clients all over the world - both private and governmental - are supplied with products and services geared to their individual requirements."

2.3.1 Beaver Cutter Suction Dredgers

Beaver dredgers (Illustration 16) present a very common form of hydraulic dredgers. They exist as cutter and wheel dredger models. Generally they consist of two-sided pontoons which are connected by coupling pontoons. Furthermore they have a ladder, two columns at the end of the vessel, and two anchors. An engine room is located at the deck. A single high-pressure submerged dredge pump is mounted on the ladder, which is driven by a diesel engine. Normally, they are used offshore.

Illustration 16: IHC Beaver Cutter Suction Dredger 6518 C

The ladder itself can only make vertical movements. The horizontal movement for the mining process has to be done by the whole boat, through a fixed rotation axis. The rotation axis is created through the main column at the end of the boat, which is rammed in the seafloor. To allow a horizontal moving process, anchors have to be placed at the front of the boat. In offshore conditions, these anchors are placed with support arms on the left and right side of the boat. In our special case in Bakouma, it would be possible to install the anchors on the surface, close to the pit by using loaders. The ladder is in direct connection with the cables of the anchors, which is itself connected with a winch on the vessel. Depending on at what side the cable is tightened through the winch, the whole boat has to do a horizontal movement (Illustration 17).

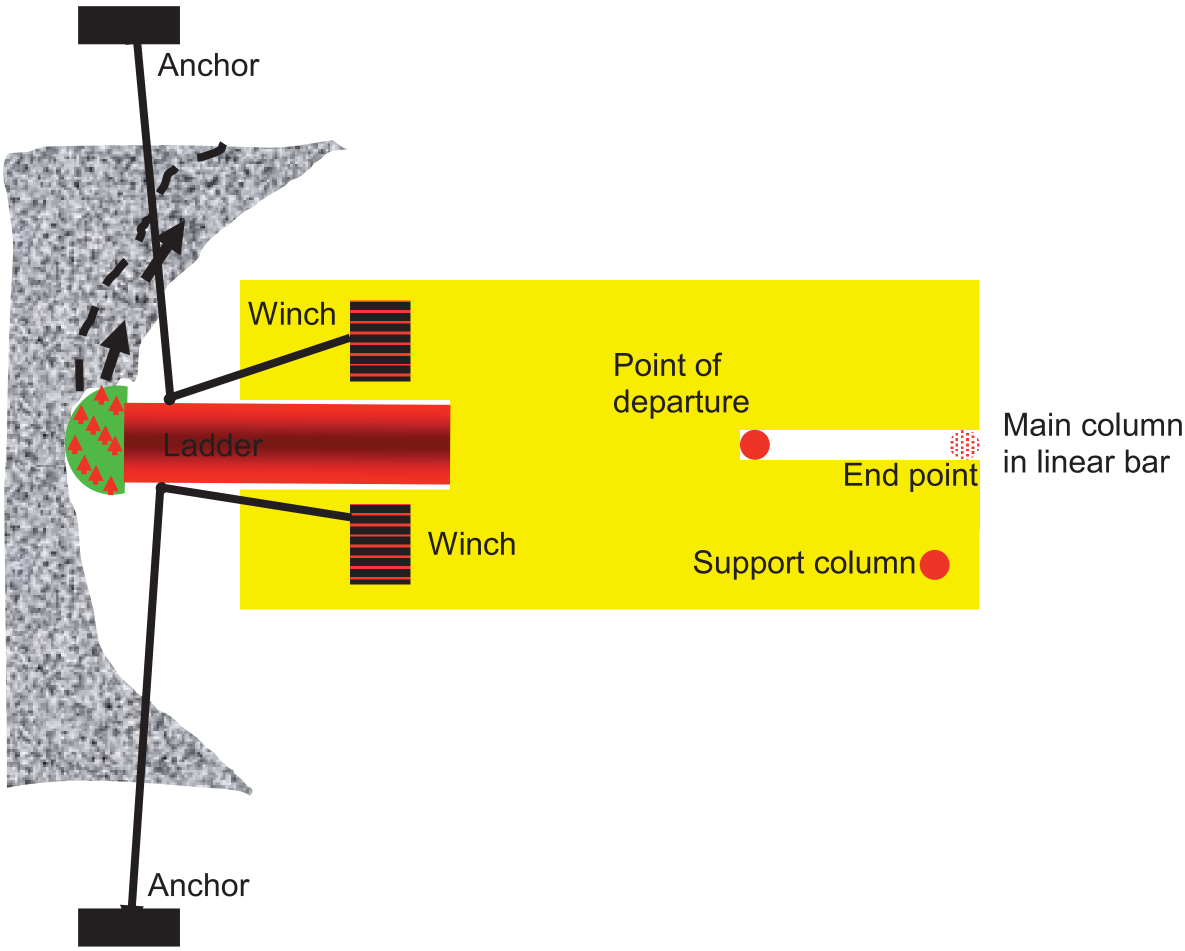


Illustration 17: Principe of a Beaver Cutter Suction Dredger

During the exploitation, after every cut, the vessel has also to make a forward movement. The boat itself cannot do this movement. The only way is to do steps with the columns. The main column, which also represents the rotation axis, is situated vertically in the end of the boat in the middle of its width in a linear bar. During the mining process, the boat has to take a step forward for each cut. Without lifting the column, it has a linear play of approximately 4 to 6 m. When the main column is situated on the end of this bar, it has to be switched to its point of departure. To avoid an uncontrolled movement of the vessel during this switching process, a second column is available. This second column is rammed in the seafloor before lifting the main column. When the main column is on its point of departure and again rammed in the seafloor, the second column can be lifted and another cycle of mining can start (Illustration 18).

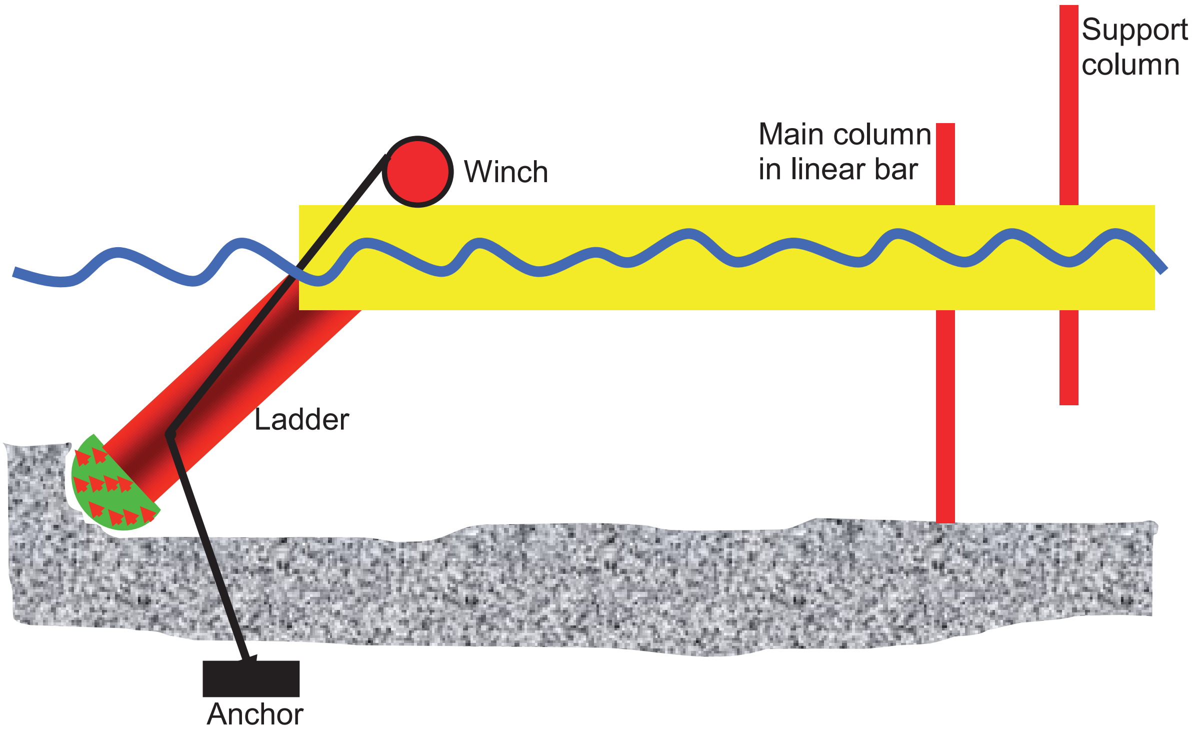


Illustration 18: Side view of Beaver Cutter Suction Dredger

Beaver Cutter Suction Dredgers have been used in different locations worldwide. The BCSD 6518 C (Illustration 16) had been used in 2003 in Great Britain, in 2004 in Indonesia, in 2007 at the Maldives and in 2008 in Korea. The BCSD 6525 C had been used in 2007 in Germany and Angola, and in 2008 in Nigeria and India. Both were used for civil constructions. The difference of the two dredgers is their maximum dredging depth which is 18 m for the BCSD 6518 and 25 m for the BCSD 6525.

2.3.1.1 IHC Beaver 6518 C

An interesting model for our mining site in Bakouma is the IHC Beaver 6518 C Cutter Suction Dredger. Its size is 32.5*12.44*2.97 m^3 (l*w*d) rather 47.2*12.44*2.97 m^3 (l*w*d) for a raised ladder. Its total weight is 382 tons and its total installed power is 2700 kW. The dredger is equipped with a cutter and its maximal dredging depth is 18 meters. The delivery pipeline has a diameter of 650 mm and the dredging pump has a power of 1571 kW at the shaft.

The cutter head has a diameter of 2.38 m with a power at the shaft of 585 kW and a maximal speed of 30 revolutions per minute.

The spuds for the forward and horizontal moving process have a length of 23.4 m, a diameter of 0.9 meters and a weight of 13.127 t.

The swing winches have a line pull of 240 kN with a maximal line speed of 20 m per minute. The diameter of their drum is 0.762 m. The used anchors have a weight of 1.2 t. The Beaver dredger can reach a swing width of 41 m for the maximal dredging depth, rather 51.5 m for the minimal dredging depth (with 35° of swing on each side).

2.3.1.2 IHC Beaver 6525 C

The IHC Beaver 6518 C dredger offers a standard option to extend the ladder, to increase the working depth to 25 m. The description of the model would then become an IHC Beaver 6525 C.

The floating platform and its installations in general would stay the same. The length of the columns has to be increased as well as the swing winches which would need more power and longer cables. An additional booster pump unit has to be installed.

Using a longer ladder has not only an impact on the working depth, but also on the swing width. An easy calculation shows that the swing line is approximately 71.5 m in the highest position of the ladder (35° swing width on each side).

The production capacity of an IHC 6525 Beaver is 1860 t/h of solids (defined from IHC).

2.3.2 Underwater Mining Vehicles (UMV)

Underwater mining vehicles can mine material underwater down to depths of 300 m. These excavation devices can move independently on the seafloor. They are connected with the surface through support vessels or support pontoons. The material which is excavated on the ground is transported through pipelines to the support facilities on the surface.

IHC offers two main options: the Tripod and the Crawler. The difference of the two machines is their moving mode. While the Crawler is a track moved vehicle, the Tripod has a moving mode on three feet. The Tripod is a completely new system and has never been used before. In comparison to that is the Crawler a known and already proven system. It was used already for diamond mining near Namibia in offshore conditions at a depth of 150 m. Further IHC was involved for crawler design for SMS deposits near Papua New Guinea and New Zealand for mining Nautilus and Neptune minerals.

2.3.2.1 Tripod

IHC gives a general description on the Tripod: the Tripod or the TWP (Triangular Walking Platform) gives a new means for accurate and efficient dredging in deep water. That means this platform, which has a lateral length of 10 m and a net weight in air of 30 t, can reach working depths down to 300 m. The basic setup composes 2 triangular frames, 1 drilling unit, 5 hydraulic cylinders and 2 pumps working in series. The upper frame uses 3 spud cylinders for the moving process on the vertical plane. The lower frame is fitted with fixed feet. Because of possible translational and rotational movements between the frames in the horizontal plane, the Tripod is able to walk in all directions. The machine can be equipped with a range of excavation tools. The ladder has a length of 8 m, which allows full rotation of the frame within the width of the dredged lane. The machine walks in its dredged lane, therefore it is sometimes necessary to step down into a dredged lane. The concept of the spud cylinders is done in a way that they can be adjusted independently, which allows for moving around on an uneven sea floor (Illustration 19).

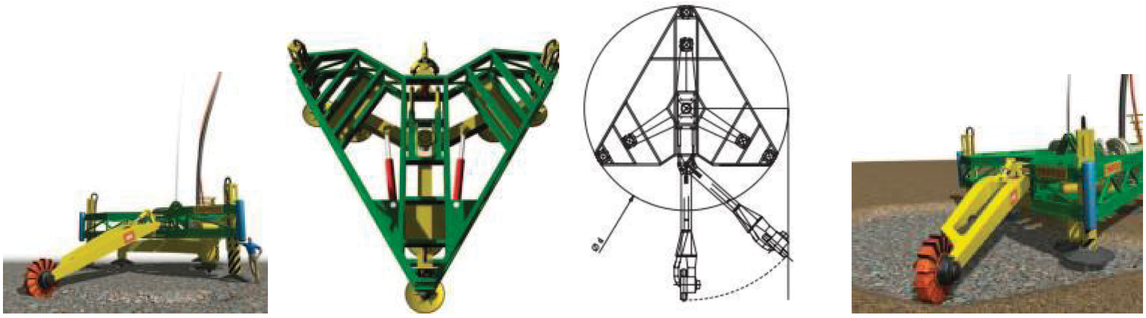


Illustration 19: Tripod

The Tripod is a completely new system. Until today this system exists only on drawing tables. Therefore there has been no actual usage that could prove the functionality of the system. The aim of its development was to define a completely new moving mode. We can see that the Tripod is an interesting and up-and-coming system.

Nevertheless it has to be improved and examined for real functionality. Even when the 3 spud cylinders for the moving mode can be used independently, there is a limit of stepping down or up in different layers, because you always need at least one of the two frames in a stable position. This is not the case when the Tripod is at the position of a step: here it has to do a step in a different layer and it has to do a movement forward at the same time. The fixed feet of the lower frame do not allow an adjustment between different heights. So this frame has to be positioned always in one level.

Further it is not certain that the Tripod is able to walk in inclined layers. The spud cylinders may not be able to be used in the same way as in horizontal positions. Further the moving mode may reach its limits.

For a mining operation it is better to use equipment that is not complicated and that do not have many moving parts to keep failures to a minimum. Non productive time has to be avoided. Further for our case in Bakouma, we will find very weak soil properties (Saprolite of 1MPa). This could be a handicap for the spud cylinders, as they risk sinking into the seafloor.

2.3.2.2 Crawler

The Crawler is a tracked vehicle, which was already used in different mining sites worldwide, like diamond mining near Namibia. Normally the underwater working device is used offshore and therefore connected to a support vessel (Illustration 20).

Illustration 20: Crawler

For our case in Bakouma this machine seems to be very interesting for mining down to 145 m. Nevertheless the system, which was often used in offshore conditions, has to be adapted to our mining site in Bakouma. Particularly surface devices and the possibility to move between the different mining layers need to be examined (see chapter 2.3.4.2).

For our case the Crawler is an approximately 65 t (in air) machine. The mining of the deposits in Bakouma will sometimes be done simultaneously, but there will never be a simultaneous mining of more than 2 mining sites. One single crawler has a production capacity of 612 t/h (defined by IHC). For the needed total production three crawlers would be necessary.

IHC proposed a system which is composed of two crawlers that are both connected with the same surface device. The underwater mining vehicles are electrically driven (1700 kW power generator for each) and they are operated from the surface. On the surface device two booster pump units (750 kW) have to be installed (one for each UMV) to ensure the transport of the slurry.

The Crawler is a tracked moved vehicle, which can reach a speed up to 1.5 km/h. In good soil conditions it can walk in all directions and it can even turn while standing.

The tracks have a dimension of 1.5*6 m^2 each. That results in a ground pressure of 3.3 t/m^2.

The dimensions of the Crawler are 9.2*5*4.5 m^3 (l*w*h). The UMVs will always be connected with the surface device through a cable (300 m). Therefore it is possible to lift them (with 3-4 m/min) at any time and to displace them (with 20 m/min) quickly anywhere in the pit (see chapter 2.3.4.2). Due to this system of displacement and lifting, there is no need to construct a ramp.

The frame of the crawler will always rest in the same position versus the tracks. That means the machine has to be in an even position versus the mining front for doing a cut. The excavation process is done punctually. The boom can make vertical and horizontal movements. For the first cut the boom has to be placed in a high position near one of the upper corners of the front wall. Then it will do a horizontal movement to the side to mine the whole width of the front wall. When the boom reaches the end of the wall, it will go down for the height of the cutter head diameter and do the next cut in the other direction. The crawler is able to reach heights up to 5 m and can do an undercut of 1 m. The crawler has a mining width of 7 m (Illustration 21).

In the best case scenario it would be possible to mine a wall of 8 m high. That means even when the cutter does not reach the total height of the wall it will start to cut the material as high as possible and the rest will break down itself. All the material that breaks down during a cutting cycle will be sucked up afterwards, when the boom is in lower positions.

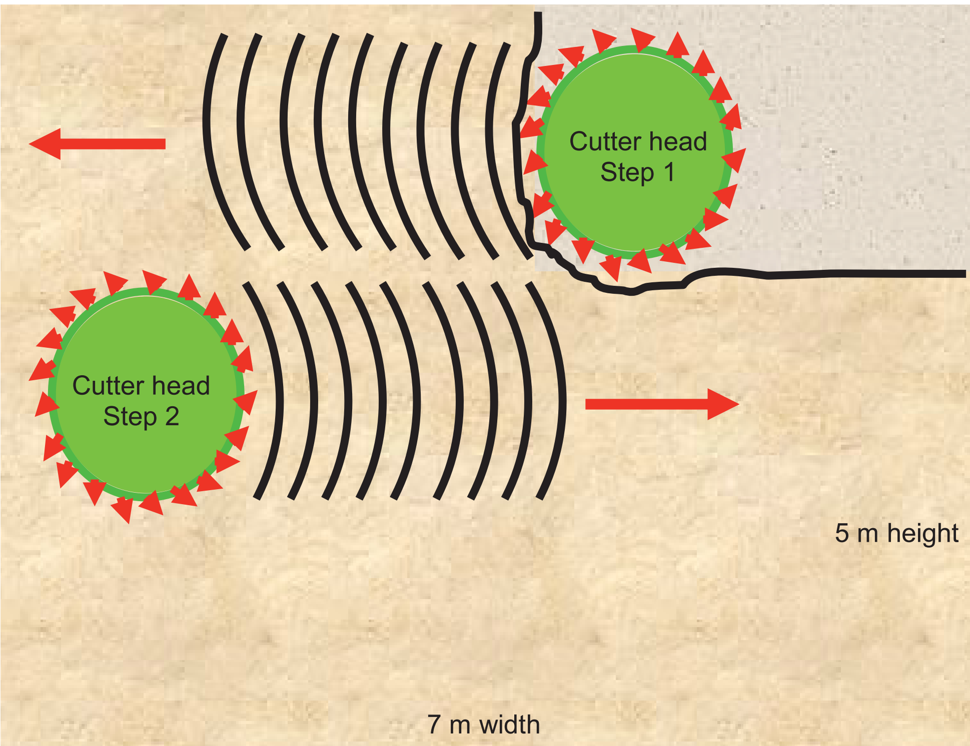


Illustration 21: Production cycle

Each Crawler is equipped with a hydraulic-driven pump with a shaft power of 700 kW. The pipe diameter is 500 mm. In higher positions the slurry can have a solid content of 30%. In deeper positions this drops to 20% because of the longer transport distances. The slurry is transported through a flexible transport pipe to the surface device. This pipe consists of 36 sections with 11.8 m each (= 424.8 m in total). The length will be adapted with the increasing working depth. From the surface device, further transport of the mined material is done in a similar way as with the BCSD. The whole pipeline is 2.42 km long, which includes the 424.8 m underwater pipeline, the 800 m floating line plus the 1200 m shore line.

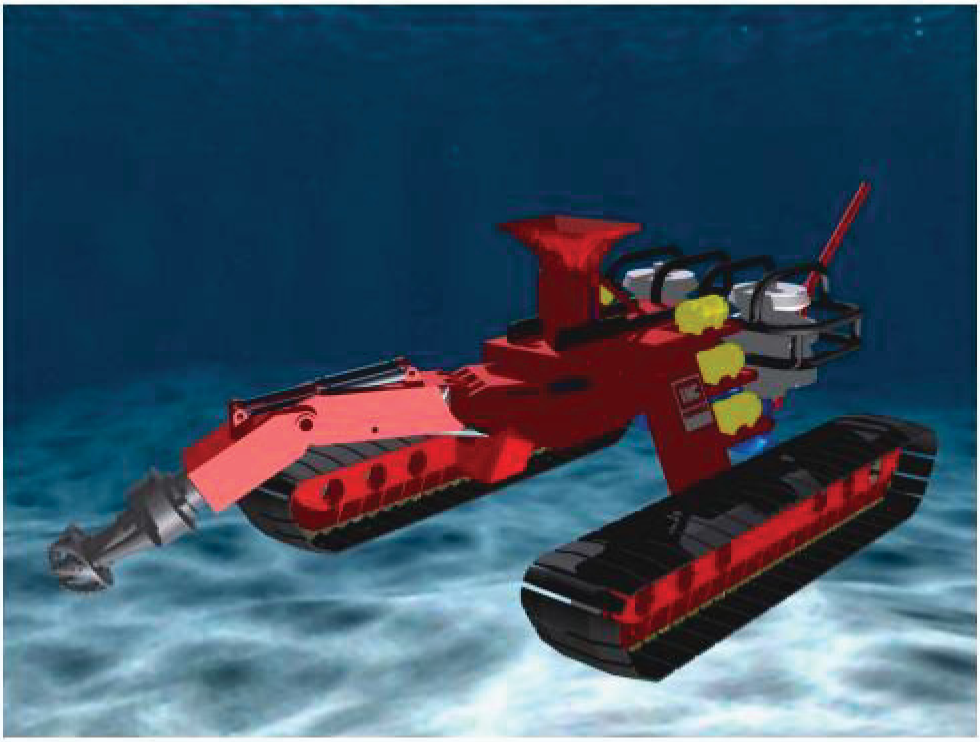


Illustration 22: Model of the Crawler

Compared to the Tripod the Crawler (Illustration 22) seems to be the better choice for our mining site in Bakouma. Firstly because it is a proven and known system which has been already used in different mining sites, and secondly because it appears to be very robust. Its moving mode with tracks is better for the existing situation (poor soil properties of 1 MPa resistance).

2.3.3 Drilling heads

IHC Holland B.V. supplies not only the needed machines, but also the necessary drilling equipments for them. The drilling units of hydraulic dredgers can vary in big dimensions. Not only must their size be adapted to the existing conditions, but also the type of the drilling head and its cutting device as well. Common and proved units are cutter heads, wheel heads, or drag heads. Depending on soil properties and the used type of machine, each system has to be determined concerning its advantages and disadvantages.

2.3.3.1 Cutter head

Cutters can have either teeth or cutting edges. In any case of maintenance work on the cutter heads the UMV/the ladder has to be lifted for having access. To save time it will probably be useful to have more cutting heads available. So the whole cutting head can be changed and the maintenance work can be fulfilled afterwards without interrupting the production.

2.3.3.1.1 Cutters with teeth

These cutters can be fitted with a variety of teeth and replaceable cutting edges in widely varying dimensions (Illustration 23). For stiffer and harder rock, narrower chisels will be selected. The teeth are fitted in adapters and can be replaced easily (Illustration 24). In the event of changing work conditions the change of the teeth is easily possible.

Illustration 23: Cutter head with teeth

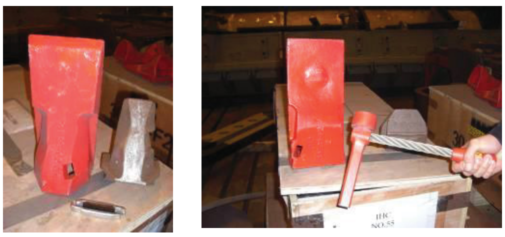


Illustration 24: Adapter and teeth

2.3.3.1.2 Cutters with cutting edges

Cutters with cutting edges exist in two different forms: the plain and the serrated form (Illustration 25). The latter can realize a higher penetration effect. Big dimensions of these cutters can reach a height of 2 m and a diameter of 3.2 m. The diameter of the suction can vary between 0.2 and 1 m for soils of lower resistance or rather between 0.3 to 0.9 m for soils of higher resistances.

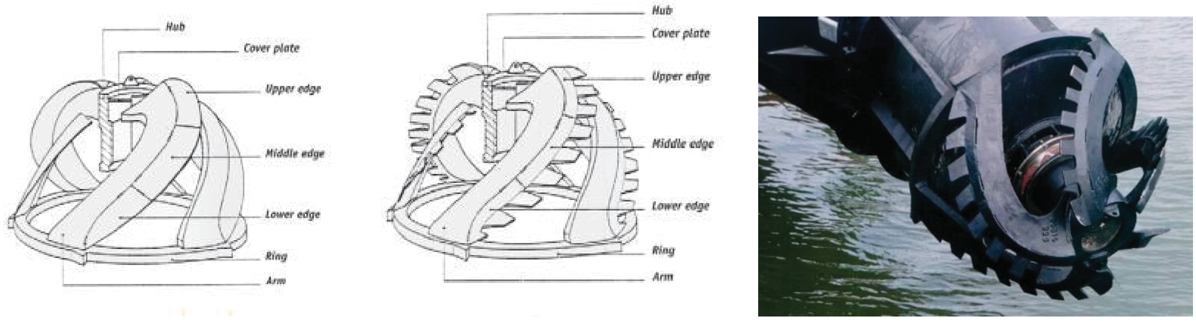


Illustration 25: Cutter with cutting edges

2.3.3.2 Wheel head

Wheel heads promise good cutting properties, a constant dredging output and a high production. Wheel heads are configured of a hub and a ring, which are connected by bottomless buckets. The lip of the suction mouth penetrates into these bottomless buckets and prevents thereby their clogging. The buckets, the lip and the suction mouth are orientated on the same plane. Wheel heads have a low sensitivity to debris, like tree stumps or rocks. The mixture density is very high and the spillage is very low. The buckets can be fitted with smooth-cutting edges or with replaceable teeth (Illustration 26). Draglines with wheel heads offer equal production in both directions of swing and they can realize an upward or downward cutting.

Illustration 26: Wheel heads

2.3.3.3 Drag head

A drag head (Illustration 27) can only be used in combination with a vessel. It activates a linear translation of the whole boat. Therefore, anchors and columns are not needed for the movement of the vessel. The most important mining parameters are the width of the visor, the penetration depth, and the tailing speed. A modern visor can do a movement up to 50° and jet nozzles and teeth are used for an efficient loosening and fluidizing. In the interior a drag head is hollow. During work time a vacuum is produced, which can be applied to transport the mixture of water and solid. The fixed part of the drag head is connected with the suction pipe. For low soil resistances the diameter of the pipe is 0.3 to 1.4 m, or 0.4 to 1.2 m for higher soil resistances. The density of the mixture is high and the resistance to flow is minimized.

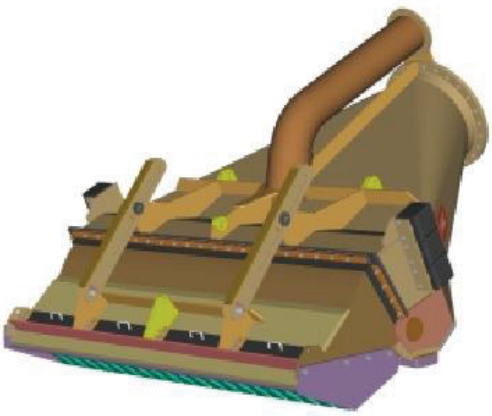


Illustration 27: Drag head

2.3.3.4 Selection of a drilling head

In Bakouma, we estimate the Saprolite rock to have a cutting resistance of 1MPa for the main mining by dredging. Furthermore, we have to assume the risk of harder inclusions (quartz). This geology requires a cutter head with teeth. IHC proposed standard cutting heads with a diameter of 1.45 m and a length of 1.83 m for the UMVs and a cutting head with a diameter of 2.38 m for the BCSD. The cutting power is 150 kW for the smaller one and 450 kW for the cutting head of the Beaver Dredger. The teeth of the cutter heads are fixed on adaptors with grips. In case of abrasion they can be changed easily. In the case of changing conditions the whole cutter head can be replaced within approximately 12 h.

The boom of the UMV and also the ladder of the BCSD can also be equipped with a wheel. A wheel drives the same way in both directions and there is also less side force needed. Nevertheless a wheel is not an option in case of harder rock pieces or harder inclusions.

A boom could also be dimensioned for changing between cutter heads and wheels.

2.3.4 Surface devices

2.3.4.1 Transport systems

All of the mined material will be sucked up directly. The pump is directly installed on the mining device. The capacity of the pump has to be adjusted with the cutting rates of the machine. For lower depths the pumps will be able to pump slurries with 30% of solid. In great depths (down to 145 m) the content of solid decreases to 20%, because of the longer pumping distances.

The flow diameter of the pipeline is 0.5 m for the UMV and 0.65 m for the BCSD. The density of the mixture (water plus solid) was calculated to 1150 kg/m^3. The floating pipeline has a length of 800 m and it will float on the water with floating parts. An installation on the surface is required to adopt the needed length of the pipeline with the advancement of the mining development. The shore line has a length of 1200 m. IHC mentioned that the pipelines have to be changed once a year. The length of the pipeline will increase with the development of the mine. That adjustment of the length will be done in the period of renewing the pipeline.

The pipelines must also be able to differ between waste material and ore. This means a split in the pipe is needed. A radioactivity measurement tool (Illustration 28) must be installed on the pipeline, as close as possible to the pit to differentiate between waste and ore. IHC has experimented to measure the radioactivity in the pipes. A critical point could be the definition of a grade which separates between ore and waste as well as defining a measure frequency. If this technique is not possible, different dredgers have to be used, either for mining ore or waste.

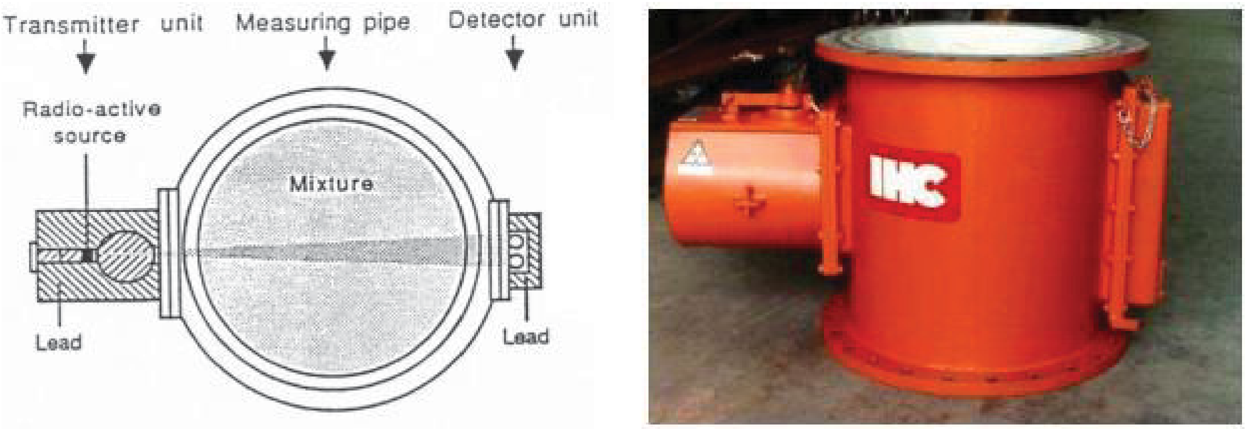


Illustration 28: Radioactivity measuring device

The distance in bringing the waste to a stockpile must be minimized, to minimize the energy consumptions of the pumps. Therefore, the stockpile has to be placed in the middle of the three existing deposits. That will make for a distance of about 1.2 km. The ore will be transported directly through pipelines to the processing plant.

2.3.4.2 UMV Surface Pontoon

In offshore conditions, underwater mining machines are connected to the surface through a support vessel. This support vessel could be replaced by a UMV surface-pontoon for our case in Bakouma. Such a pontoon was never used before, but its individual elements are well known and proven technologies. In general the pontoon is made of a floating platform with an opening in the middle, whose size is adapted to the size of the UMV. Over this opening a frame with a winch is installed. The UMV is connected through a cable to this winch on the pontoon (Illustration 29). Therefore it is possible to lift the UMV, for example, for maintenance.

The UMV pontoon cannot move itself, but it is displaced through four hoisting winches that are installed on each corner. The cable endings of these winches are fixed on the surface through anchors. The anchors on the surface can be moved with loaders. Therefore, it is possible to move the pontoon to/from any position on the water surface and it is also possible to move the UMV anywhere.

During production of the UMV, the connecting cable between the pontoon and the UMV is not tightened, but kept sufficiently short to avoid entanglement.

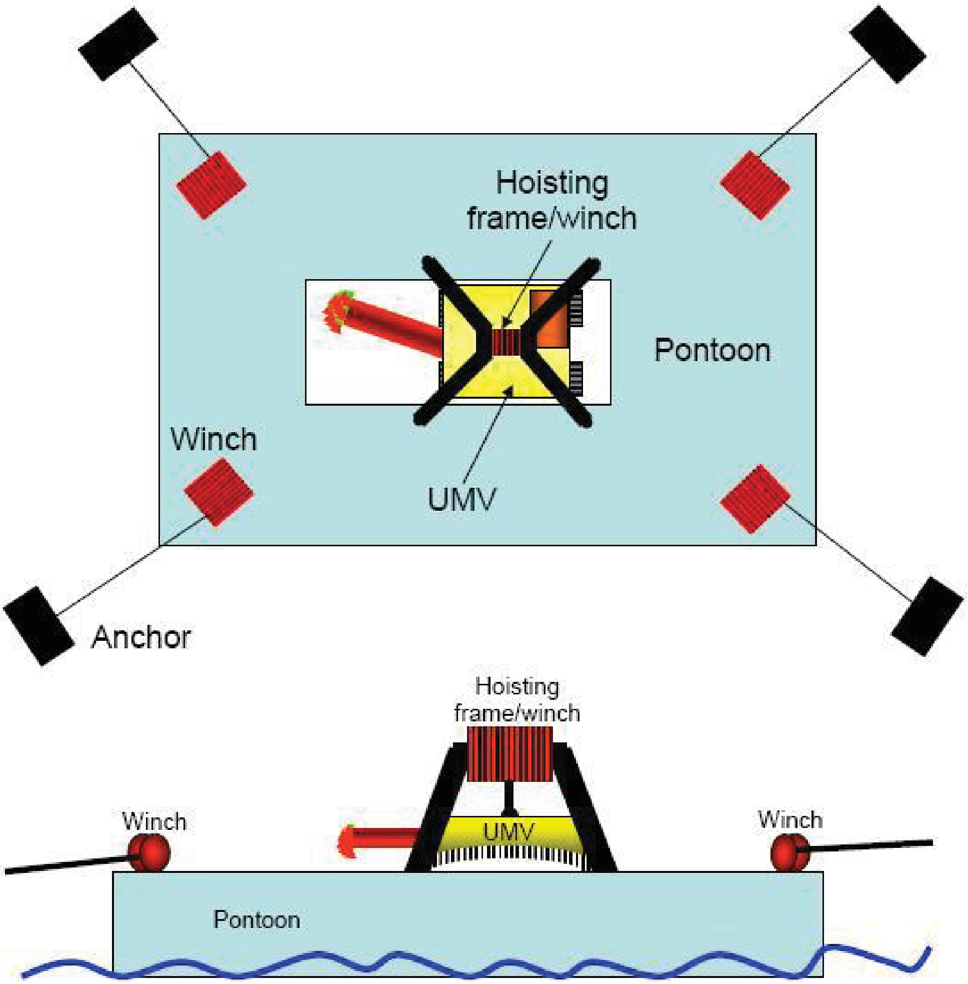


Illustration 29: UMV Surface Pontoon

IHC offered a possibility to use a pontoon that can attend to two UMVs at the same time. This UMV pontoon consists of two separated hoisting portals (one for each UMV). The main pontoons of the surface device have a dimension of 28*4*4 m^3 (l*w*h) and a weight of 160 t. The utility pontoons, which are used for placing the UMV to do maintenance work, have a dimension of 12*8*1.8 m^3 (l*w*h) and a weight of 20 t. The two hoisting portals are equipped with hoisting winches, which have a capacity of 80 t and a 200 m long wire with a diameter of 60 mm (Illustration 30).

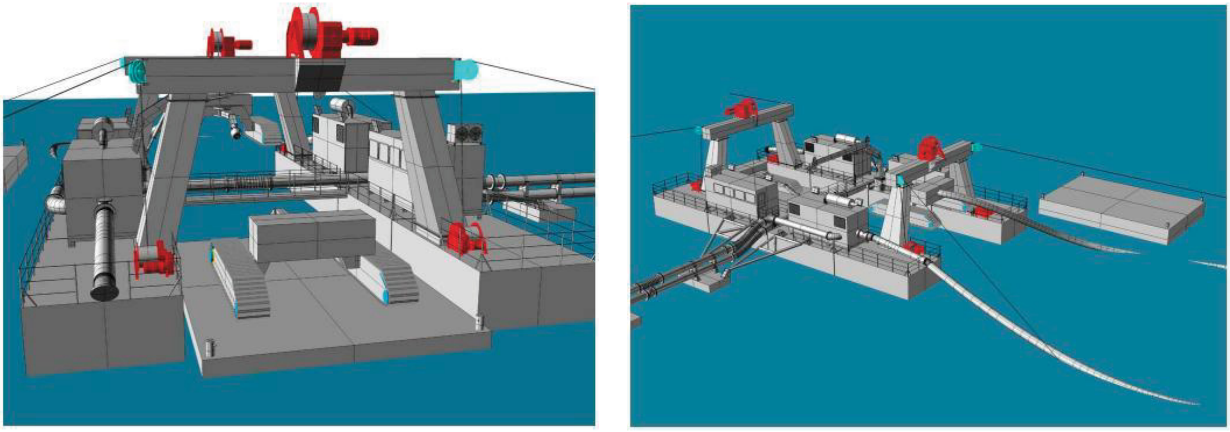


Illustration 30: Proposed UMV pontoon from IHC

Because of a temporarily simultaneous mining of the deposits it must be considered that a pontoon which coaches two UMV is always constrained to work in the same pit. To avoid this situation it may be useful to use separated systems (one pontoon for each UMV). Otherwise, using one single pontoon that coaches two UMV offers the advantage to reduce needed anchor installations on the surface.

2.3.4.3 Access to BCSD and UMV pontoon

In order to allow the access to the BCSD and to the UMV pontoon, a support vessel will be needed. This support vessel is mainly inevitable to bring spare parts to the workplace. For persons it may be possible to install a floating bridge on the floating pipeline parts.

2.3.5 General organization of dredging mining of a deposit

For dredging, three main scenarios are available: a), b) & c). All scenarios include classical mining for the first 15 m in dry conditions, with trucks and shovels. This 15 m result from the fact that the ground water table is at approximately 10 m under the surface and that dredging equipments require a minimum water depth of 5 m for their installation. The minimum water depth of 5 m has to be respected over the whole mining process, to guarantee the functionality of the boats and for assuring that the pumps of the UMVs stay under water. For a dredging mining scenario the equipments could be used in the following order:

a) - Beaver Cutter Suction Dredger for mining the following 20 m (5 m of water)
   - Underwater Mining Vehicle for everything deeper than the maximum range of the Beaver dredger;

b) - Only Underwater Mining Vehicles for everything deeper than 15 m;

c) - Only Beaver Cutter Suction Dredgers, with an adjustment of the water table.

Scenario a) seems to be very promising, because of the known and often proven technology with BCSDs. They offer the lowest possible dredging costs (see chapter 3.6.1) and could be used for mining higher and not so rich layers. Furthermore, the utilization of the UVMs allows reaching deeper and richer ore zones (Illustration 31).

Scenario b) uses only UMVs. Its drawbacks are the higher capital costs and operating costs (see chapter 3.6.2) and the mainly unknown and not proven method itself.

Scenario c) is very promising concerning capital and operating costs. Its main difficulty could be a stable adjustment of the water table, which has to be assured over the whole mining time.

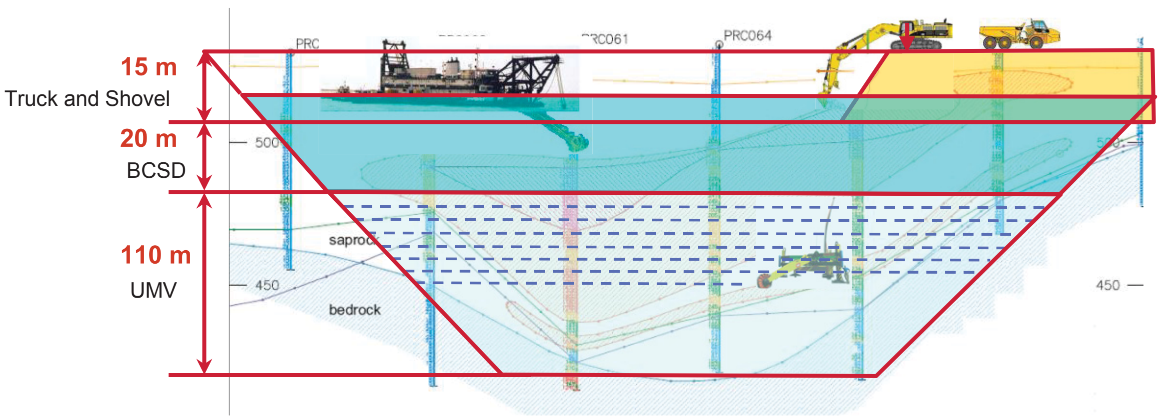


Illustration 31: Mining scenario a)

2.4 Conclusions

In the case of Bakouma, mechanical dredging does not seem to be the best choice because of the limitation of the working depth.

An underwater dredging device like a Crawler seems to be not limited to the working depths in Bakouma. Nevertheless, the more promising solution seems to be a combination of two dredging systems: a BCSD for the primary mining down to approximately 20 m, and an underground mining device like the Crawler for further mining down to the maximum depth. The combination of the two dredging methods is possible because of the similar mining equipment and surface devices like the transport unities. The combination makes the most out of the main potency of each system, which is a lower dredging cost per t for the BCSD and a working depth that is not limited for the UMV.

The utilization of beaver cutter suction dredgers with ladders, for realizing hydraulic dredging, has a limited working depth. However, the water level in the pit could be adjusted with the beavers itself, when taking into account that the product is slurry with 20 to 30% of solids. This water-level-adjustment would respect the limited working height of the BCSDs and allow reaching deeper zones of the deposits only with beavers. Because of the lowest possible dredging costs with BCSDs the solution with an adjustment of the water level seems to be very promising.

Underwater mined material -independent if it is ore or waste- has to be drained on the surface and the residual water has to be cleaned before dumping it into nature. To avoid transporting too much slurry for long distances in pipelines, selectivity is a critical point. Mining could be done selectively when using different dredging machines for mining ore or waste slurry. Therefore the distinction between ore and waste has to be done underwater. If this distinction underwater cannot be handled, the division has to be done in the pipelines on the surface. IHC has already done experiments to measure the radioactivity in pipelines.

The fact of working underwater implies working blind. Nevertheless there are promising systems that can make animated virtual reality images for the machine driver. These techniques can produce an image of the whole underwater pit, not only to correctly position the mining machine, but also to control the mining development and to work selectively. Therefore it is necessary t have a good knowledge about the pit.

In essence, hydraulic dredging is not only a possible option for the mining of uranium deposits in Bakouma, but also demonstrates a great deal of promise in this mining technique.

3 Mine planning

For the dredging scenario the dredgers will produce slurry with 20 to 30% of solids. The volume of the slurry that has to be dredged will depend on its solid percentage (Table 4).

Slurry	**20**	**%**	**30**	**%**
Ore	165	t/h	165	t/h
Liquid	659	t/h	384	t/h
Total mineral slurry	823,76	t/h	549,17	t/h
Waste	1583	t/h	1583	t/h
Liquid	6332	t/h	3694	t/h
Total waste slurry	7914,85	t/h	5276,57	t/h
TOTAL	**8738,61**	**t/h**	**5825,74**	**t/h**

Table 4: Slurry production rates

Three possible mining scenarios were already described in chapter 2.3.5 and are all possible from a technical view. Nevertheless the capital and operating costs (see chapter 5) vary greatly for each scenario, which comes from different needed equipment. Mining with UMVs is a lot more expensive than mining with BCSDs. Therefore the mining scenario only with UMVs has to be eliminated. The scenario with BCSD and UMVs also has high mining costs in comparison to the mining scenario only with BCSD, with an adjustment of the water table. Therefore, the scenario which combines BCSDs and UMVs was also eliminated.

For the sake of completeness, UMVs and BCSDs were both included in a first mining organization. A further detailed mining plan was only done for BCSDs (scenario c from chapter 2.3.5).

3.1 Classical mining

Classical mining will be used for the mining of the first 15 m of overburden. This will be done in two steps (2 * 7.5 m). When a sufficient surface is mined and the minimal water depth of 5 m is reached, work for the installation of the BCSD can be done. The volume to be mined by traditional mining methods is approximately ¼ of the total volume.

3.2 Mining with BCSD 6525 C

Currently, only the technical specification for the BCSD 6518 C is available. The most important parameter that changes between the BCSD 6518 C and the BCSD 6525 C is the length of the ladder. This length gives the maximal dredging depth as well as the maximal and minimal swing width for the mining process. The swing widths, with a swing of 35° on each side, were recalculated over an approximation for the BCSD 6525 C: the maximal swing width is 67 m for the minimal dredging depth and the minimal swing width is 55 m for the maximal dredging depth (20 m + 5 m water).

One cut was estimated to be 2.38 m deep. This depth correlates with the diameter of the cutter head. Respecting an inclination angle of 23°, it was estimated to create steps with an individual height of 4.76 m and width of 11.8 m with the BCSD.

Beaver dredgers operate only in one direction (forward). They do not have a system to turn around at the end of the cutting length. For the first final positions there would not even be place to do so. Therefore, we are obliged to pull the whole dredger back to its starting position for a further, deeper cut. This pulling back operation will be done by winches that have fixed endings on the surface near the pit. The winches also do allow displacing the dredger sideways. For reducing the length of one cut the deposit has to be divided into more production zones. In our case we divided it into Volume 1, Volume 2, and Volume 3, which are equal (Illustration 32).

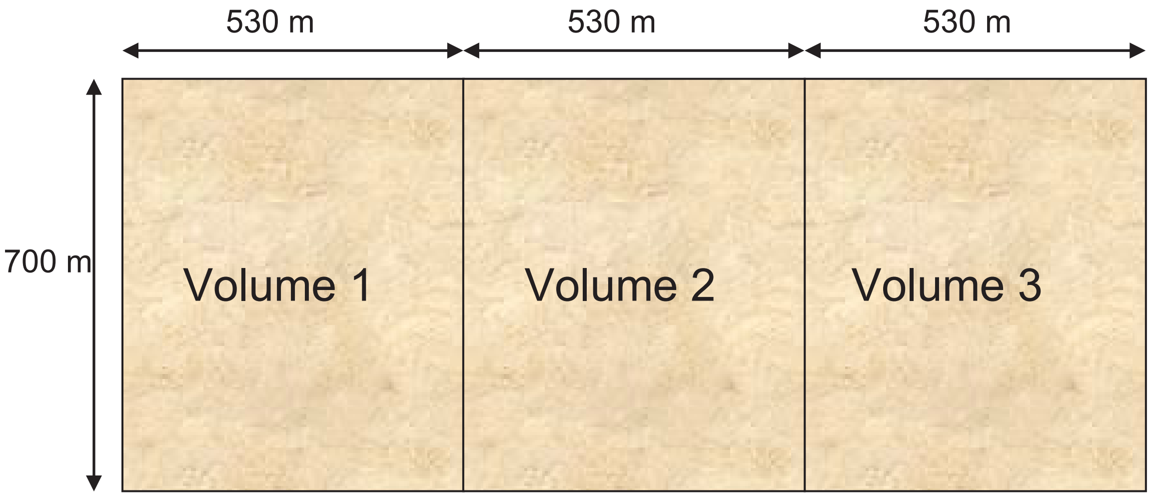


Illustration 32: Division of the deposit in 3 mining zones

The length of the operating section (~700 m) divided by the swing width (~71 m), gave the needed mining frequency of 10 to 11 cuts in horizontal direction (Illustration 33). The dredging height for the BCSD divided by the cutting height gives a needed mining frequency in vertical direction (Illustration 34). The water depth has to be adjusted for reaching deeper layers.

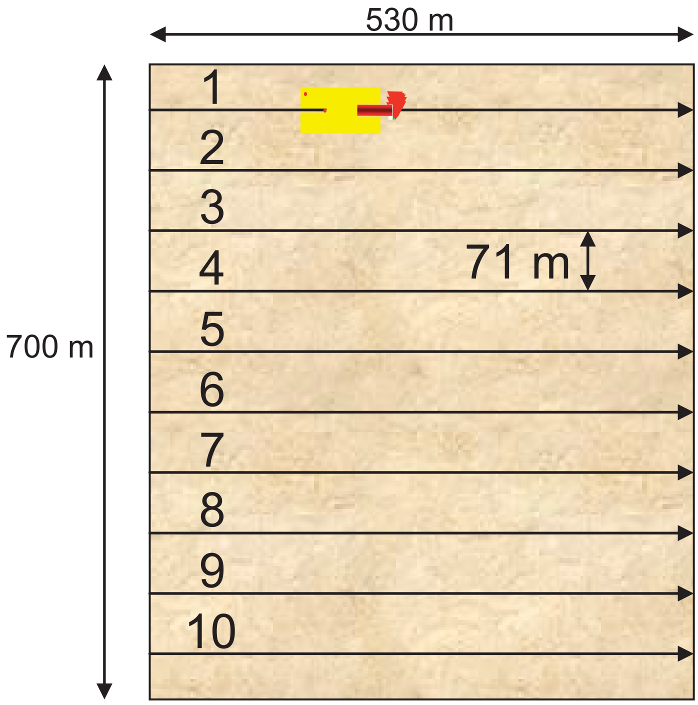


Illustration 33: Top view, mining operation with BCSD for one mine section

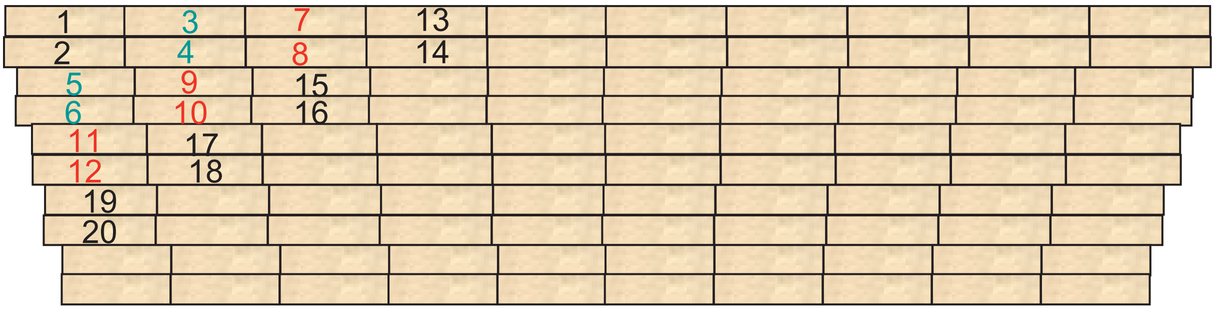


Illustration 34: Vertical view, possible mining development with BCSD; levels of 2.5 m height; one single water level

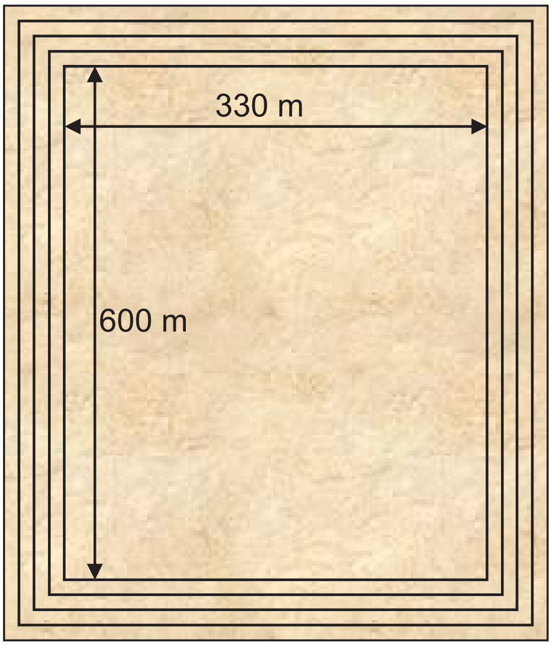


Illustration 35: Top view, final mine section after mining with BCSD for one water level

3.3 Mining with UMVs

Even if mining with UMVs was eliminated, because of higher capital and operating costs, their mining procedure will be shown for the sake of completeness.

In case of a previous mining operation with the BCSD, this has to be finished before mining with the UMV system. The reason is the need of cables to move the UMV pontoon on the water surface. The installation of more systems over the same surface at the same time is not possible. A solution could be a further division of the surface into smaller sub-surfaces. Therefore, sufficient security distances between the systems have to be respected. Anchors which have been used for the mining with the BCSD could be used directly for the UMV system.

The first action in starting the mining process with a UMV is the construction of a ramp. This ramp will have a width of 14 m and an inclined length of 28 m. For doing so the UMV has to be placed with the pontoon in one corner of the mining area. The UMV will then start to undercut (possible for -1 m) for creating a mining front with a final height of 5 m. The inclination of the ramp is 18%. The maximal mining width for the UMV is 7 m. Therefore the whole width of the ramp has to be constructed in two steps: after finishing the first half of the ramp, the UMV will mine the whole length of the mining area. Then it will be pushed back with the pontoon at the position next to the already constructed ramp, for a further construction of the second half.

Afterwards the UMV will mine a canal with 14 m width all around the full mining area in two steps (2 * 7 m). Therefore it will always be displaced back with the pontoon, because a mined width of 7 m does not allow an individual turn with the UMV on the ground. At the end also the constructed ramp will be mined, which is not needed any more. The individual steps of the process are shown with numbers in Illustration 36. Afterwards the UMV can start mining in tours without any pushing back process.

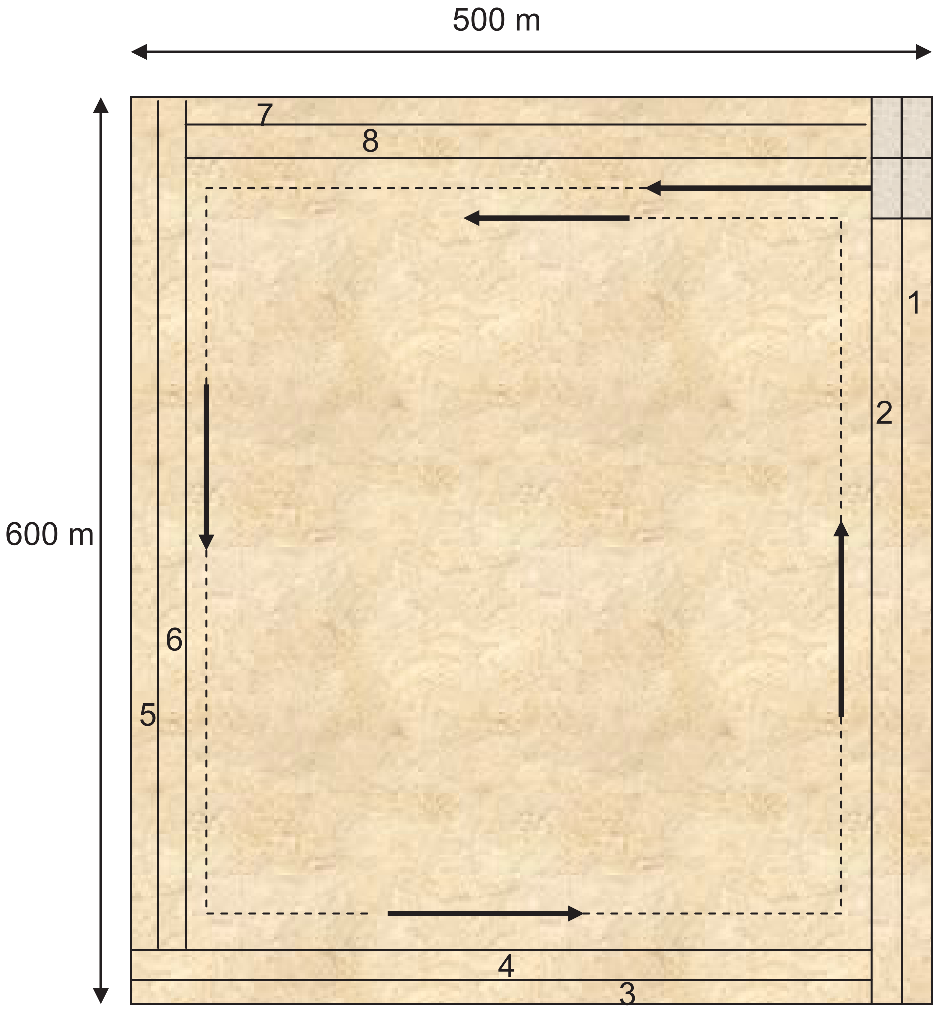


Illustration 36: Top view, mining operation with UMV system

In the same way several slices will be mined for reaching the final depth of the deposit. In our demonstration example the UMV has to mine slices with a height of 5 m each (Illustration 37 & 38).

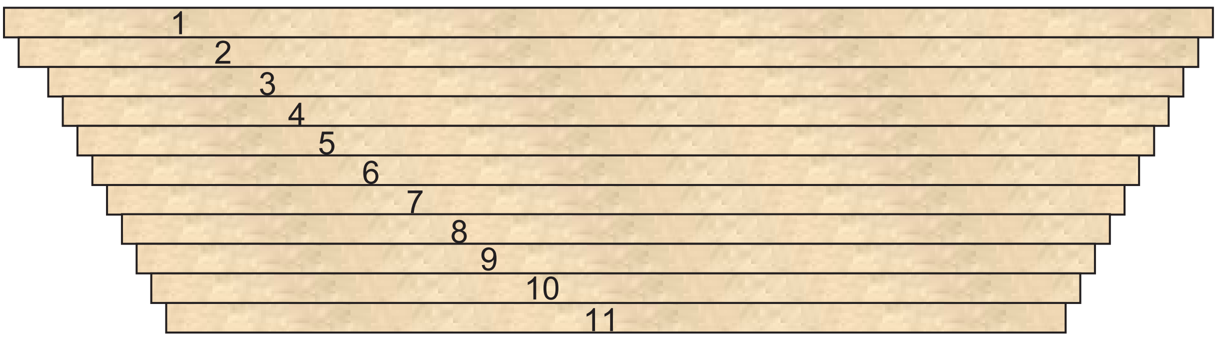


Illustration 37: Vertical coup, mining development with UMV system

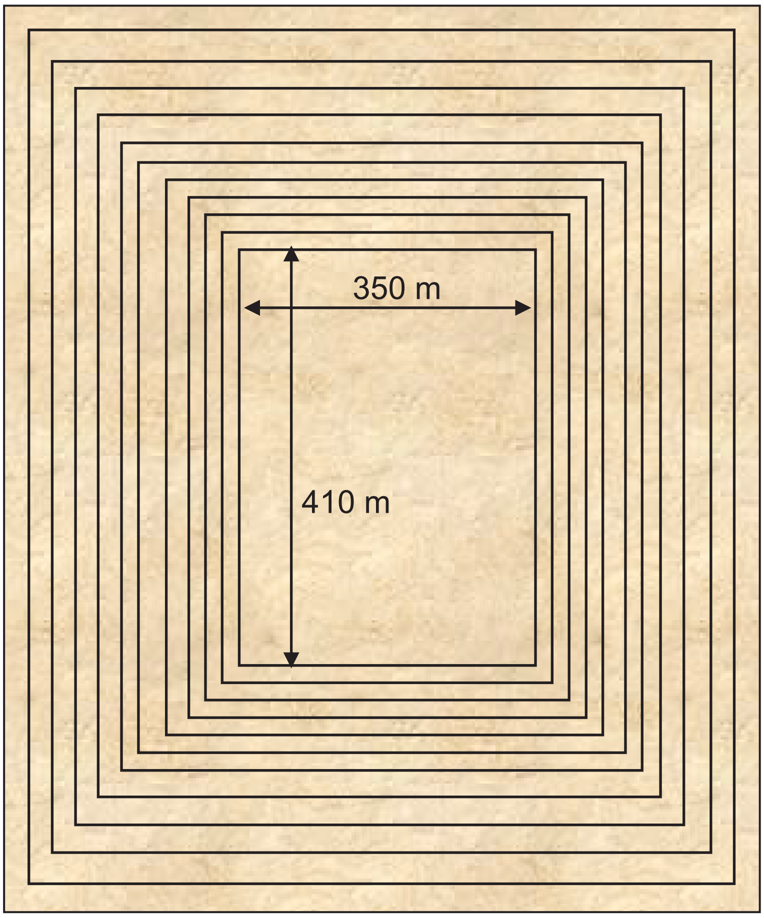


Illustration 38: Top view, final mine section after mining with UMV

3.4 Detailed mine planning for BCSDs

The mining of the Bakouma deposits by dredging in the Central African Republic was studied to be done only with Beaver Cutter Suction Dredgers. In Surpac a block model was created with block sizes of 25*25*10 m³ in which we imported all available information concerning uranium grade. The cut-off grade for designing the pits of the Patricia, Pato-Pama and Fosse deposits was 1 kg/t U_3O_8. Because of the conceptual phase we did not design the exact expanding of the single steps for every mining level as described in chapter 3.2 (Illustration 39).

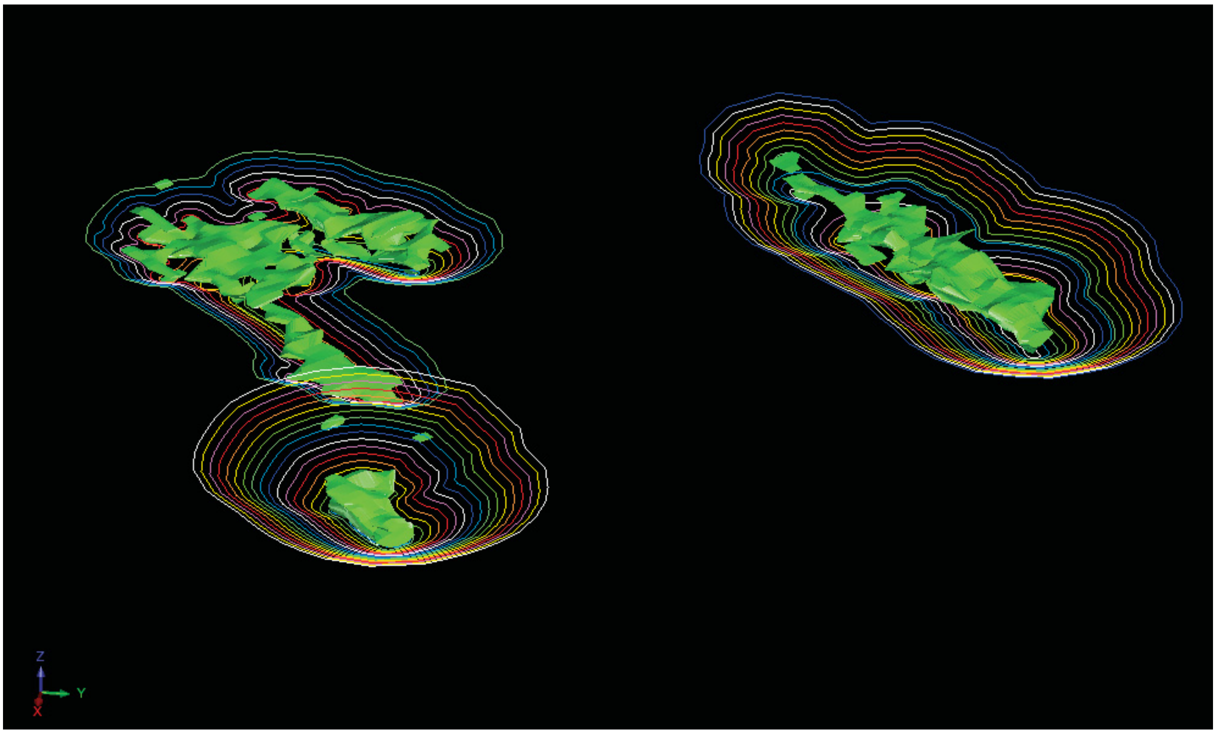


Illustration 39: Outlines of the in Surpac created pits and ore deposits; left above Pato-Pama, left below Fosse, right Patricia

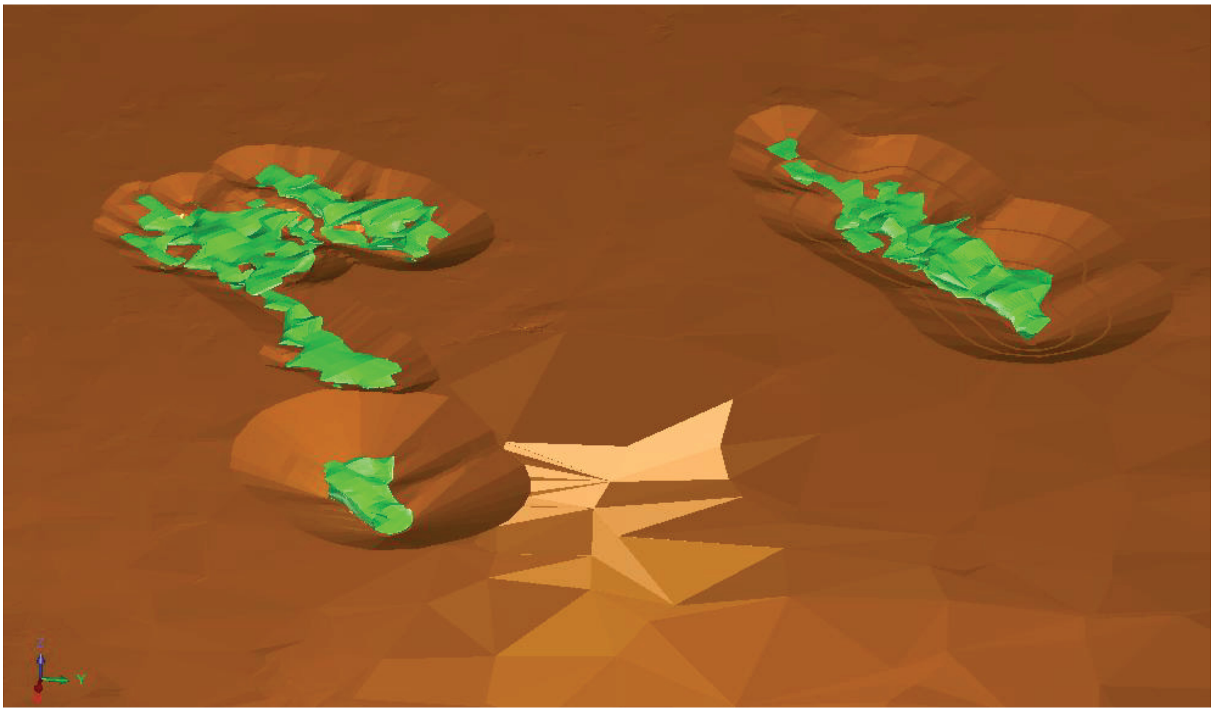


Illustration 40: Cut of topography with the pits and ore deposits; left above Pato-Pama, left below Fosse, right Patricia

The pit designs showed an overlap of the Pato-Pama and Fosse pits (Illustration 40). This is an advantage of displacing the BCSD from one pit to the other. To displace the BCSD from the Patricia pit to the Fosse or Pato-Pama pit, it has to be dismounted, or a canal for its movement has to be created.

The pit designs allowed making block model reports inside their outer limits. The 10 m height blocks were divided into 4 levels with an equal height of 2.5 m, which roughly represents the mining height of one cut with a BCSD. The block model report gave detailed information about mineral and waste volumes inside one mining level. This information allowed defining different mine planning scenarios. The aim was to stabilize the yearly mineral production of 830 kt for a constant feeding of the treatment plant.

The mine planning was done in two different ways: total mining of the deposits and mining of the deposit with accepting losses. A mine planning with losses means that waste beavers with higher capacities mine as fast as possible poorer zones for getting soon access to the rich ore zones. Afterwards, the richer ore zones will be mined with ore beavers only.

In any case the planning is constrained by finishing a whole level before starting to mine the following. For each scenario we used once levels of 2.5 m and once levels of 15 m. 2.5 m height levels present approximately the mining height of one cut of a BCSD. 15 m height levels present a realistic working height of beavers.

For the scenarios without losses, we used 4 BCSDs, of which two were exclusively used for mining waste, and the other two exclusively for mining ore (Table 5). The total capacities of the two ore beavers together roughly represent the needed capacity to reach the objective of ore production which is 165 t/h. In total the waste Beavers have a higher capacity as the minimum required capacity to reach the objective of waste production (1583 t/h). At all time at least one waste and one ore BCSD have to stay together.

Capacity		
Beaver 1 (waste)	1860	t/h
Beaver 2 (ore)	110	t/h
Beaver 3 (waste)	787	t/h
Beaver 4 (ore)	56	t/h
Total waste capacity	2646	t/h
Total ore capacity	166	t/h

Table 5: BCSD and their needed productions for mine planning without losses

For the mining scenarios with losses we used three BCSDs, where two were exclusively used for waste production (Table 6). The ore production beaver was designed to do the total needed ore production alone.

Capacity		
Beaver 1 (waste)	1860	t/h
Beaver 2 (ore)	166	t/h
Beaver 3 (waste)	787	t/h
Total waste capacity	2646	t/h
Total ore capacity	166	t/h

Table 6: BCSD and their needed productions for mine planning with losses

The starting point for the mine planning scenarios was at the Patricia deposit. The Fosse deposit was preferred to come a close second. The last deposit to be mined would therefore be Pato-Pama. For transferring a beaver from one pit to the next, we added 100 days without any production for this beaver.

3.4.1 Mining of 2.5 m height levels

For mining of 2.5 m height levels the yearly needed objective of ore production of 830 kt could be reached in seven but not subsequently years. The first significant decrease in the ore production in the 11th year comes from a transfer of the ore beaver No. 2 from Patricia to Fosse. That means the total production capacity of the ore beavers could not be used at that time. The decreasing production in the 13th year is due to the starting phase of ore production in the Pato-Pama deposit with beaver No. 4 and the end phase of production of beaver No. 2 in the Fosse deposit. The most important production decrease is in the 14th year, when beaver No. 2 has to be transferred to the Pato-Pama deposit and ore beaver No. 2 does still not use its full capacity (Illustration 41). The production stops coming from transfer of beavers from one pit to the next cannot be eliminated. However, production ruptures due to the heterogeneity of the deposits could be eliminated at starting phases of mining when accepting losses in poorer zones (Table 7).

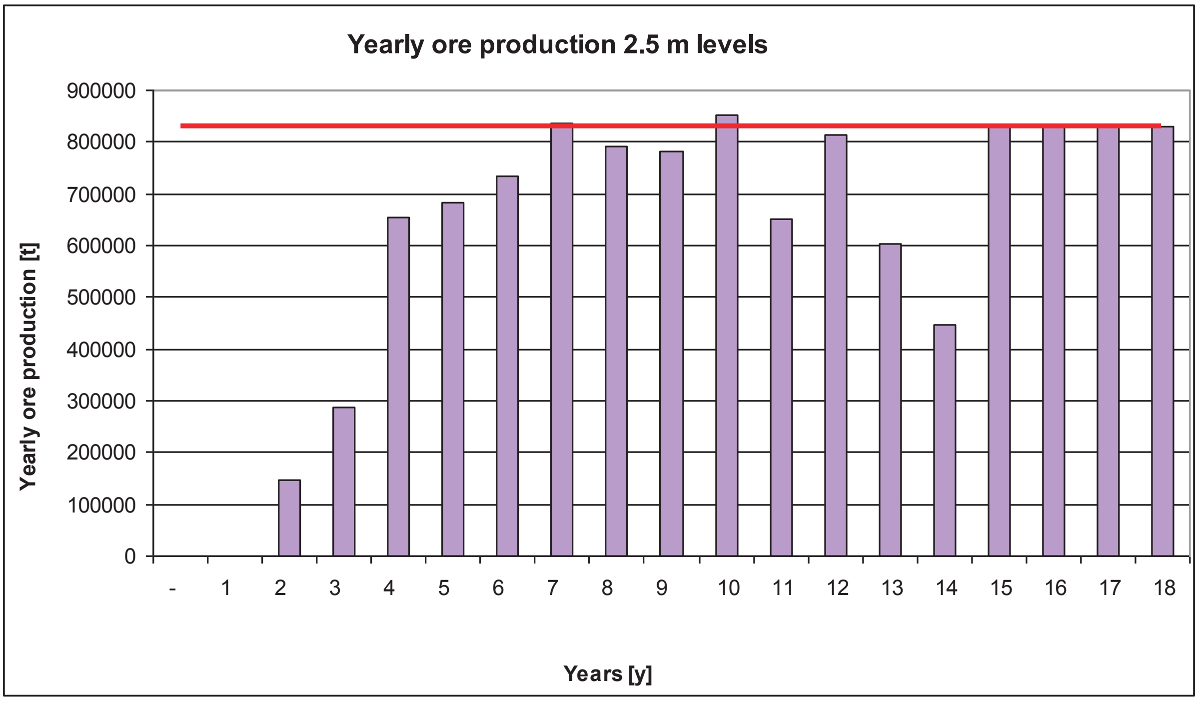


Illustration 41: Yearly ore production of 2.5 m levels

	Production years							
	Beaver 1		Beaver 2		Beaver 3		Beaver 4	
Patricia	0	2	1,15	10,55	0	10,55	-	
Fosse	2,29	10,93	10,93	13,25	10,93	13,25	2,29	11,74
Pato-Pama	11,22	18,16	14	18,15	-		12,03	18,16

Table 7: Application time of beavers in the deposits for mining 2.5 m levels

3.4.2 Mining of 2.5 m height levels with losses

For getting a more constant yearly ore production, we accepted losses at the starting phases. That means waste beavers were used to mine poorer zones for giving as soon as possible access to richer ore zones. These losses represent 6.3% of the total ore production. Layers -where the capacity ratio of ore beaver to waste beaver did not allow reaching the objective of production- were mined only with waste beavers soon as they reached the rich ore zones. Because of the utilization of one unique ore beaver we have depressions in the production at the moments of its displacement (9th and 13th year) from one to the next pit (Illustration 42 and Table 8).

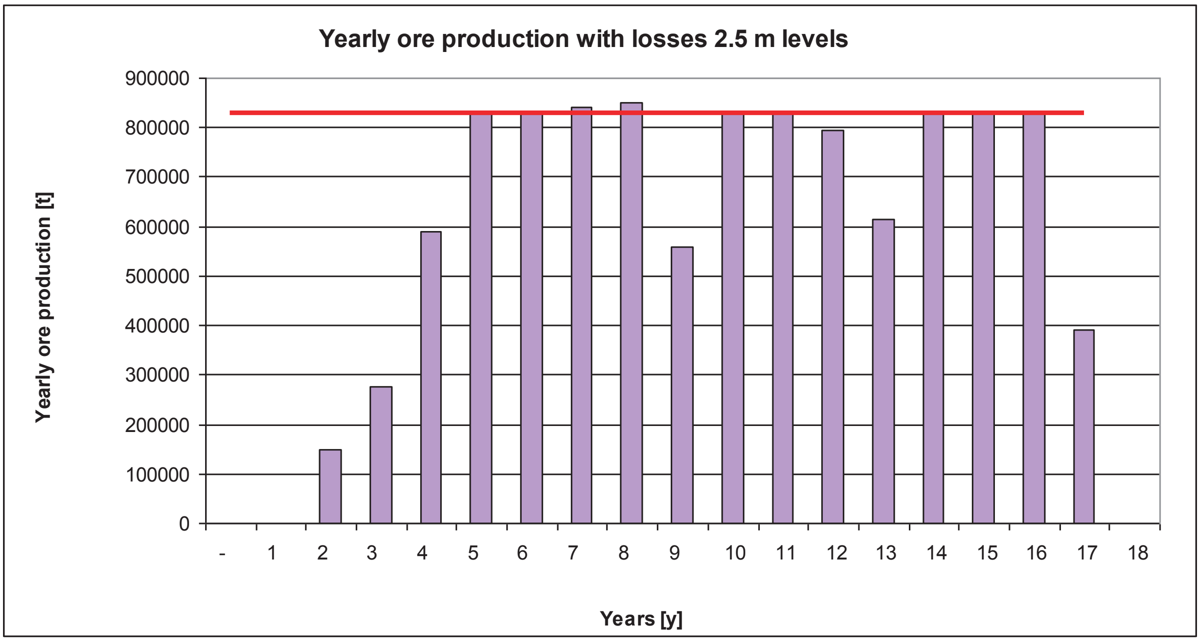


Illustration 42: Yearly ore production of 2.5 m levels with accounted losses

Production years						
	Beaver 1		Beaver 2		Beaver 3	
Patricia	0	2	1,15	8,29	0	8,29
Fosse	2,29	8,71	8,58	11,85	8,58	11,85
Pato-Pama	9,29	16,47	12,14	16,47	-	

Table 8: Application time of beavers in the deposits for mining 2.5 m levels with losses

3.4.3 Mining of 15 m height levels

Even if one cut of the BCSD is approximately only 2.5 m in height its drill arm allows mining down to a depth of 20 m. This fact allows doing a kind of “pit optimization” over the maximum mining height. We tried optimizing the ore production when mining layers of 15 m. This allowed boosting production rates in the starting phases. So we reached a production of 214 kt of ore in the second year compared to 148 kt for a mining of 2.5 m layers without looses after the same time. In the third year we reached a production of 560 kt of ore compared to 287 kt for a 2.5 m level mining. Nevertheless the expansion of the height of the layers to 15 m implicated longer time units to mine a total layer that makes the exact adaption of the dredgers in the pits harder to perform. The total result was a less stable yearly ore production (Illustration 43 and Table 9).

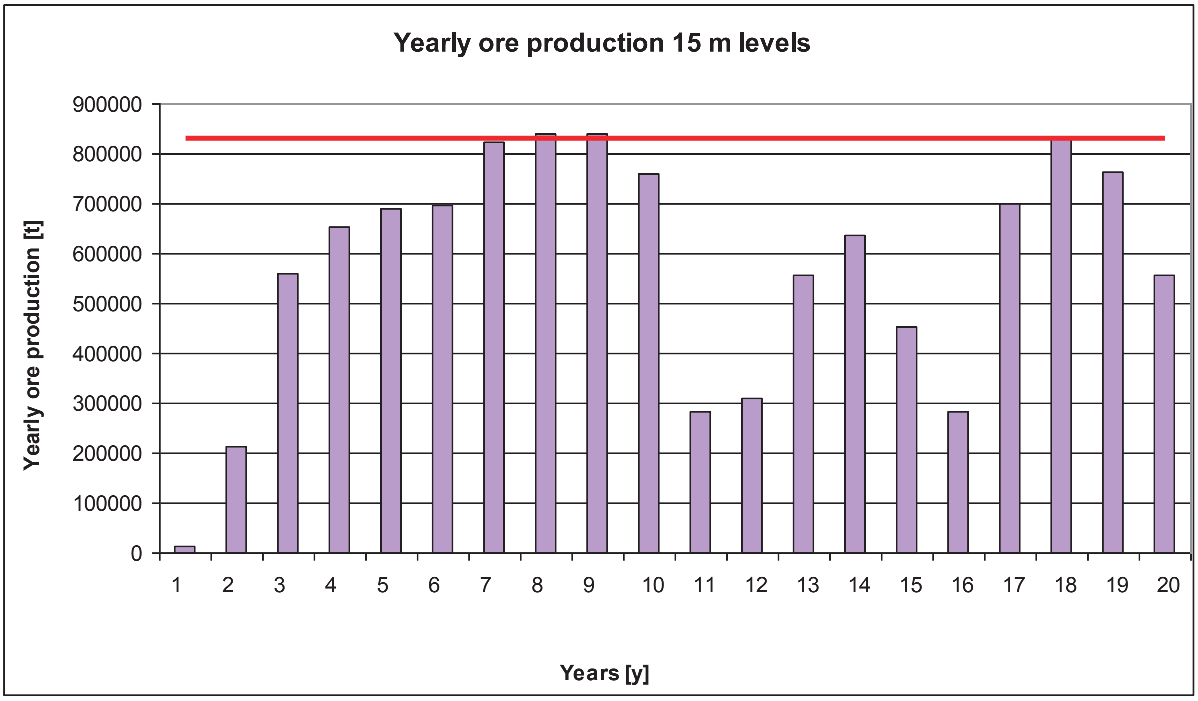


Illustration 43: Yearly ore production of 15 m levels

Production years								
	Beaver 1		Beaver 2		Beaver 3		Beaver 4	
Patricia	-	2,36	0,83	9,85	-	9,85	-	
Fosse	2,65	11,91	11,91	14,31	11,91	14,31	2,65	11,91
Pato-Pama	12,19	20,50	16,23	20,50	-		12,19	15,30

Table 9: Application time of beavers in the deposits for mining 15 m levels

3.4.4 Mining of 15 m height levels with losses

The optimized mining of 15 m height levels with accepted losses already factored in the third year a production of 753 kt compared to 275 kt in the 2.5 m level scenario with losses. The full objective of production can already be reached in the fourth year. In comparison, a full production in the 2.5 m level scenario cannot be reached before the fifth year (Illustration 44 and Table 10).

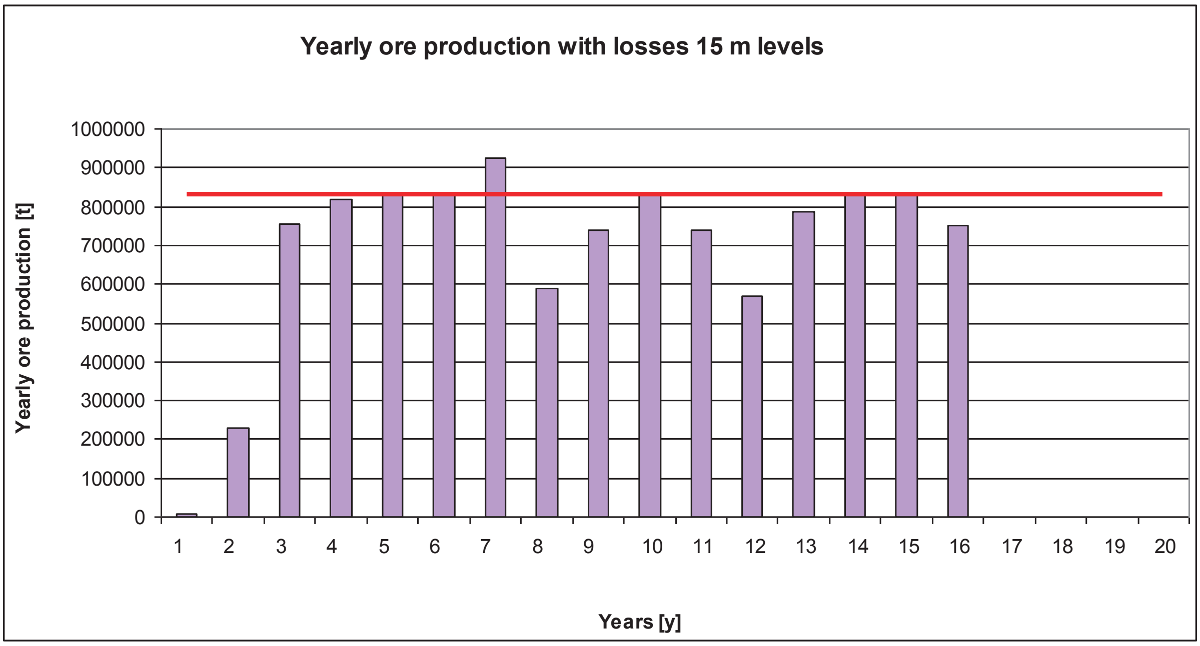


Illustration 44: Yearly ore production of 15 m levels with accounted losses

Production years						
	Beaver 1		Beaver 2		Beaver 3	
Patricia	0	2,29	1,67	7,34	0	7,34
Fosse	2,58	9,28	7,62	10,89	2,58	10,89
Pato-Pama	9,57	15,91	11,18	15,91	-	

Table 10: Application time of beavers in the deposits for mining 15 m levels with losses

3.4.5 Stabilisation of ore an U metal production

The optimisation of the production curbs for ore (Illustration 47) and U metal (Illustration 48) over the whole lifetime of the mine is possible. However, a unique optimisation of these two curbs implicates a very strong diversifying waste production curb (Illustration 49). Its variation lies between 696 kt and 19 827 kt. With the defined waste beavers (see chapter 3.4.1) we excess the maximum capacity of 13.3 Mt/y (green line in diagram) at four moments of the mine life for short periods (Illustration 45 and 46). To reach the optimised production curbs, it would be necessary to move the waste mining capacity to the maximum needed value. But this would again degrade the utilisation ratios which are already very low.

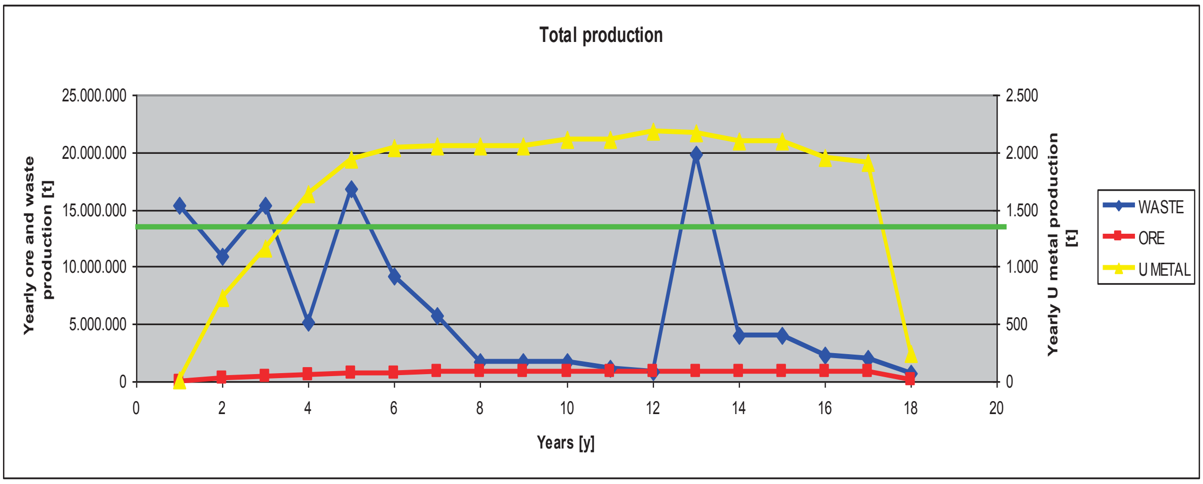


Illustration 45: Waste, ore and U metal production

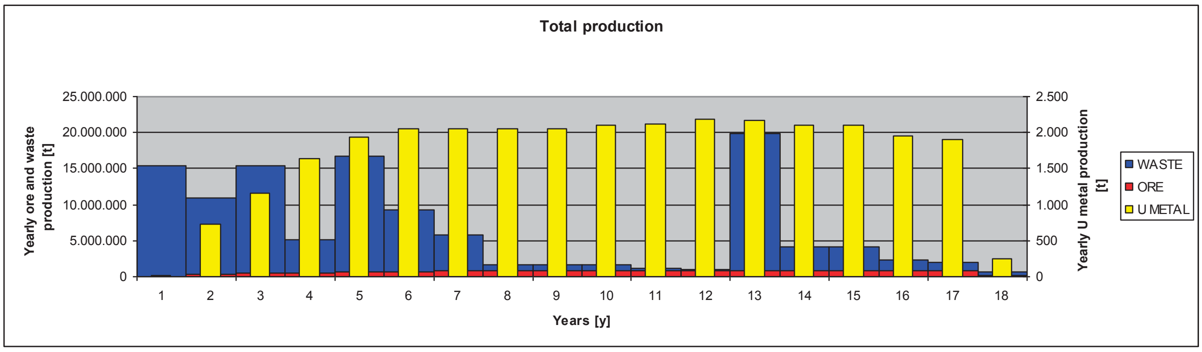


Illustration 46: Waste, ore and U metal production

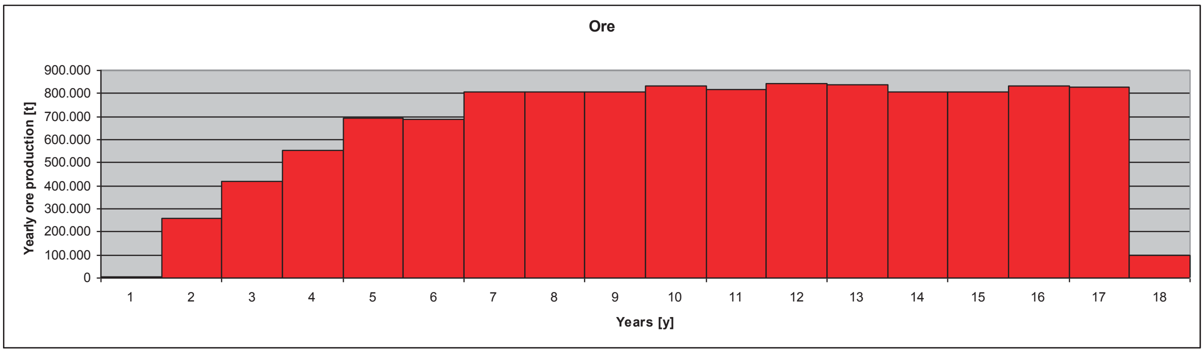


Illustration 47: Ore production

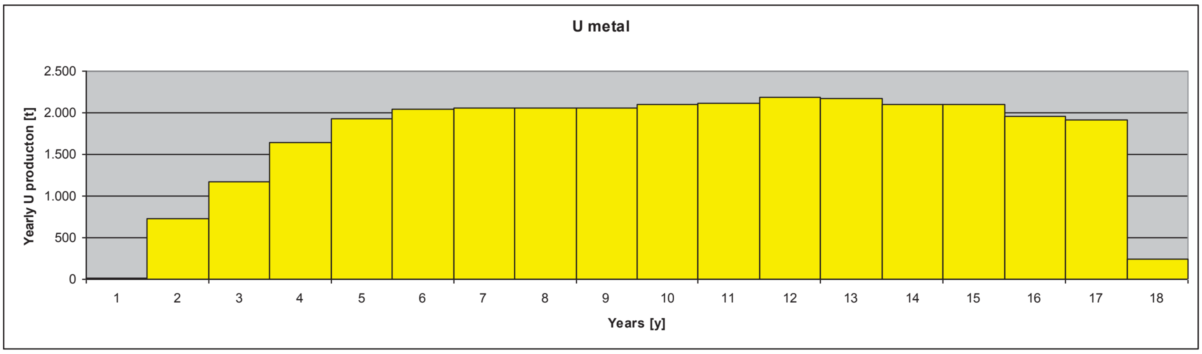


Illustration 48: U metal production

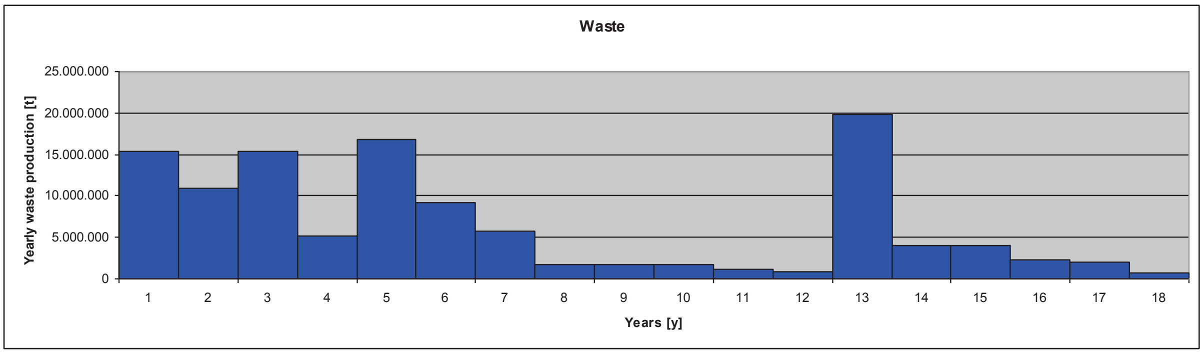


Illustration 49: Waste production

The utilisation rates of the ore beavers reach after short time periods their maximum. At the same time decreases the utilisation rate of the waste beavers continuously to very low values of sometimes under 10%, before increasing once again at the end of the mining process of each deposit (Illustration 50, 51 and 52).

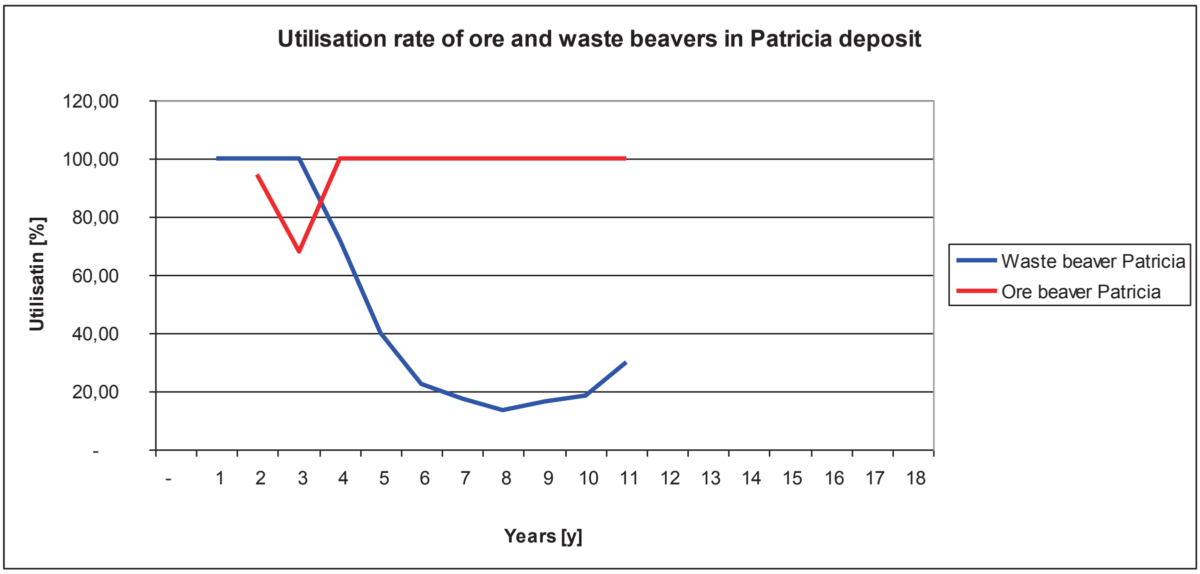


Illustration 50: Utilisation rate of ore and waste beavers in Patricia deposit

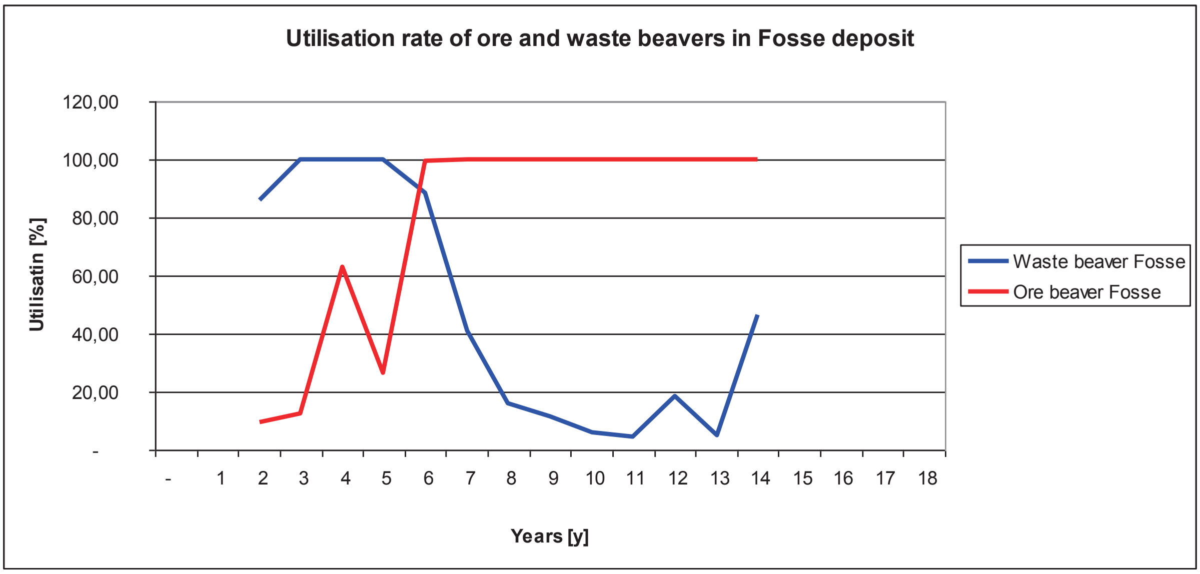


Illustration 51: Utilisation rate of ore and waste beavers in Fosse deposit

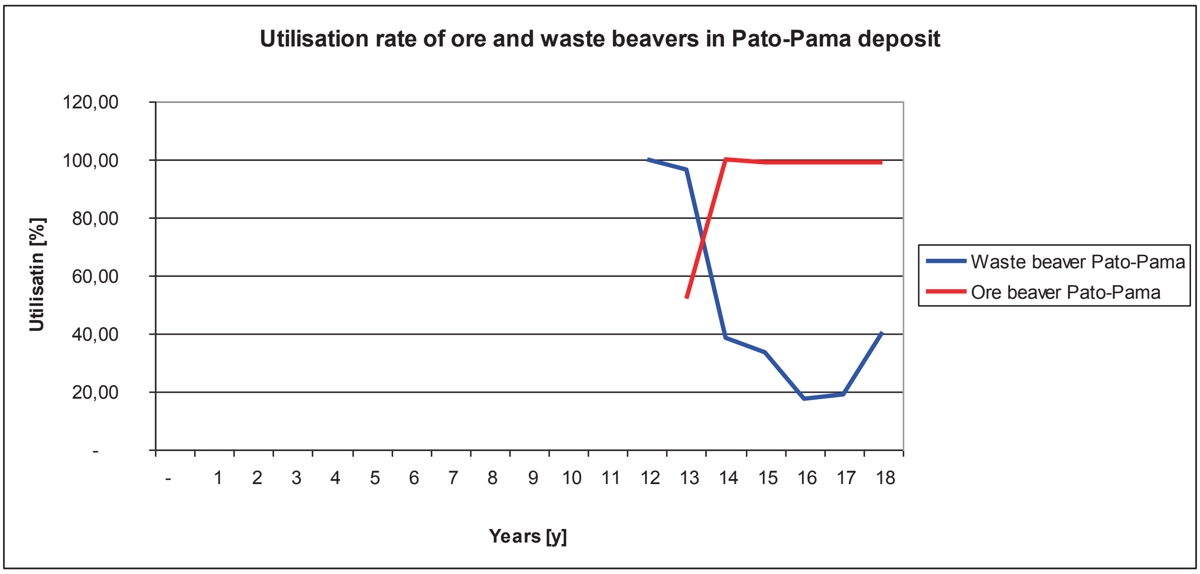


Illustration 52: Utilisation rate of ore and waste beavers in Pato-Pama deposit

It is possible to improve that result if the mining level under water is not horizontal, but has a maximum difference of 20 m. Unfortunately it was not possible to study this due to lack of time.

3.5 Selectivity study

In atom reactors water is used for protecting the environment against nuclear radiation. That means water lengthens the radiation. This fact shows that it will be hard to measure radioactivity under water.

In open pits, Areva uses radioactivity instruments to measure radiation in a certain frequency (for example every 5 m). These instruments are manageable by one person, who carries and positions it correctly. Its weight is approximately 10 kg. To do a radiation measure, the instrument is placed on the ground. The time for one measure takes just a few ms. Uranium can be detected only a few dm away from the reading point.

For our underwater mining site in Bakouma it will not be possible to measure the radioactivity directly at the front. An individual mechanism will be needed for doing frequent readings. The current idea is using a boat that can move on the water surface to undertake measurements of a certain frequency. Therefore the same instrumentation, as for an open pit, will be used, but in an adapted form. It will be positioned on the ground from the boat through a cable. The necessary frequency of reading points depends on the regularity of the deposit. The measurement has to be undertaken in imperturbable positions. That means, the front where the excavating device is working at the moment has to be avoided.

The measurement of radiation in pipelines with a diameter of 650 mm will not be possible. The reasons are that the instrumentations cannot be positioned directly on the material and in addition the flow of the mined material is very high. It would be possible to measure the radioactivity on band conveyors, but therefore the material must be previously drained and dried.

IHC Holland B.V. proposed a radioactivity-measure-system (see chapter 2.3.4.1) which does not measure the radioactivity of a material but the density by sending radiation through the material. This does not help to be selective, because the density between waste and ore will not differ a lot, and therefore it will not give information about the uranium content.

3.6 Cost calculation for dredge mining

The following calculations are based on the IHC "Conceptual study Bakouma uranium dredge mining" Concept from February 2009. The IHC calculations were based on an annual production of 7.5*10^6 m^3 and 5040 effective working hours (Table 11) for mining equipment per year. All calculations are based on a homogenous material with a UCS of 1 MPa.

ORGANIZATION OF WORKING TIME AT THE MINE SITE		
Number of days per year	350	d/y
Number of shifts per day	3	p/d
Number of hours per post	8	h/p
Number of hours per year	8400	h/y
WORKING TIME FOR EQUIPMENTS		
Rate of availability of equipments	75%	
Total number of working hours per year	6300	h/y
Rate of production	80%	
Number of production hours per year	5040	h/y
Number of non productive hours per year	1260	h/y
Ratio non productive hours/productive hours	0,25	
WORKING TIME FOR PERSONNEL		
Hours per year and working post	8400	h/y
Number of working days per year	211	d/y
Number of days per year excluding weekends	261	d
Number of holidays	10	d
Number of leave days	25	d
Number of absent days	15	d
Number of working hours per year	1688	h/y
Number of effective hours per person and year	1266	h/y
Number of persons per working post	7	
Number of persons per working post and per shift	2	

Table 11: Effective production hours per year for equipments and personnel

In comparison to classical mining, dredging is not affected by the rainy season in the Central African Republic. Therefore, the production is possible during the whole year. In the calculation, 15 days without production are included, which accounts for big maintenance operations (changing pipelines etc.). Dredging allows for working in three shifts, because working underwater indicates the same circumstances during daytime and nighttime.

3.6.1 Capital and operating costs for BCSD

My visit to IHC Holland B.V. allowed me to define the first technical vertices for underwater mining sites in Bakouma. Beaver Cutter Suction Dredgers are well-known and often used in civil construction. IHC Holland supplied more than 600 of these standard machines worldwide.

Beaver dredgers could do the mining after the removal of the first 15 m of overburden with classical mining. This course of action is very interesting because Beaver Dredgers promise the lowest possible costs per m^3 of dredged material.

The IHC Concept allowed calculating capital (Table 14) and operating costs (Table 12 and 13) for a Beaver Cutter Suction Dredger:

ENERGY COSTS		
Power of pumps	5058	kW
Engine power	1291	kW
Fuel consumption of pumps	208	g/kW/h
Fuel consumption of engine	191	g/kW/h
Power factor at production	80	%
Power factor at non production	30	%
Hypothesis 1: Diesel motor		
Fuel consumption per hour of production	1,0	t
Fuel consumption per hour of non-production	0,4	t
Total consumption of fuel per hour of production	1,1	t/h
Fuel costs	1145,0	$/t
Costs of fuel per hour	1301,1	$/h
Costs of fuel per m³	1,9	$/m³
Hypothesis 2: Electrical motor, fuel central		
Consumption of kWh per hour of production	5079,2	kWh
Consumption of kWh per hour of non-production	1904,7	kWh
Total consumption of kWh per hour of production	5555,4	kWh
Costs per kWh	0,3	$/kWh
Costs of electricity per hour	1388,8	$/h
Cost of electricity per m³	2,0	$/m³
Hypothesis 3: Electrical motor, hydropower		
Costs of kWh	0,015	$/kWh
Costs of electricity per hour	83,3	$/h
Costs of electricity per m³	0,1	$/m³
MAN POWER		
Number of qualified working posts	4	
Number of non qualified working posts	4	
Cost for qualified personnel	10	$/h
Costs for non qualified personnel	10	$/h
Dispenses for man power	176,9	$/h
Costs of man power per m³	0,3	$/m³
MAINTENANCE		
Variable with production	2,5	$/m³
Variable with working hours	45	$/h
Fixed costs	200000	$/an
Costs linked with hours of production	56,3	
Costs linked with annual fixed costs	31,7	
Costs per hour	88,0	$/h
Costs per m³	2,6	$/m³
PIPE LINE		
Life time	10	Mm³
Floating line	1,6	M$
Shore line	0,78	M$
Costs per m³	0,24	$/m³

Table 12: Calculation of operating costs for 1860 t/h capacity Beaver Cutter Suction Dredger

Operating costs	Fuel		% Fuel	Fuel electricity		% Fuel electricity	Hydraulic electricity		% Hydro-power
Energy costs	1,9	$/m³	37,4	2,0	$/m³	38,9	0,1	$/m³	3,7
Man power costs	0,3	$/m³	5,1	0,3	$/m³	5,0	0,3	$/m³	7,8
Maintenance costs	2,6	$/m³	52,8	2,6	$/m³	51,5	2,6	$/m³	81,2
Pipe line costs	0,2	$/m³	4,8	0,2	$/m³	4,7	0,2	$/m³	7,4
TOTAL costs per m³	5,0	$/m³		5,1	$/m³		3,2	$/m³	
TOTAL costs per ton	2,5	$/t		2,6	$/t		1,6	$/t	
TOTAL costs per ton ore	26,4	$/t ore		27,1	$/t ore		17,2	$/t ore	

Table 13: Summary of operating costs for BCSD

Energy costs: For the moment three different energy resources for dredging mining were analyzed: fuel, electricity with fuel power plant, and with hydropower. The fuel costs per ton as well as the energy costs per kWh produced from a fuel central and through hydropower were defined by AREVA (Contact Z. El Marzouki).

Diesel motors and electrical motors, which get energy from a fuel central, have nearly the same costs. The cheapest way is an electrical motor, which gets energy through hydropower.

Man power: Included posts: 1 chef, 1 pilot, 1 mechanician, 1 electrician, 1 for machine room, 1 for service boat, 2 on deck.

Maintenance: Essentially the costs come from spare parts (2.5 $/m³ from 2.6 $/m³). The values were taken from the IHC Concept. The "variable with production" value was doubled for including transport costs, which is fixed with 600 $/t.

Pipeline: Lifetime of a pipeline was calculated through the flow rate.

The used power has a strong impact on the production costs. By using fuel this impact reaches nearly 40% of the production costs. Hydropower is the cheapest way to mine with a BCSD.

The costs for maintenance, particularly the spare parts, have over 50% of influence on the production costs. For hydropower, this value reaches over 80%. The calculation is based on several assumptions.

The values for capital costs were taken from the IHC Concept.

Beaver Cutter Suction Dredger		
Value Dredger	10,5	M$
Value Booster	1,98	M$
Number of boosters	2	
Transport costs	1,3	M$
TOTAL COSTS BCSD	15,75	M$

Table 14: Capital costs for a 930 m³/h capacity BCSD 6525 C

3.6.2 Capital and operating costs for UMV

The IHC Concept allowed the calculation of capital (Table 17) and operating costs (Table 15 and 16) for an Underwater Mining Vehicle System with 1 pontoon and 2 UMVs, which can do a production of 1224 t/h (2*612 t/h).

ENERGY COSTS		
Power of pumps	2896	kW
Power of generator	2500	kW
Fuel consumption of pumps	208	g/kW/h
Fuel consumption of generator	208	g/kW/h
Power factor at production	80	%
Power factor at non production	30	%
Hypothesis 1 : Diesel motor		
Fuel consumption per hour of production	0,9	t
Fuel consumption per hour of non-production	0,3	t
Total consumption of fuel per hour of production	1,0	t/h
Fuel costs	1145,0	$/t
Costs of fuel per hour	1124,5	$/h
Costs of fuel per m³	2,3	$/m³
Hypothesis 2 : Electrical motor, fuel central		
Consumption of kWh per hour of production	4316,8	kWh
Consumption of kWh per hour of non-production	1618,8	kWh
Total consumption of kWh per hour of production	4721,5	kWh
Costs per kWh	0,3	$/kWh
Costs of electricity per hour	1180,4	$/h
Costs of electricity per m³	2,4	$/m³
Hypothesis 3 : Electrical motor, hydropower		
Costs of kWh	0,015	$/kWh
Costs of electricity per hour	70,8	$/h
Costs of electricity per m³	0,1	$/m³
MAN POWER		
Number of qualified working posts	5	
Number of non qualified working posts	4	
Cost for qualified personnel	10	$/h
Costs for non qualified personnel	10	$/h
Expenses for man power	199,1	$/h
Costs of man power per m³	0,4	$/m³
MAINTENANCE		
Variable with production	3,58	$/m³
Variable with working hours	112,5	$/h
Fixed costs	300000	$/an
Costs linked with hours of production	140,6	
Costs linked with annual fixed costs	47,6	
Costs per hour	188,2	$/h
Costs per m³	4,0	$/m³
PIPE LINE		
Life time	10	Mm³
Floating line	1,6	M$
Shore line	0,78	M$
Costs per m³	0,24	$/m³

Table 15: Operating costs for a 1224 t/h capacity UMV-System (1 pontoon + 2 UMV)

Operating costs	Fuel		% Fuel	Fuel electricity		% Fuel electricity	Hydraulic electricity		% Hydro-power
Energy costs	2,3	\$/m³	33,2	2,4	\$/m³	34,3	0,1	\$/m³	3,0
Man power costs	0,4	\$/m³	5,9	0,4	\$/m³	5,8	0,4	\$/m³	8,5
Maintenance costs	4,0	\$/m³	57,4	4,0	\$/m³	56,5	4,0	\$/m³	83,4
Pipe line costs	0,2	\$/m³	3,4	0,2	\$/m³	3,4	0,2	\$/m³	5,0
TOTAL costs per m3	6,9	\$/m³		7,0	\$/m³		4,8	\$/m³	
TOTAL costs per ton	3,5	\$/t		3,5	\$/t		2,4	\$/t	
TOTAL costs per ton ore	36,6	\$/t ore		37,2	\$/t ore		25,2	\$/t ore	

Table 16: Summary of operating costs for UMV system

Energy costs: Same comments as in chapter 3.6.1.

Man power: Included posts: 1 chef, 2 pilots, 1 mechanic, 1 electrician, 1 for machine room, 1 for service boat, 2 on deck.

Maintenance: Essentially the costs come from spare parts (3.6 \$/m³ from 4 \$/m³). Same comments as in chapter 3.6.1.

Pipeline: Same comments as in chapter 3.6.1.

The used power has again a strong impact on the production costs. By using fuel, either if it is a diesel motor or an electrical motor with energy through a fuel central, this impact reaches over 30% of the production costs. Hydropower is the cheapest way to mine with a UMV.

The costs for maintenance, particularly the spare parts, have over 55% influence on the production costs. For hydropower this value reaches over 80%.

The rate of availability of the equipments is quite low, because all of the equipment works in series (UMV, pump, underwater pipeline, pontoon, booster, floating pipeline, shore line). If one of the systems is out of order, the whole process is impossible.

The values for capital costs were taken from the IHC Concept.

UMV		
Value System (2 UMV + 1 pontoon)	64,4	M\$
Value Booster	1,975	M\$
Number of boosters	2	
Transport costs	0	M\$
TOTAL COSTS UMV SYSTEM	68,3	M\$

Table 17: Capital costs for UMV System

3.6.3 Average dredging and mining costs

As shown in chapter 3.6 the operating costs between the BCSD and the UMV dredging system differ. Because of the more important dredging cost of UMV systems their application as a unique system has to be eliminated. The application of two dredging systems at the same time offer average dredging costs (Table 18).

OPERATING COSTS FOR DREDGING		Fuel		Fuel electricity		Hydropower electricity	
BCSD	TOTAL per m³	5,0	$/m³	5,1	$/m³	3,2	$/m³
	TOTAL per ton	2,5	$/t	2,6	$/t	1,6	$/t
	TOTAL per ton ore	26,4	$/t	27,1	$/t	17,2	$/t
UMV	TOTAL per m³	6,9	$/m³	7,0	$/m³	4,8	$/m³
	TOTAL per ton	3,5	$/t	3,5	$/t	2,4	$/t
	TOTAL per ton ore	36,6	$/t	37,2	$/t	25,2	$/t
Average	TOTAL per m³	6,2	$/m³	6,3	$/m³	4,2	$/m³
	TOTAL per ton	3,1	$/t	3,1	$/t	2,1	$/t
	TOTAL per ton ore	32,6	$/t	33,3	$/t	22,1	$/t

Table 18: Dredging operating costs

The dredging systems demand mining of the overburden with classical mining systems. The average mining costs were calculated when taking into account an operating cost of 1.15 $/t for classical mining. The mining site was therefore divided into 3 volumes. Volume 1 will be mined with classical mining methods, volume 2 with the BCSD, and volume 3 with the UMV system. The average mining costs (Table 20) were calculated over the working proportions of the three different mining systems (Table 19).

Proportion classical mining	25,8	%	16,8	Mm³
Proprotion BCSD	28,8	%	18,8	Mm³
Proportion UMV	45,3	%	29,5	Mm³
TOTAL	100,0	%	65,2	Mm³

Table 19: Working proportions of mining systems

AVERAGE MINING COSTS	Fuel		Fuel electricity		Hydropower electricity	
TOTAL per m³	5.2	$/m³	5.2	$/m³	3.7	$/m³
TOTAL per ton	2.6	$/t	2.6	$/t	1.8	$/t
TOTAL per ton ore	27.4	$/t	27.8	$/t	19.5	$/t

Table 20: Average mining costs

3.7 Conclusion

Mining costs for dredging can vary greatly regarding the used equipments. UMV systems were eliminated from further studies because of their higher capital costs (68.3 M$ to 15.75 M$) and higher operating costs (2.4 $/t to 1.6 $/t) compared to the BCSD. The capital costs for mining with UMV systems would still rise when taking in account that one system produces 1224 t/h which represents approximately two thirds of the total needed production. In comparison the BCSD produces 1860 t/h.

The average mining cost for dredging with an UMV system exceeds the mining costs for classical mining as well as dredging with the BCSD. Also, an average mining cost for the combination of both dredging systems does exceed mining costs for classical mining. That is why we had to eliminate the UMV system from our further studies. A dredging scenario with BCSD as unique dredging system was assumed.

The BCSD seems to be a technically and economically possible solution near traditional mining methods. In the cost calculation we did not add transport costs which are approximately 8% of capital costs, a contingency of 25% and costs for auxiliary equipments.

A good mining plan adjusts the ore, U metal, and waste production at the same time, which optimises utilisation of the well-sized mining machines. Mining with BCSD demands a worst case mining layer after layer. This principal limitation is intensified by the fact that BCSDs have a weak mobility and can only mine strips. Selective mining or work-sharing between ore and waste beavers could barely be realised. Further BCSD can only be transferred one time from one pit to another because of the necessary time consumption.

Another limiting factor of mining with the BCSD could be the adjustment of the water level in the pit. The produced slurry has a solid content of 20% to 30%. With slurry production of 5826 t/h to 8739 t/h, the Beavers would suck 1133 l/s to 1942 l/s of water. The water flow in the pit is 1000 l/s to 3000 l/s. That means the BCSD could control the water level up to the moment of greater water inflow into the pit than the maximum pumping capacity of the BCSD.

4 How to process slurry from dredge mining

The mined slurry must be treated in 2 different ways depending if it is waste or ore slurry.

4.1 Treatment

The ore slurry produced by dredging has to be treated in such a way that its characteristics, with minimum additional treatment stages, equal material mined with traditional mining methods. That allows using the same downstream treatment arrangement.

The treatment differences of traditional mined ore and dredged ore and certainly their proper advantages and disadvantages will afterwards be shown.

4.1.1 Treatment of traditional mined material

Traditional mining methods have a more or less dry mining product, which is an important parameter at the entry of the processing plant. Furthermore, the grain size is not constant and needs to be crushed at the start. The flow sheet for treatment of traditional mined ore (Illustration 53) will here be presented. This allows for an easier understanding and a further comparability to the other treatment scenarios.

The flow quantities between the treatment units change because of different machine availabilities which are directly converted into the flows (Table 21). The availability for crushing units is 75% and the availability for grinding units is 80%.

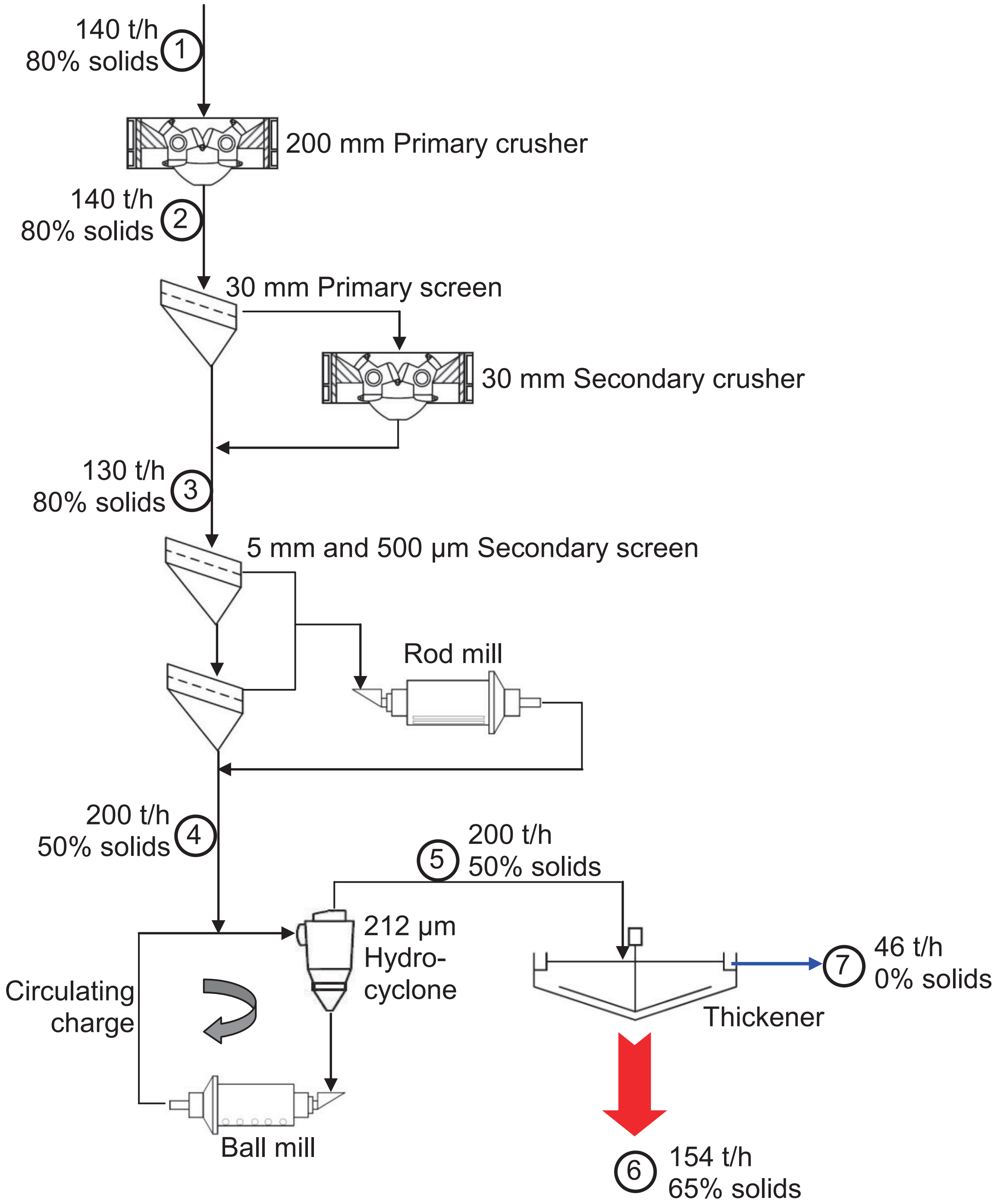


Illustration 53: Flow sheet for traditional mined ore

Traditional	1	2	3	4	5	6	7
Solid							
t/h	112	112	104	100	100	100	0
Density [t/m³]	2,65	2,65	2,65	2,65	2,65	2,65	2,65
Liquid							
t/h	28	28	26	100	100	54	46
Density [t/m³]	1	1	1	1	1	1	1
Slurry							
t/h	140	140	130	200	200	154	46
Slurry density [t/m³]	2,0	2,0	2,0	1,45	1,45	1,68	1
% solid	80	80	80	50	50	65	0

Table 21: Treatment flows for traditional mined ore

In any case traditional mined ore has to be crushed in a primary crusher. The humidity of the material at the entry of the processing plant is approximately 20%. After the primary crushing, all materiel will be screened at 30 mm. The oversized particles will be crushed in a secondary crusher. The underflow of the screen and the crushed material will go to a secondary screen unit. All material which does not pass the 500 µm screen comes to a rod mill. The underflow of the 500 µm screen and the milled material will be forwarded to a cyclone classification system. Water will be inserted to the rod mill feed for cyclone classification. The cyclone system separates at 212 µm. All material that is bigger than that separating size goes into a ball mill in close circuit. The circulating charge is estimated at approximately 300 to 400% of the secondary screen and crushing discharges. All material which is smaller than 212 µm goes into a neutral thickener. Neutral thickener feeds slurry that a have solid content of 50% and will be concentrated up to 65% in the underflow.

4.1.2 Treatment of dredged ore slurry

The aim of the treatment of slurry mined by a dredger is to obtain approximately equal products at the exit of the neutral thickener. The larger water quantities in the overflow demand an additional treatment in the case of slurry with a lot of fine materials for a further recuperation or rejection of the water.

The scenarios will only be qualitatively comparable due to different assumptions concerning baseline dimensions, like production rates or fine proportion.

The treatment of dredged ore slurry will be examined according to four scenarios:

- 20% of solids in slurry and few fines (Illustration 55 and Table 23);
- 30% of solids in slurry and few fines (Illustration 56 and Table 24);
- 20% of solids in slurry and much fines (Illustration 57 and Table 25);
- 30% of solids in slurry and much fines (Illustration 58 and Table 26).

The scenarios with a lot of fines probably have more than 80% of very fine material smaller than 212 µm. This assumption is based on the results of screening tests (Illustration 54) which were done with material coming from different depths of the Patricia and Pato-Pama deposits (Table 22).

Grain size	Deposit and depth of material						Average
	Patricia				Pato-Pama		
	4 m	15 m	20 m	50 m	85 m	90 m	
% <100µm		90,24		90,42	83,35	91,4	88,85
% <250µm	57,74	91,48	83,94	98,42	90	94,45	86,01
% <500µm	67,56	93,48	87,80				82,95
% >250µm				1,58	10	5,55	5,71
% >500µm	32,44	6,52	12,20				17,05

Table 22: Granulometry table of sieving

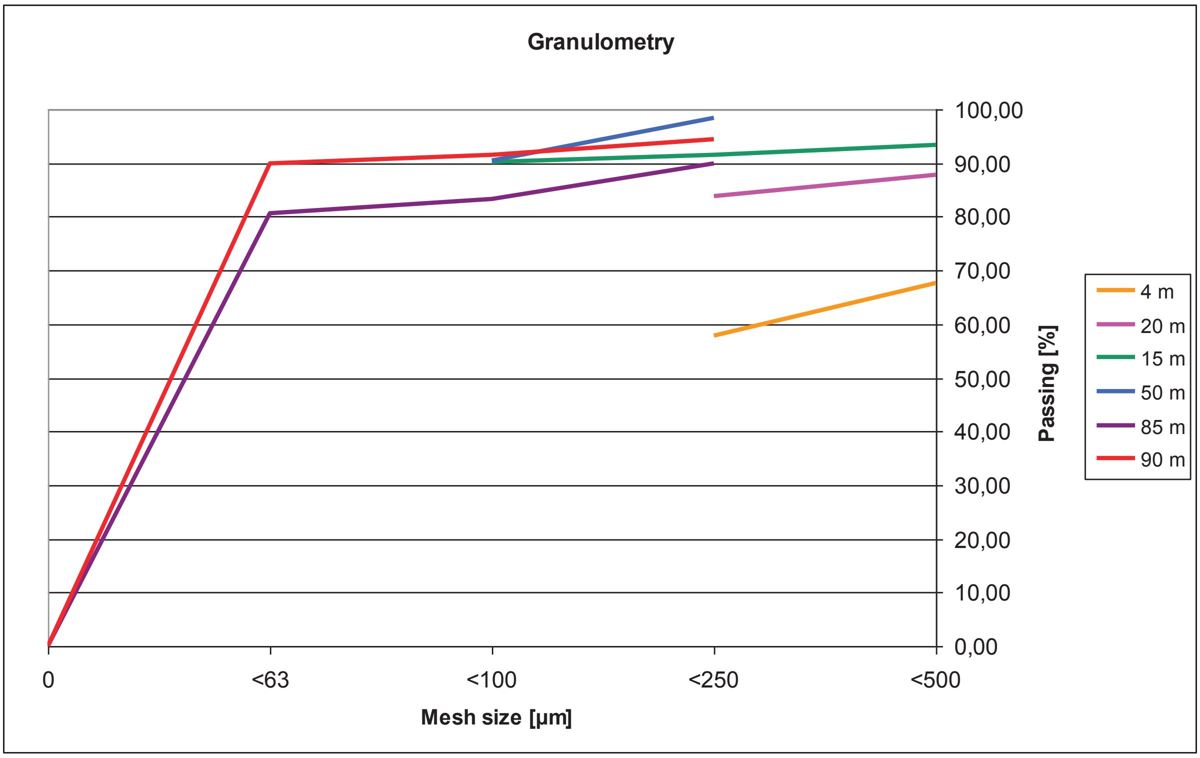


Illustration 54: Particle size distribution

The big percentage of very fine material allows eliminating or reducing crushing and milling steps.

The flow quantities between the treatment units change because of different machine availabilities which are directly converted into the flows.

4.1.2.1 Ore slurry with 20% of solids and few fines

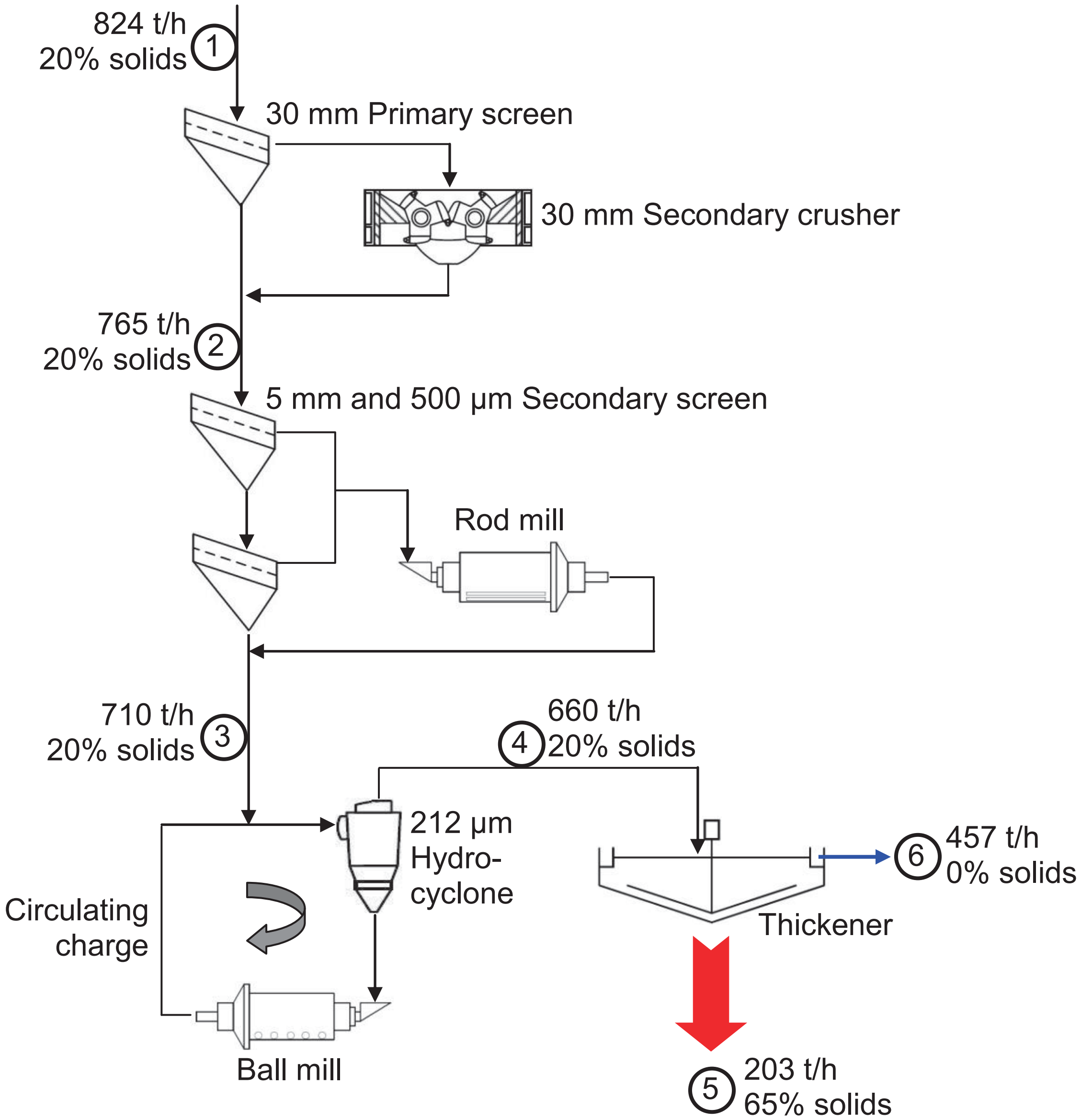


Illustration 55: Flow sheet for dredged slurry with 20% of solids and little fine material

Dredged ore 20% solids	1	2	3	4	5	6
Run of mine						
t/h	165	153	142	132	132	0
Density [t/m³]	2	2	2	2	2	2
Liquid						
t/h	659	612	568	528	71	457
Density [t/m³]	1	1	1	1	1	1
Slurry						
t/h	824	765	710	660	203	457
Slurry density [t/m³]	1,11	1,11	1,11	1,11	1,48	1
% solid	20	20	20	20	65	0

Table 23: Treatment flows for dredged slurry with 20% of solids and little fine material

4.1.2.2 Ore slurry with 30% of solids and few fines

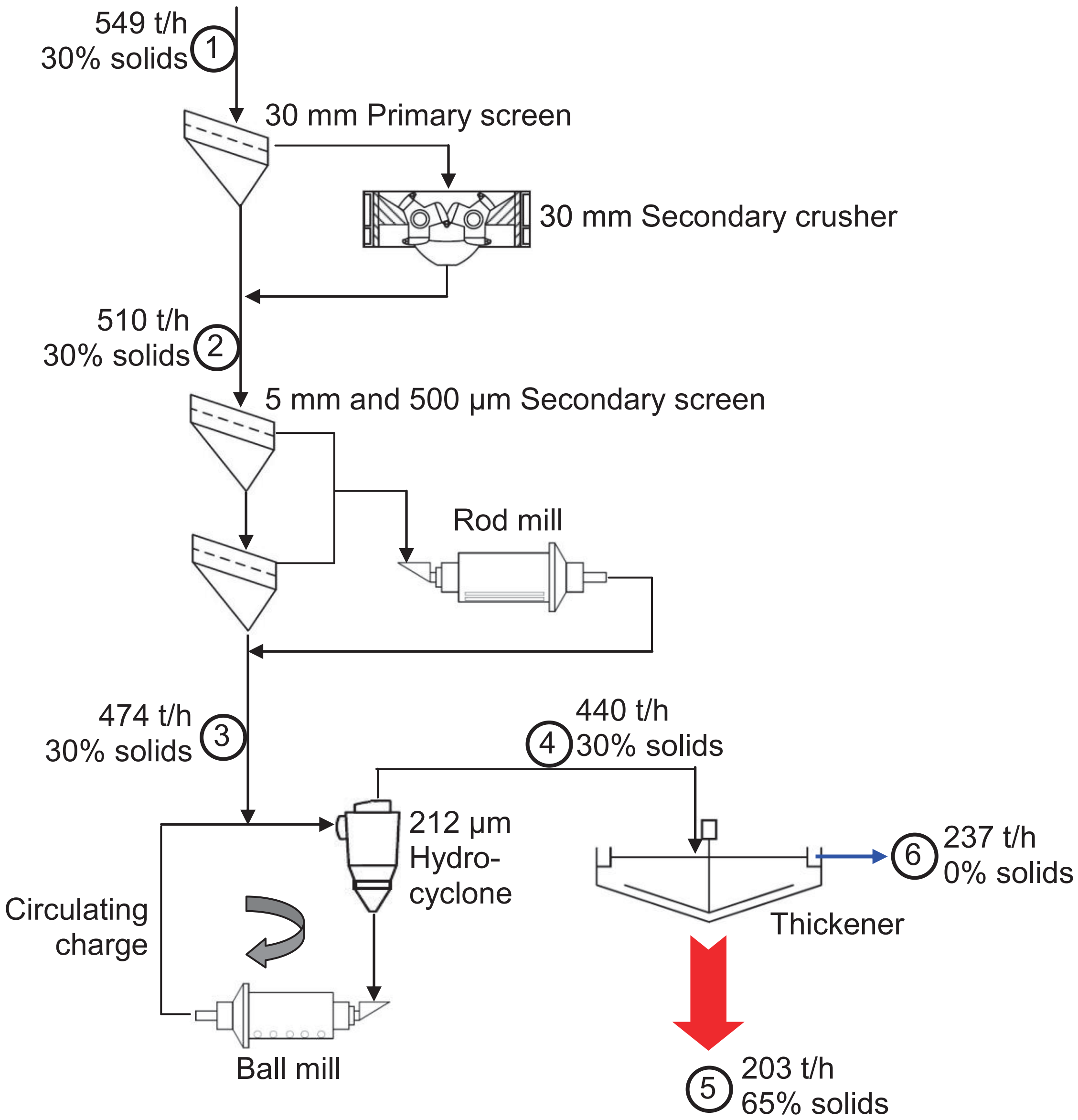


Illustration 56: Flow sheet for dredged slurry with 30% of solids and little fine material

Dredged ore 30% solids	1	2	3	4	5	6
Run of mine						
t/h	165	153	142	132	132	0
Density [t/m³]	2	2	2	2	2	2
Liquid						
t/h	384	357	331	308	71	237
Density [t/m³]	1	1	1	1	1	1
Slurry						
t/h	549	510	474	440	203	237
Slurry density [t/m³]	1,18	1,18	1,18	1,18	1,48	1
% solid	30	30	30	30	65	0

Table 24: Treatment flows for dredged slurry with 30% of solids and little fine material

Plant circuits for dredged ore do not demand a primary crusher because the dredger already breaks the material at a maximum grain size of 200 mm.

If it is assumed that the dredged product has only few fines, the process flow sheet from the primary screen is very similar to the traditional scenario. The main difference concerns the water quantities, which are a lot higher for the dredging scenario.

The bigger part of the water will directly pass through the primary screen. The 30 mm overflow, which goes directly to the secondary crusher, would have approximately a humidity of 10%. The overflow of the secondary screen would probably have a humidity of 25%.

If thickener overflow is clear enough, it will be recycled into the plant for internal use, or directly forwarded to water treatment for radium removal before discharging into the tailings storage facility as it would also be done in a traditional treatment scenario.

The downstream ore treatment will be equal to the traditional treatment process and will not be presented here.

4.1.2.3 Ore slurry with 20% of solids and much fine material

The thick line in Illustration 57 represents the main flow.

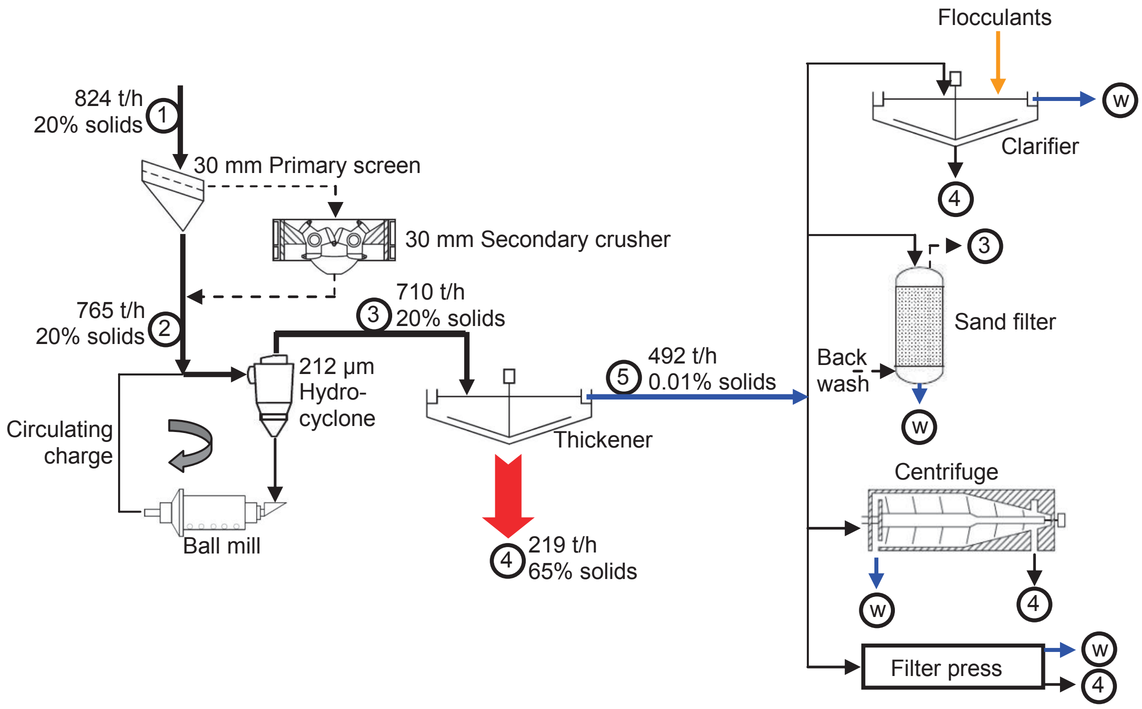


Illustration 57: Flow sheet for dredged slurry with 20% of solids and much fine material

Dredged ore 20% solids	1	2	3	4	5
Run of mine					
t/h	165	153	142	142	0,049
Density [t/m³]	2	2	2	2	2
Liquid					
t/h	659	612	568	76	492
Density [t/m³]	1	1	1	1	1
Slurry					
t/h	824	765	710	219	492
Slurry density [t/m³]	1,11	1,11	1,11	1,48	1
% solid	20	20	20	65	0,01

Table 25: Treatment flows for dredged slurry with 20% of solids and much fine material

4.1.2.4 Ore slurry with 30% of solids and much fine material

The thick line in Illustration 58 represents the main flow.

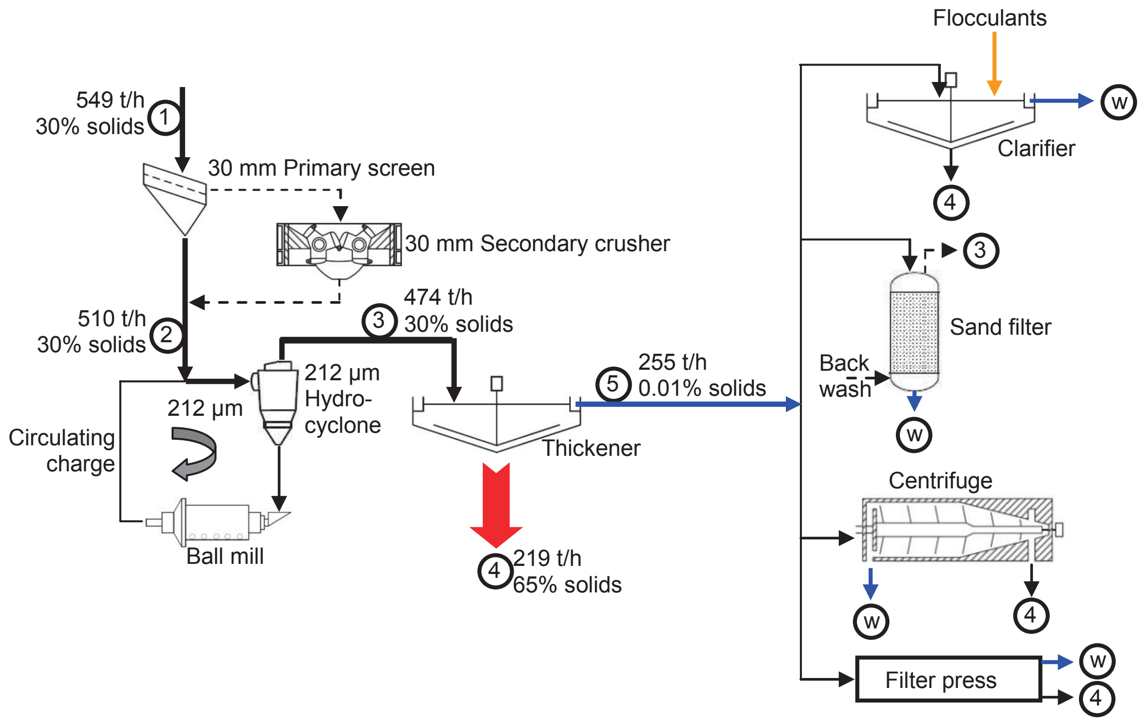


Illustration 58: Flow sheet for dredged slurry with 30% of solids and much fine material

Dredged ore 30% solids	1	2	3	4	5
Run of mine					
t/h	165	153	142	142	0,025
Density [t/m³]	2	2	2	2	2
Liquid					
t/h	384	357	331	76	255
Density [t/m³]	1	1	1	1	1
Slurry					
t/h	549	510	474	219	255
Slurry density [t/m³]	1,18	1,18	1,18	1,48	1
% solid	30	30	30	65	0,01

Table 26: Treatment flows for dredged slurry with 30% of solids and much fine material

In any scenario, the secondary crusher has to stay in the process because of the possibility of having grains bigger than 30 mm in the dredged material, even if lots of fines are expected.

The main part of the flow will directly pass the cyclone and go to the neutral thickener. The flow of material bigger than 212 µm represents only a very small part and will stay in the circulating charge of the ball mill. The rod mill has been removed from those two circuits, since smaller milling requirements are expected.

The water in the overflow of the thickener could still contain a significant amount of solids because of longer sedimentation times of very fine material. This requires an additional water treatment stage.

The treatment of the water could be done in different ways:

Treatment units and function for success	
Clarifier	Sedimentation rate; volume of slurry to be treated; flocculant addition
Sand filter	Volume of slurry to be treated; particle size distribution
Centrifuge	Volume of slurry to be treated
Filter press	Mass of solids; filterability of solids

The efficiency of the individual systems depends on the mass or volume of solids, sedimentation rate, particle size distribution, filterability of solids, and the addition of flocculant during sedimentation.

4.1.3 Treatment of waste slurry

In a traditional mining scenario, the waste can directly be transferred to a waste dump. In the case of dredged material the mined product is slurry and has to be dewatered before bringing it to a waste dump.

We have to differ between 20 and 30% of solids in the waste slurry (Table 27 and 28).

Waste scenario 20%	1
Run of mine	
t/h	1583
Density [t/m³]	2
Liquid	
t/h	6332
Density [t/m³]	1
Slurry	
t/h	7915
Slurry density [t/m³]	1,11
% solid	20

Table 27: Waste slurry with 20% of solids

Waste scenario 30%	1
Run of mine	
t/h	1583
Density [t/m³]	2
Liquid	
t/h	3694
Density [t/m³]	1
Slurry	
t/h	5277
Slurry density [t/m³]	1,18
% solid	30

Table 28: Waste slurry with 30% of solids

The amounts of fine material demand different treatment arrangements. However the first steps of the waste treatment could be reasonably assumed to be a combination of screens and cyclones.

The water content in the oversize would be low enough to transport the material with a band-conveyor directly to a stockpile. The passing of the screen, which is smaller than 1 mm and the main part of the water coming along, are forwarded to a cyclone with a separating size of 100 µm. The coarse material of the cyclone, which is expected to have a 60 to 70% solid content, can be combined with coarse material from the primary screen. Water would constitute the main part of the overflow of the cyclone system with fine material smaller than 100 µm. Water that is not clean enough is further treated before being released into the environment.

The advantage of this option is the compactness of the required machines. For the first mining site, a stockpile is necessary. After finishing the mining process of the first pit the waste material of a new mining site could be stocked in the old pit, for minimizing land consumption.

4.1.3.1 Waste slurry with few fines

Material with few fines could be treated in the following ways:

- Direct decantation in a big basin (Illustration 59);
- Screen and Cyclone + Sand filter (Illustration 60);
- Screen and Cyclone + Drainage (Illustration 60).

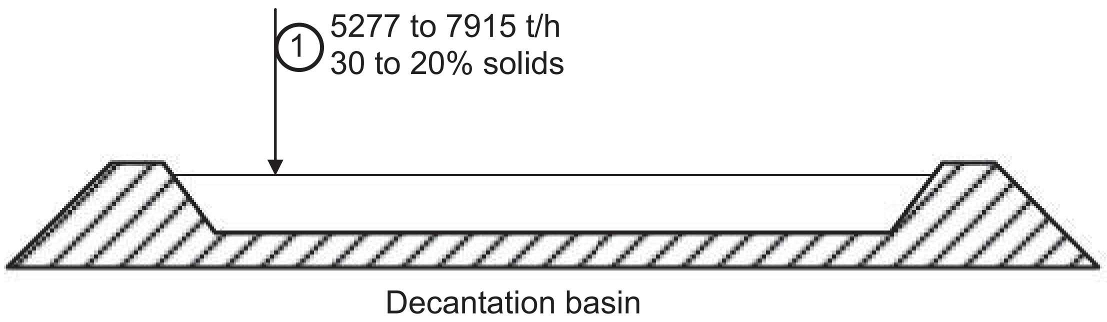


Illustration 59: Decantation basin

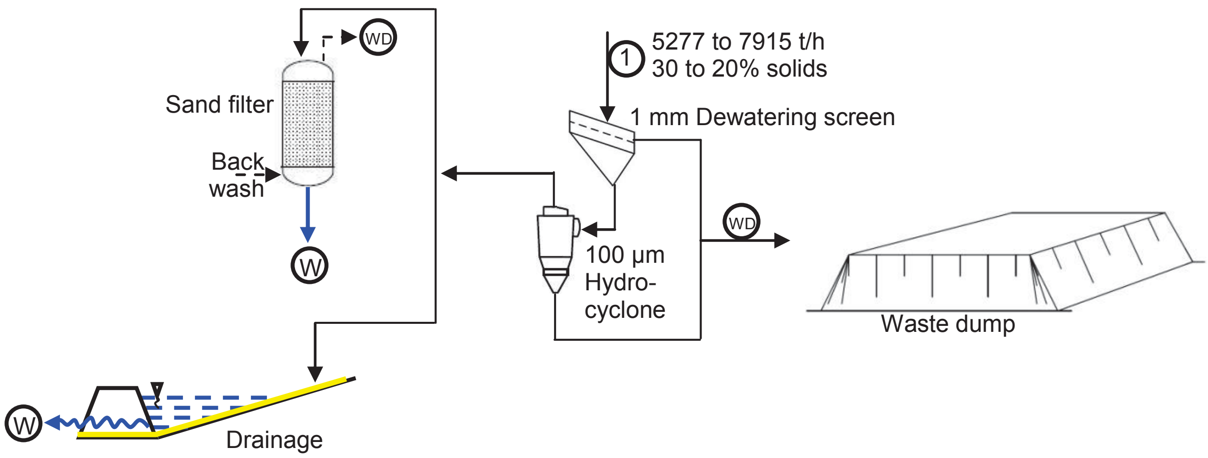


Illustration 60: Treatment of waste slurry with little fine material

4.1.3.2 Waste slurry with much fine material

Very fine material could be treated in the following ways:

- Screen and Cyclone + Thickener and Clarifier (Illustration 61);
- Screen and Cyclone + Thickener and Sand filter (Illustration 61);
- Screen and Cyclone + Thickener and Filter press (Illustration 61);
- Screen and Cyclone + Thickener and Centrifuge (Illustration 61).

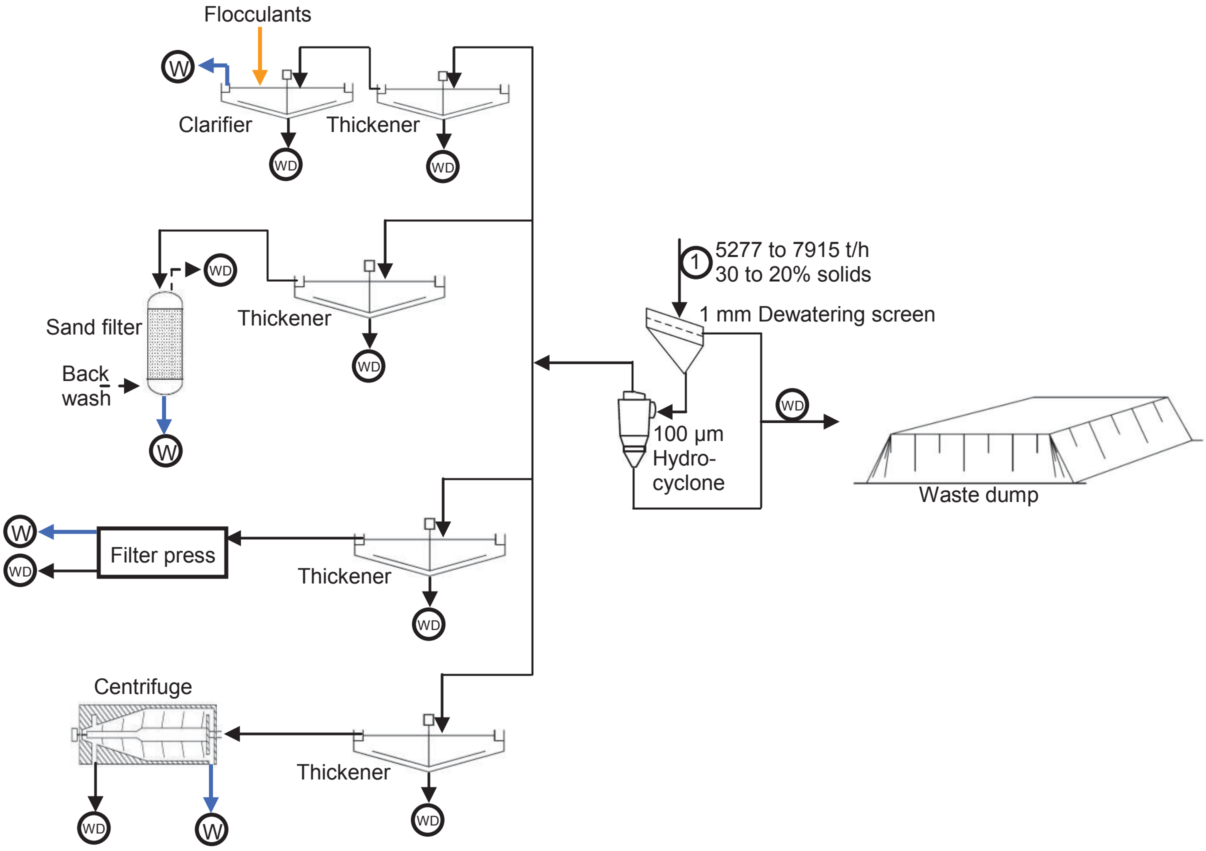


Illustration 61: Treatment of waste slurry with much fine material

The choice of the optimal arrangement depends on the quantities of the fine material, the mass of solids, and the volume to be treated.

4.2 Decantation tests and results

A main treatment aspect for the dredging scenario is the separation of the diluted slurry to a solid concentrated slurry and clear solution. If the slurry contains significant quantities of very fine material, this separating process will be harder to perform because of relatively slow sedimentation rate of very fine particles.

Decantation tests were performed for sizing thickeners. The sedimentation characteristics were studied in different conditions and on different rock samples.

4.2.1 Characterisation of Bakouma rock material

Rock cores from three drillings in Illustration 62 (PRCDD06 1 (Table 29), DAMDDH01 (Table 30), DAMDDH02 (Table 31)) showed a geological evolution of Bakouma rock material with increasing depth. Most rock samples consist mainly of very fine material. Measurement of the velocity was undertaken on solid rock samples through radial and longitudinal measurements.

PRCDD06 1					
0	**0,15**	Silty clay with big quartz grains which are millimeter scattered, colour gray viridescent.			
1,4	**1,5**	Ferruginous duricrust, heterogeneous with a large number of holes and canals from 2 to 5 mm in diameter with fillings and cutaneous of white evaluation.		**Volume mass [g/cm^3]** 2,38 3,35	**Velocity [m/s]** 2974,3 2253,3
4	**4,1**	Silty marbled clay, essentially gray and white with ferruginous nodules which appear residual. Points and white level under the duricrust = degradation of the duricrust.			
5	**5,1**	Clay abradable silt, chalky, colour red purple, containing nodules and centimeter elements, either in white colour or ochre- viridescent ferruginous colour.			
7,4	**7,5**	Sandy white clay red veined, complex structure, contorted, probably by plastic flow through drilling. Contains transparent silica chips and more irregular elements in holes. Residual or neoform silica; general colour of drilling powder is rose.			
10,5	**10,6**	Silty clay, chalky, fine grains, colour rose purple with white stripes, general colour rose-purple.			
14,3	**14,4**	Equally, general color of core is rose.			
15,75	**15,85**	Clayish silt, chalky, rose and purple, with short white strips and ferruginous inclined lines and small holes which are covered with iron oxides.			
19	**19,1**	Core of 7-8 cm length, which is well preserved, but the internal structure shows that the internal annulus is totally reoriented and the internal of the core shows also shearing and debonding. The initial material was plastic. Silty rose-purple clay with white strips and small iron oxide accumulations.		**Volume mass [g/cm^3]** 2,1	**Velocity [m/s]** 824,2
20	**20,1**	Clayish white-beige silt, with some purple iron traces; small very fine silica discs = silica fossils.			

Table 29: Geological identification of PRCDD06 1 drill cores

DAMDDH01					
1	**1,1**	Clayish-sandy ground.		**Volume mass [g/cm^3]** 1,72	**Velocity [m/s]** 1505,9
3,3	**3,4**	Sandy clay, clear white, containing granulates and goethite iron nodules with morphologic irregularities; core is very heterogenic.			
5,5	**5,6**	Iron poriferous duricrust with nodules and argilomorphe, laminating facies, white fillings in some holes.		**Volume mass [g/cm^3]** 2,85	**Velocity [m/s]** 1615,8
10,3	**10,4**	Clayish silt, chalky, red with flies and small white veins; deformed structure through drilling.			
15	**15,1**	Clayish silt, chalky, fine, clear colour, beige and rose with very fine silica discs.			
20	**20,1**	Clayish silt, chalky, fine, white with small red and purple stripes.			
25	**25,1**	Clayish silt, chalky, clear colour, white and red stripes. Some silica discs. Lots of figures of plastic flow of core.		**Volume mass [g/cm^3]** 2,18	**Velocity [m/s]** 801,6
30	**30,1**	Chalky silt, red-purple ferruginous with flies and small white stripes.			
35,1	**35,2**	Chalky silt, rose-purple and white, dominant colour is clear.		**Volume mass [g/cm^3]** 2,03	**Velocity [m/s]** 647,3
40	**40,1**	Sandy silt, chalky, colour rose-rubiginous; ferruginous ochre accumulations and white lines. The structure is destroyed through drilling.		**Volume mass [g/cm^3]** 2,24	**Velocity [m/s]** 1068,6
44,7	**44,85**	Clear limestone, beige-white, fractured, red-purple ferruginous fillings in fractures and also white product.		**Volume mass [g/cm^3]** 2,78 2,63	**Velocity [m/s]** 6724 4972,4

Table 30: Geological identification of DAMDDH01 drill cores

DAMDDH02					
1,2	**1,3**	Sandy grey clay, traces of radix, ground.			
5	**5,1**	Sandy white silt.		**Volume mass $[g/cm^3]$** 1,84	**Velocity [m/s]** 1109,1
10,1	**10,2**	Chalky silt, clear white to beige colour with ferruginous red-purple and ochre flies.			
15,3	**15,4**	Chalky silt, rose-rubiginose. Figures of plastic flow of the core.			
20	**20,1**	Clayish silt clear and rose.			
24,9	**25**	Chalky silt, clear, white with centimetre big hard angular and white elements. The hard elements look to be from more resistant sedimentary levels.			
29,5	**29,6**	Calcaire silt, rose-rubiginose, homogenous.		**Volume mass $[g/cm^3]$** 2,94	**Velocity [m/s]** 5599,5
33,1	**33,2**	Clair limestone, fine layered, fractures filled with red-purple oxide of iron.			
35	**35,1**	The same, but with break-ups along the fractures.			
39,9	**40**	Clair limestone, fractured, displacement of fractures = 1-2 cm. The fractures are filled with red iron oxides.		**Volume mass $[g/cm^3]$** 2,67 2,78	**Velocity [m/s]** 5112,2 5452,2
43,9	**44**	Massif limestone, fine layered, colour dark grey bluish.		**Volume mass $[g/cm^3]$** 2,82 3,07	**Velocity [m/s]** 6691,1 5584,4

Table 31: Geological identification of DAMDDH02 drill cores

All orange values in Table 29, 30 and 31 for the volume mass are approximations: the volume was considered to be a cylinder with a diameter of 80 mm for the PRCDD061 and 60 mm for the DAMDDH01 and the DAMDDH02 cores. The length of the cylinder was considered to be the difference of the depths.

The orange values for the velocity were obtained through radial measurement. The diameter of the cores was checked and the exact value was introduced in the calculations.

Four rock samples were hard enough to cut their ends to obtain even surfaces. The exact volume of these cores was calculated and the velocity was measured longitudinally (Table 32).

Drill core	Depth [m]		Volume mass [g/cm^3]	Velocity [m/s]
	from	to		
PRCDD06 1	1,4	1,5	2,38	2974,3
DAMDDH01	44,7	44,8	2,78	6724,0
DAMDDH02	39,9	40	2,67	5112,2
DAMDDH02	43,9	44	2,82	6691,1

Table 32: Longitudinal velocity test results

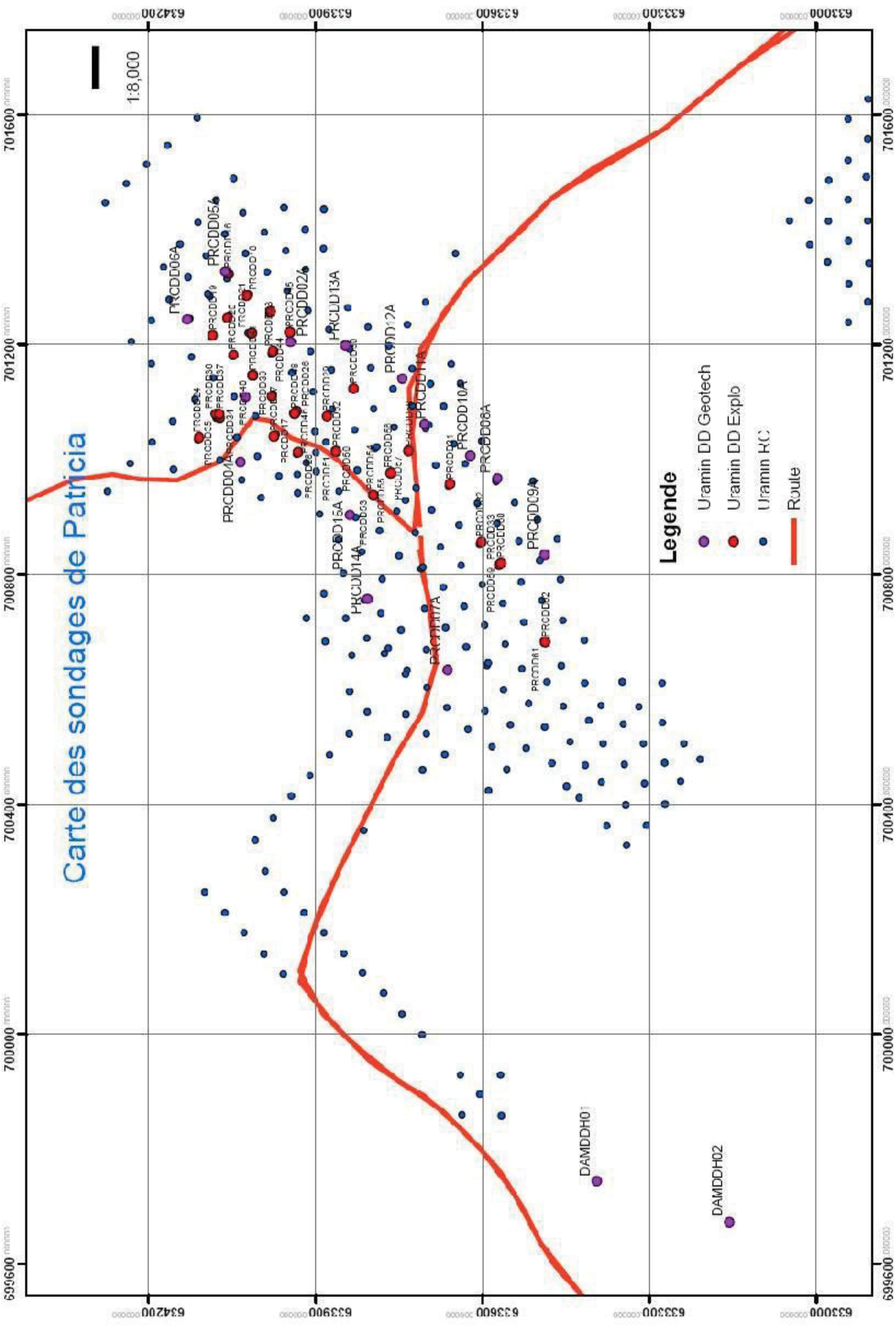


Illustration 62: Positions of drill soundings

4.2.2 Test 1: Laterit and Saprolite rock samples

For defining decantation characteristics, 500 g of laterit and saprolite rock samples were tested.

The rock samples, originally humid, were dried out during the time of storing. During this process, non-natural nuggets were formed. These nuggets were dispersed, by putting the whole material into distilled water for 3 h.

During the sedimentation process, bigger grains tend to settle immediately. Main time is for the decantation of the very fine material (Illustration 67). That is why the whole test samples were wet-sieved first at 500 µm and then at 250 µm. The oversize was dried at 30 °C over 4 days and weighed afterwards (Table 33):

PRCDD061 (Laterit)		
4-4,1		
Total mass	500	g
Mass >500µm	162,2	g
Mass <500µm >250µm	49,1	g
Mass <250µm	288,7	g
PRCDD061 (Saprolit)		
20-20,1		
Total mass	500	g
Mass >500µm	61	g
Mass <500µm >250µm	19,3	g
Mass <250µm	419,7	g

Table 33: Granulometry of test samples for decantation test 1

The slurry containing all material smaller than 250 µm was put into several beakers. For reducing the water volume to at the outmost 2 l for each test sample, the slurry was centrifuged for 1 h.

For the decantation tests, the slurries were given into graduated cylinders (Illustration 63) and the separating line (Illustration 64) between supernatant and slurry was observed. The material compound at the base of the test tubes packed together, but even after one week the water at the top of the tubes was not clear (Illustration 65 and 66).

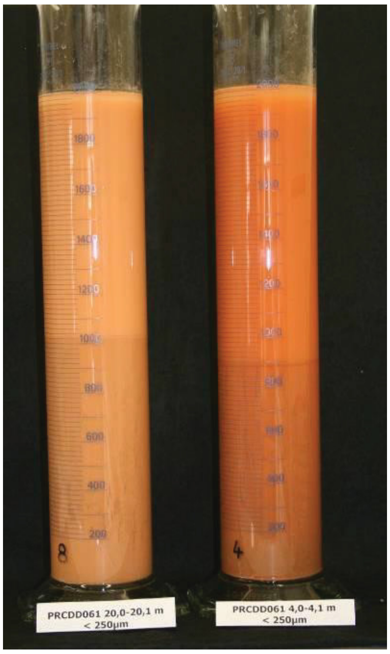


Illustration 63: Saprolite (left) and Laterit (right) sample in graduated cylinders

Illustration 64: Separating line of Laterite sample after 24 minutes

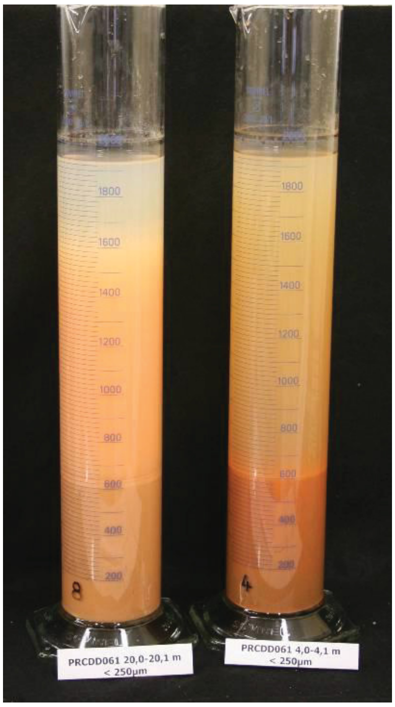
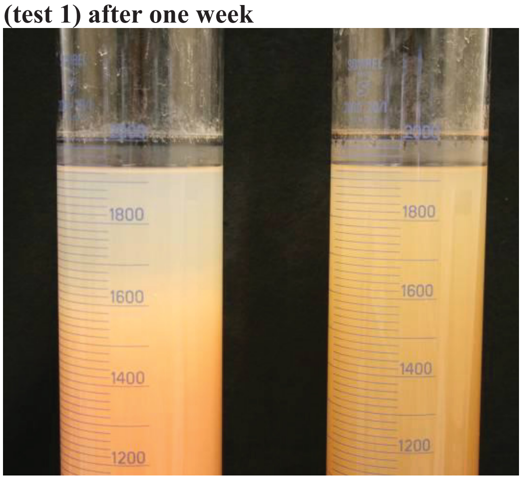


Illustration 65 and 66: Decantation test progress (test 1) after one week



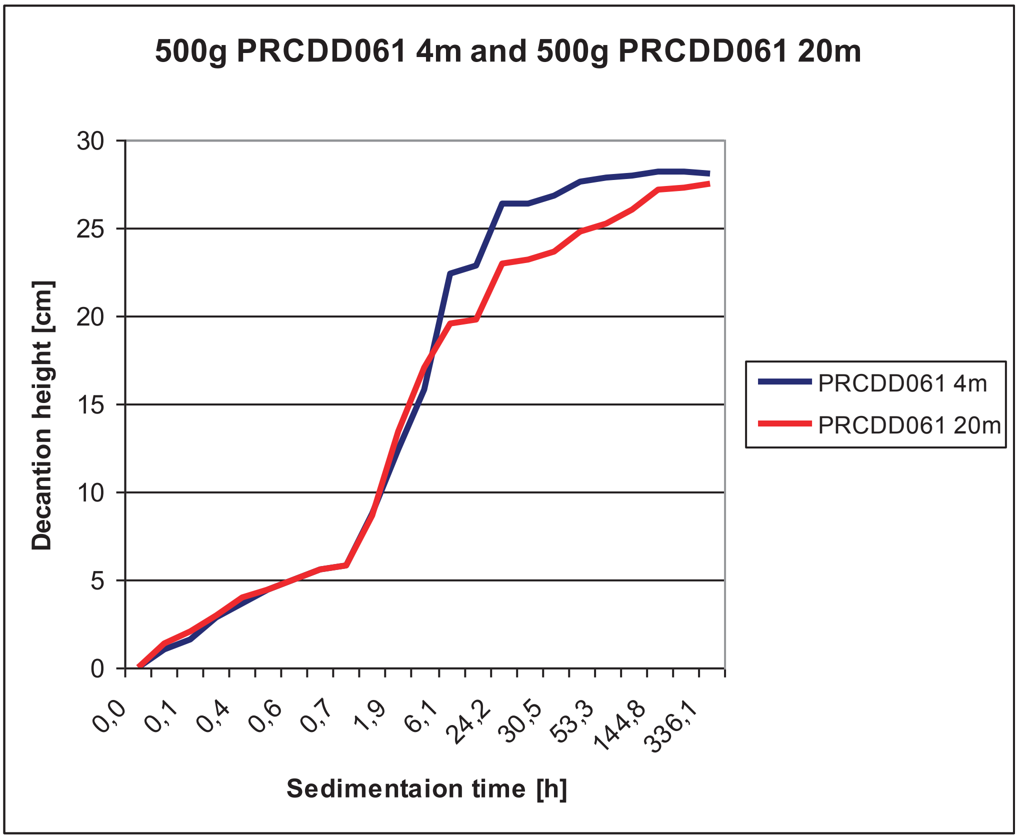


Illustration 67: Decantation time for test 1

These tests had three critical points for being considered as representative:

- the rock samples are not representative of the deposit, since they have been cut from the border of the deposit and only from the most superficial layers (alteration zone). Test samples coming from a deeper position of the deposit would probably behave differently;
- the initial concentration (500 g less oversizing in 2 l) does not represent a value of a slurry in industrial conditions. The industrial slurry would probably decant faster (thixotropy);

288.7 g/2l	19% solid in slurry
419.7 g/2l	13.5% solid in slurry

- the suspension was made with distilled water. Natural water or tap water normally allows for decanting faster.

4.2.3 Test 2: Saprolite sample in different water conditions

Decantation characteristics of a saprolite test sample under different water conditions were defined. Therefore a 500 g rock sample coming from the Patricia deposit was wet sieved and all material smaller than 100 µm (Table 34) was split out in five glass cylinders.

PRCDD061		
15,75-15,85		
Total mass	500	g
Mass >500µm	32,6	g
Mass <500µm >250µm	10	g
Mass <250µm >100µm	6,2	g
Mass <100µm	451,2	g

Table 34: Granulometry of sample for decantation test 2

Five tests with unequal solution qualities, which means a basic NaOH, a $CaCl_2$ and acid solution were prepared. For comparisons we also prepared test tubes with distilled water and tap water (Table 35 and Illustration 68 and 69).

Acid solution	pH acid	12,61	g/l
NaOH solution	pH basic	5,19	ml/l
$CaCl_2$ solution		11,1	g/l
Distilled water			
Tap water	carbonates		

Table 35: Solutions for test 2

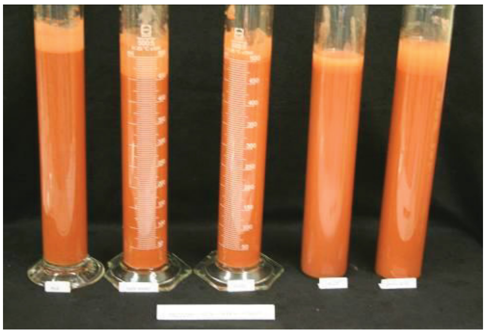



Illustration 68 and 69: Decantation test 2 at the beginning (left) and after 5 hours (right); Samples from left to right in H_2O, tap water, $CaCl_2$, NaOH, oxalic acid

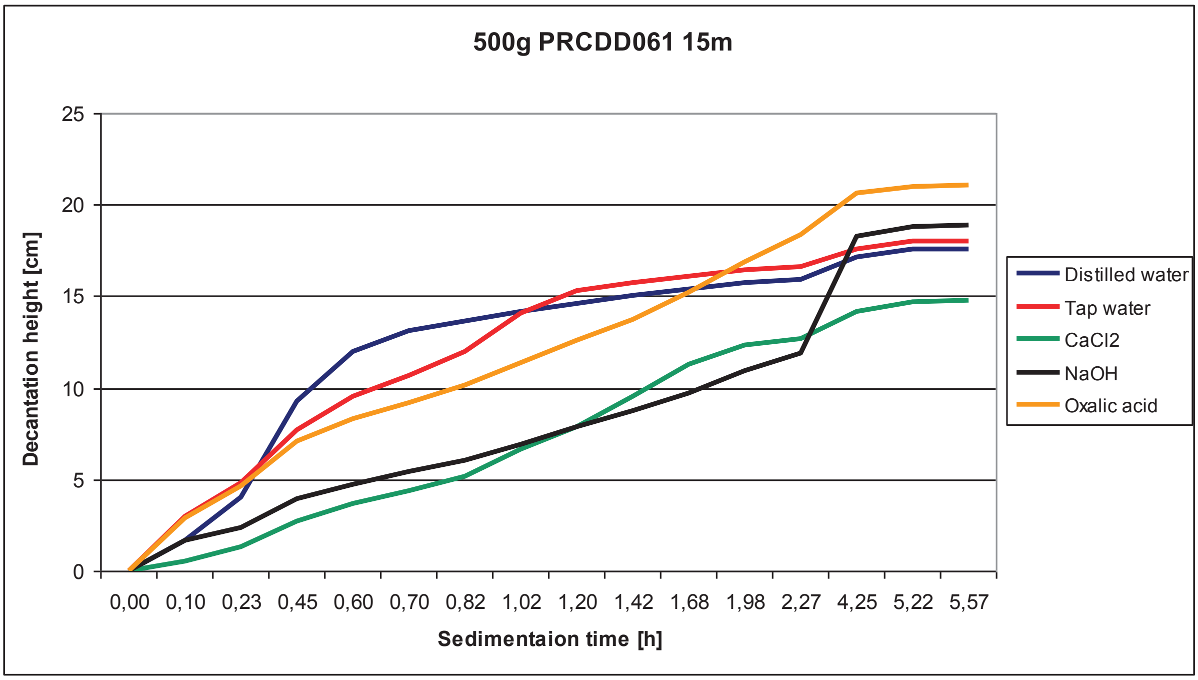


Illustration 70: Decantation time for test 2

All test samples decanted immediately (Illustration 69) and approximately uniformly (Illustration 70). This fact could be ascribed to a liquid which was used during the drilling and which modified the rock samples. At the same time it shows that it is possible to reach fast decantation.

No comparison can be done due to the uniform behaviour of all the test samples.

4.2.4 Test 3: Saprolite sample in different water conditions

The rock samples coming from the Patricia deposit were equally prepared as rock samples for test 2. Material smaller than 250 µm was partitioned in 5 test tubes (Table 36).

PRCDD061		
20-20,1		
Total mass	500	g
Mass >500µm	61	g
Mass <500µm >250µm	19,3	g
Mass <250µm	419,7	g

Table 36: Granulometry of sample for test 3

NaOH, $CaCl_2$ and acid solutions were prepared. For comparisons we also prepared test tubes with distilled water and tap water (Table 37).

Distilled water			
Tap water	carbonates		
$CaCl_2$ solution		11,099	g/l
Acid solution	pH acid	12,607	g/l
NaOH solution	pH basic	5,19	ml/l

Table 37: Solutions for test 3

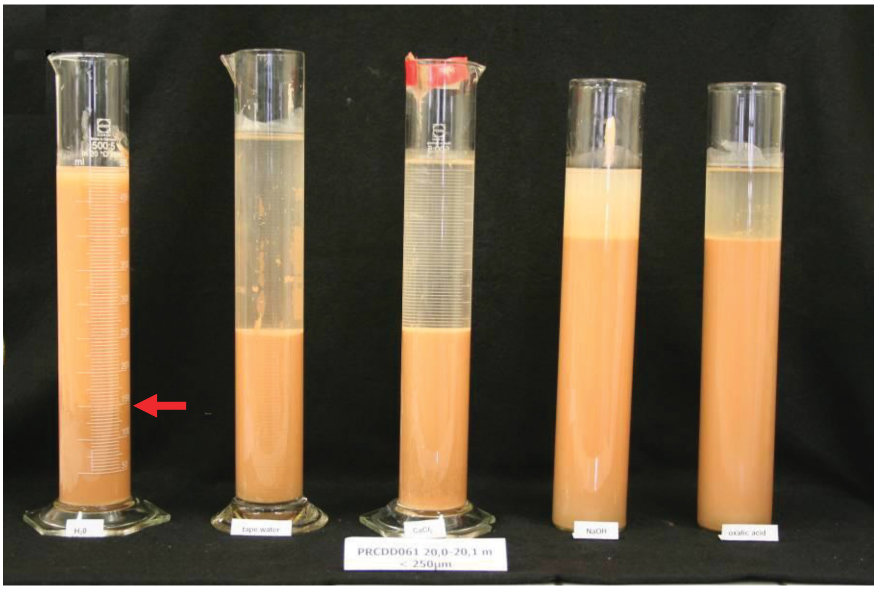


Illustration 71: Decantation test 3 from left to right sample in H_2O, tap water, $CaCl_2$, NaOH, oxalic acid

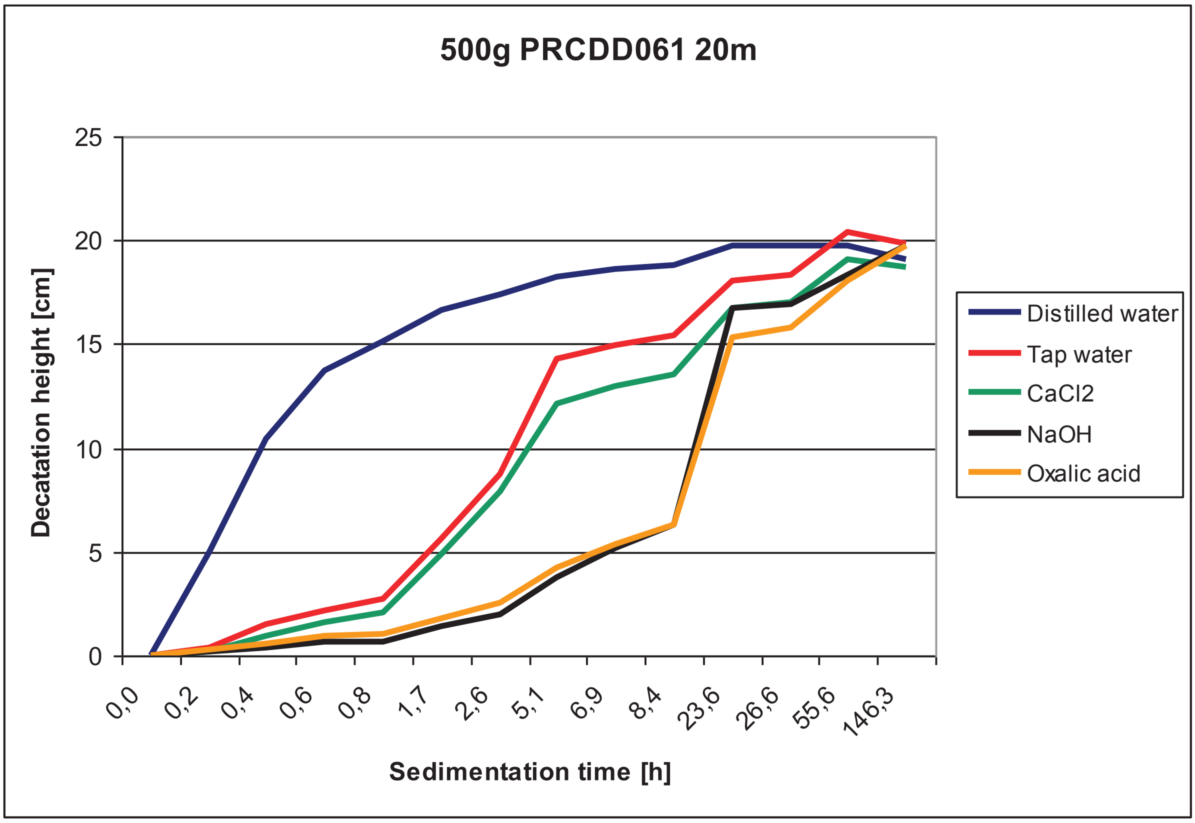


Illustration 72: Decantation time for test 3

In comparison to test 2 we were able to find out different behaviours between the decantation tests (Illustration 71). The slurries with NaOH and oxalic acid had similar behaviours as well as the $CaCl_2$ and tap water test slurries. To define a decantation progress the separating line between supernatant and slurry was read. It seemed that the distilled water test slurry showed the best decantation behaviours, but the supernatant at the top of the cylinder stayed cloudy and could not be considered to be clear.

The tap water and the $CaCl_2$ test tubes showed the best decantation results. The good decantation with the tap water can be explained by the carbonate content, which appears in the tap water in the laboratory (Illustration 72).

4.2.5 Test 4: Saprolite sample coming from the deposit

The fourth test series was used to approach real conditions. That means we produced test slurries with 20 and 30% per weight of solids.

A sieving operation allowed for defining the percentage of material bigger than 100 µm. In other words, 500 g material coming from a depth of 50 m from the Patricia deposit were wet-sieved at 100 µm and dried again. This operation showed that approximately 10% of the material is bigger than 100 µm (Table 38).

Total mass	500	g
Mass >250µm	7,9	g
Mass <250µm >100µm	40	g
Mass <100µm	452,1	g

Table 38: Granulometry of sample for test 4

For the decantation test we used 200 + 20 g of original material for each test tube. The additionally-added 20 g represents the 10% of oversize material bigger than 100 µm, which would decant immediately.

Limestone was used as flocculant in two solutions one with 30% and another with 20% per weight solids. For comparison we prepared a 30% solid solution in tap water and another in distilled water (Table 39).

Liquid		pH	% solids
Limestone	1.2g	11.83	20
Limestone	0.2g	10.45	30
Distilled water		6.16	30
Tap water		6.82	30

Table 39: Details of test solutions

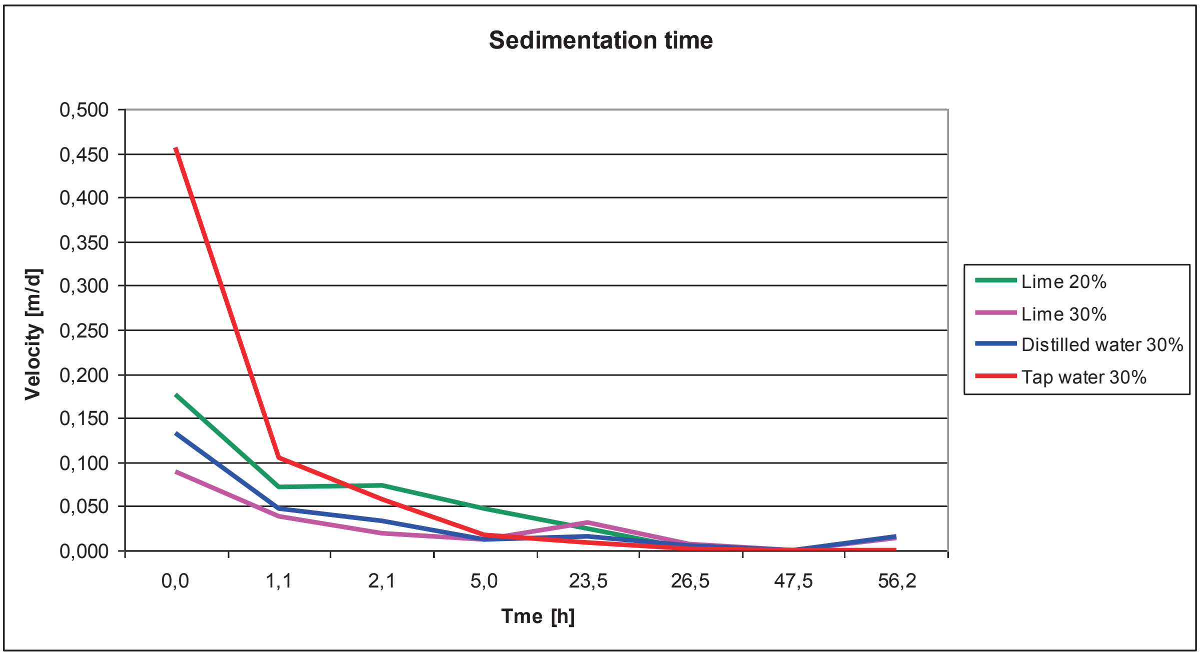


Illustration 73: Decantation velocity

The initial decantation velocity was the highest in tap water. This phenomenon can be explained by the carbonates that are in the tap water and which behave as flocculant. The decantation velocities of the test tubes with distilled water and with limestone as flocculant behave similarly. After one day, the decantation process barely progressed and was almost finished (Illustration 73).

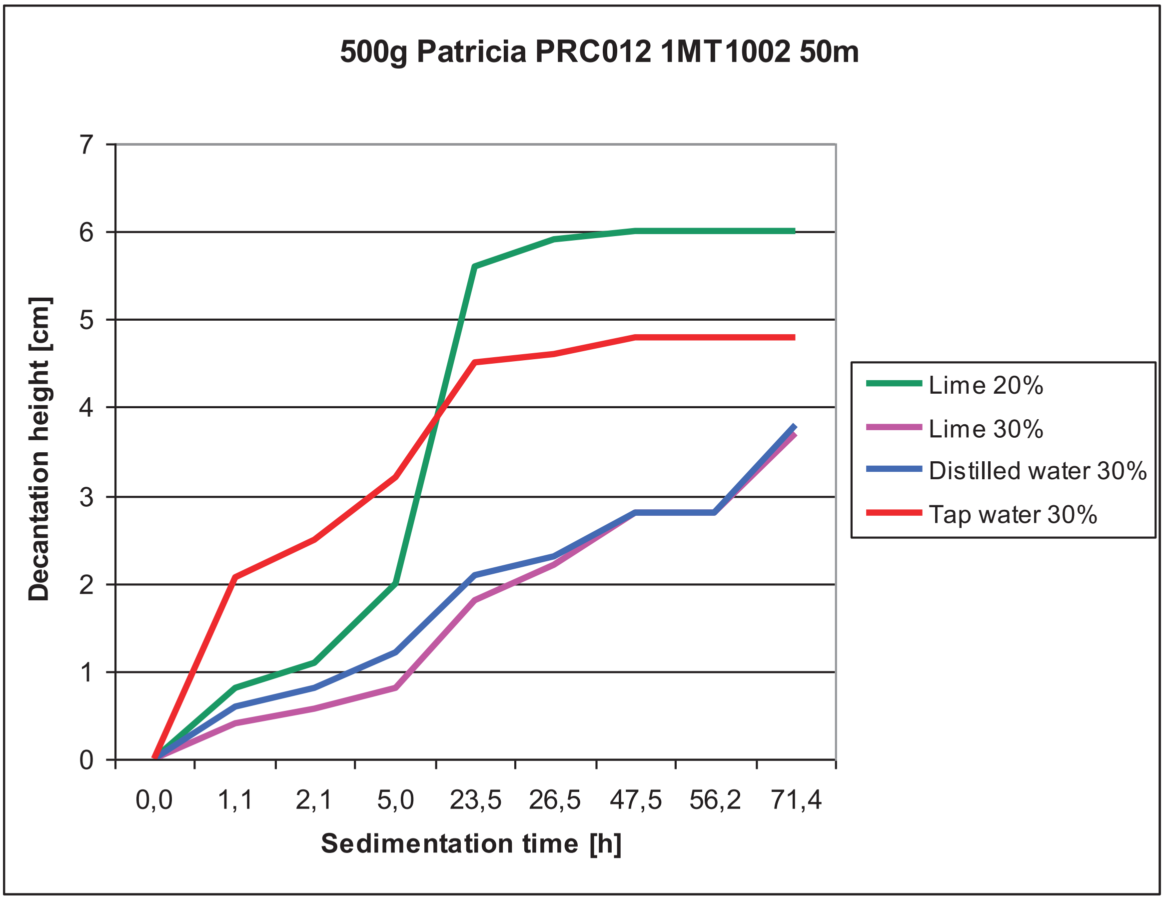


Illustration 74: Decantation time for test 4

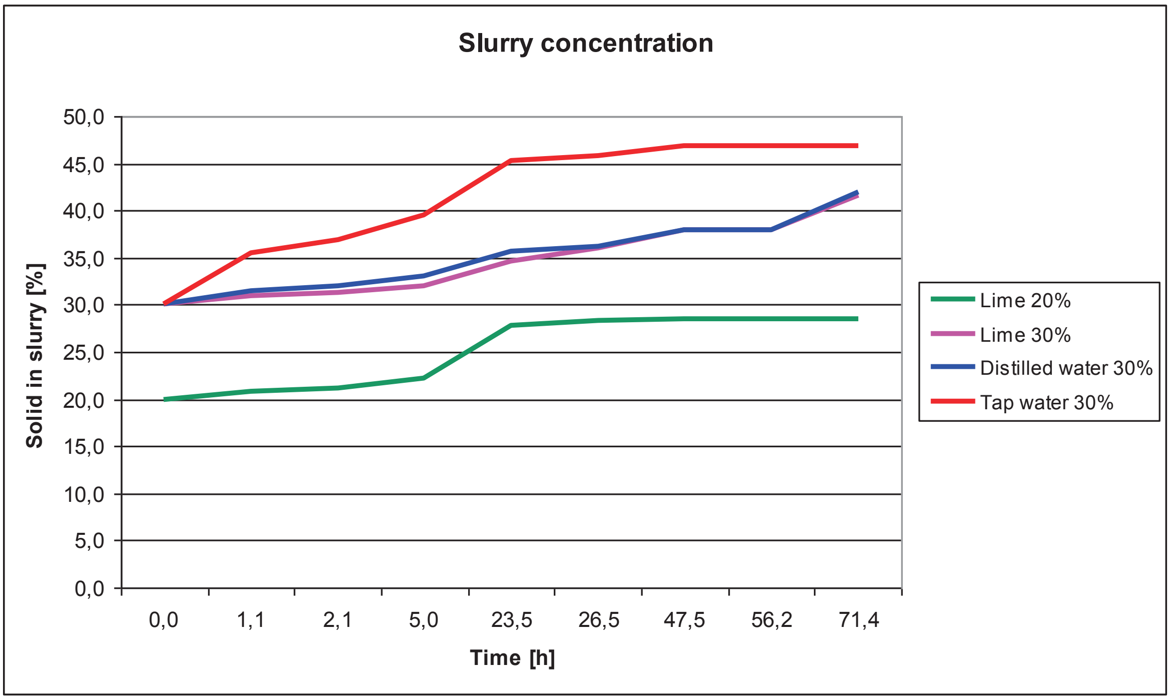


Illustration 75: Slurry concentration

The best decantation results occur in tap water. The concentration of solid in slurry rises during 2 days from 30 to 47%. The test with distilled water and the test with flocculant do behave very similarly. After 2 days, their concentration raised from 30 to 37% of solids in slurry. The test with 20% of solids raised its initial concentration in the slurry to 28% during 2 days (Illustration 75). That means even if the decantation height seems to be bigger for the test with 20% of solids and flocculants, the concentration in the slurry is approximately the same as in the distilled water test and the test with 30% of solids and flocculants (Illustration 74).

This result shows that limestone is not an appropriate flocculant for the Bakouma samples. Its flocculation ability is however fairly well known for sulphide minerals. An inappropriate flocculant can have an adverse effect on sedimentation, since the shape of the resulting aggregate influences sedimentation rate.

However, the decantation behaviour would improve when taking correct flocculants.

4.3 Dimensioning of treatment units

4.3.1 Thickeners

The results of the decantation tests allowed for sizing thickeners. The water in Bakouma does not have carbonates and is similar to our distilled water. That means the unit area of the thickener will be defined by the distilled water curb (Illustration 76).

The unit area was defined from the method of Coe Clevenger for Non-Flocculated Pulps:

$$\frac{m^2}{tpd} = \frac{\frac{1}{C} - \frac{1}{C_u}}{v}$$

$C = \left[\frac{kg}{l}\right]$,test solids concentration

$C_u = \left[\frac{kg}{l}\right]$,underflow solids concentration

$v = \left[\frac{m}{d}\right]$,initial settling rate at test conditions

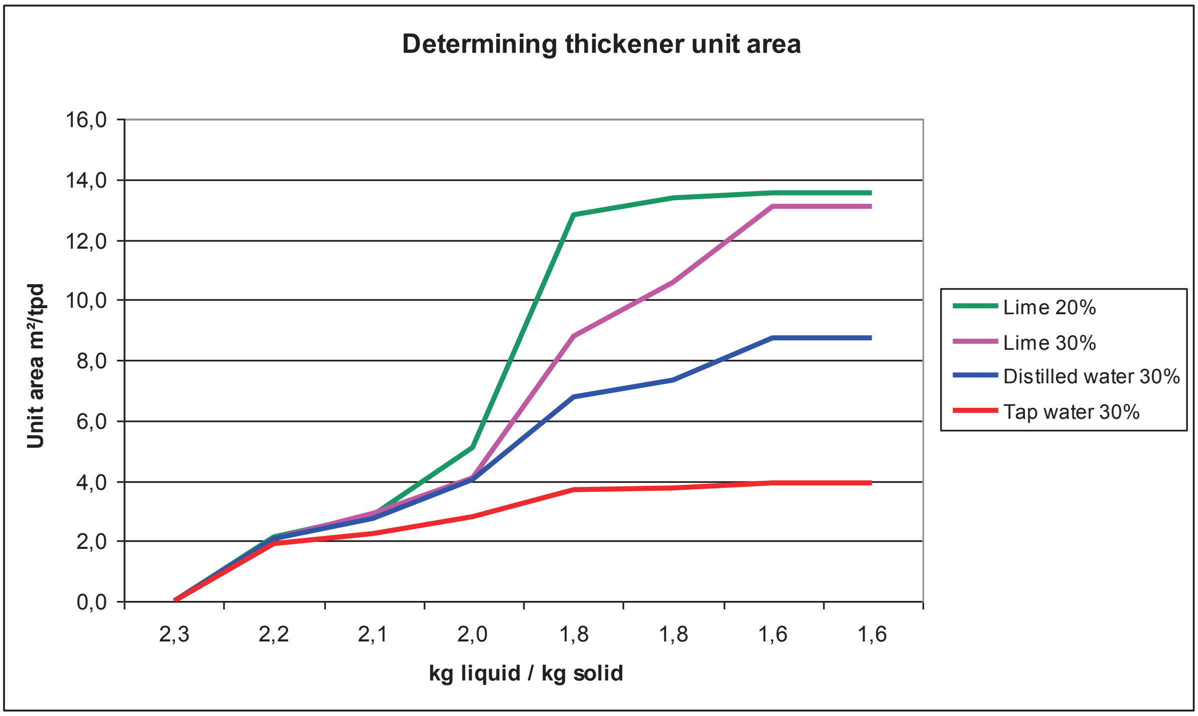


Illustration 76: Determining thickener unit area

The biggest thickeners that are mentioned in the Mine and Mill Equipment Costs book 2008 have a diameter of 122 m. This limit was used for our calculations. The number of needed thickeners depends on the quantity of the total flow, which itself depends on the solid density of the slurry.

The main problem concerns the waste flow, because of its important flow quantities. Depending on the solid content of the slurry, 4 to 6 thickeners will be needed (Table 40).

THICKENER 20% solids		
Needed number of thickeners	6	
Flow 20% solid content	7915	t/h
Flow per thickener	1319	t/h
Unit area	8,7	m²/th
Underflow solid	65	%
Overflow solid	0	%
Thickener surface	11477	m²
Diameter	121	m
THICKENER 30% solids		
Needed number of thickeners	4	
Flow 30% solid content	5277	t/h
Flow per thickener	1319	t/h
Unit area	8,7	m²/th
Underflow solid	65	%
Overflow solid	0	%
Thickener surface	11477	m²
Diameter	121	m

Table 40: Dimensioning of thickeners

4.4 Comparison of treatment scenarios

The five treatment scenarios demand different equipment arrangements. The “base case”-scenario is always the starting point for designing and comparing the treatment scenarios for the dredged slurry (Table 41).

Equipment type	Base case	Dredging 20% solid slurry		Dredging 30% solid slurry	
	"few fines"	"few fines"	"lot of fines"	"few fines"	"lot of fines"
Ore Processing					
Crusher I [mm]	200				
Screen I [mm]	30	30	30	30	30
Crusher II [mm]	30	30	30	30	30
Screen II [µm]	500	500		500	
Rod Mill [µm]					
Cyclone [µm]	212	212	212	212	212
Ball mill [µm]					
Thickener [%]	65	65	65	65	65
Additional ore treatment					
Clarifier OR					
Sand filter OR					
Centrifuge OR					
Filter press					
Complex waste Treatment					
Option 1					
Option 2					
Option 3					
Option 4					
Simple waste Treatment					
Option 5					
Option 6					
Option 7					

Option 1	Screen + Cyclone + Thickener + Clarifier
Option 2	Screen + Cyclone + Thickener + Sand filter
Option 3	Screen + Cyclone + Thickener + Filter press
Option 4	Screen + Cyclone + Thickener + Centrifuge
Option 5	Decantation basin
Option 6	Screen + Cyclone + Sand filter
Option 7	Screen + Cyclone + Drainage

	Removal of equipment
	Not necessary treatment
	Necessary treatment
	Eventually needed treatment

Table 41: Comparison table of equipments

Equipment type	Dredging	
	"few fines"	"lot of fines"
Additional ore treatment		
Clarifier OR		
Sand filter OR		
Centrifuge OR		
Filter press		
Complex waste Treatment		
Option 1		
Option 2		
Option 3		
Option 4		
Simple waste Treatment		
Option 5		
Option 6		
Option 7		

Efficiency	Very favoured equipment combination
Operability	Possible equipment combination

Table 42: Comparison table of treatment scenarios

The ore treatment arrangement for dredged slurry gives advantages compared to a traditional treatment scenario. That means the slurry that contains only material smaller than 200 mm allows for removing the primary crusher. For the scenario with much fine material even the screen II and the rod mill could be potentially eliminated. The secondary crusher and the ball mill would have a lower utilisation for the “lot of fines”-scenario, what would have an impact on the required size of the ball mill and maintenance requirement on the crusher. In both cases, energy consumption would be lowered. At the same time, slurry with much fine material would require an additional water treatment. The costs for additional water treatment machines could be compensated by the removal of the equipments from the main flow.

Dredged slurry demands an additional treatment of the waste material. A scenario with few fines demands a simple waste treatment. The most common form would be the usage of a decantation basin which represents the waste dump at the same time. The water coming out of the basin would further have to be treated. The waste treatment options with combinations of screen, cyclone and sand-filter or screen, cyclone and drainage are both solutions which are operable. It has to be taken into account that one single screen would not suffice because of the very high slurry flow. A multi-deck screen would be needed with approximately 12 screens of 3*6 m². The same screen unit would be needed for a waste scenario with much fine material that demands complex waste treatment. The most efficient unities for treating the water at the overflow of the cyclone would be a combination of thickeners and clarifiers. The combination of thickener and centrifuge has to be avoided, because centrifuges do not guarantee clear water at the overflow. The combinations of thickener and sand-filter or thickener and filter-press are both operable possibilities (Table 42).

5 Estimate of capital and operating costs

A dredging mining site where the mined product is slurry is not only compared with a traditional mining scenario concerning capital and operating costs for the mining machines, but also treatment equipments have to be compared.

The capital and operating costs of the equipment were taken from the “Mine and Mill Equipment Costs” guide dating from 2008. Therefore, costs for repair labor, diesel, lubricants and electric power were adapted to Bakouma conditions. Parts costs were doubled. These modifications change the values for the operating cost (Table 43).

	Bakouma				MME	
Repair labor	10	$/h	34,0	%	29,4	$/h
Diesel	1,23	$/l	114,1	%	1,078	$/l
Lubricants	3,48	$/l	99,9	%	3,482	$/l
Electirc power	0,246	$/kWh	337,0	%	0,073	$/kWh
Parts			200	%		

Table 43: Costs of resources in Bakouma compared to specifications from the Mine and Mill Equipment Guide 2008

5.1 Capital and operating costs for traditional mining

Several options of equipment for classical open pit mining were already selected. When using option 1 for the removal of the overburden and option 2.2 for ore mining (see Table 44), the average operating costs for classical mining in an open pit were defined by Areva to be approximately 1.15 $/t for a ratio of waste to ore of 10.5. The average operating cost per t ore were calculated to be 13.32 $/t ore.

	Operating costs [$/bcm]						Total operating costs		Capital costs
	Mining & Loading		Transport (max 2.5km)				[$/bcm]	[$/t]	[$]
Cuirass	Bull	Backhoe Excavator	Trucks CAT 730				3,08	1,41	5.140.000
	0,55	0,87	1,66						
Overburden OPTION 1	Backhoe Excavator 5m³		Trucks CAT 730				2,53	1,16	13.035.000
	0,87		1,66						
Overburden OPTION 1.1	Backhoe Excavator 6m³		Trucks CAT 730				2,45	1,12	12.411.200
	0,89		1,57						
Overburden OPTION 2	Backhoe Excavator		Mobile Crusher	Portable Conveyor	Shiftable Conveyor	Stacker	2,14	0,98	13.503.040
	0,87		0,45	0,24	0,44	0,15			
Overburden OPTION 2.2	Backhoe Excavator		Mobile Sizer	Portable Conveyor	Shiftable Conveyor	Stacker	2,22	1,01	20.589.469
	0,87		0,53	0,24	0,44	0,15			
Overburden OPTION 3	BWE		Trucks CAT 730				1,76	0,81	3.424.000
	0,29		1,48						
Overburden OPTION 4	BWE		Portable Conveyor	Shiftable Conveyor		Stacker	1,11	0,51	8.063.040
	0,29		0,24	0,44		0,15			
Ore mining OPTION 1	Backhoe Excavator		Trucks CAT 730				2,78	1,27	5.571.500
	0,93		1,85						
Ore mining OPTION 2	Backhoe Excavator		Mobile Crusher	Portable Conveyor	Shiftable Conveyor	Stacker	2,32	1,06	9.584.540
	0,93		0,45	0,24	0,44	0,27			
Ore mining OPTION 2.2	Backhoe Excavator		Mobile Sizer	Portable Conveyor	Shiftable Conveyor	Stacker	2,40	1,10	14.308.826
	0,93		0,53	0,24	0,44	0,27			

Table 44: Capital and operating costs for traditional mining

In any case of the dredging scenarios, a precedent classical mining of the deposits by truck and shovel is needed. The first reason is that a Beaver Cutter Suction Dredger requires a minimum water depth of 5 m. The water table at the Bakouma deposits is at 10 m under the surface, which requires mining of at least 15 m with classical mining. The already selected equipments for the removal of the overburden could also be chosen for the mining of the first 15 m for the dredging scenario.

5.2 Capital and operating costs for BCSD

The mine planning in chapter 3.4 showed that 1 unique BCSD would not be enough to do the total production. For our chapters 3.4.1 and 3.4.3 describing mining scenarios, we used 4 dredgers (Table 45) with different capacities. For the chapters 3.4.2 and 3.4.4 describing mining scenarios that accounts losses, we used 3 BCSD (Table 46) with different capacities. This results in different capital costs. The installed power of smaller dredgers was calculated over the known installed power of the BCSD 6525 C (Beaver 1) and the relation of their required capacities. The investment costs were calculated over the formula:

$$Inv = (hp)^{\frac{2}{3}} \cdot k$$

$$k = 0.066173$$

	Total installed power [hp]	Capacity [t/h]	Capital costs [M$]
Beaver 1	3672	1860	15,75
Beaver 2	217	110	2,39
Beaver 3	1554	787	8,88
Beaver 4	111	56	1,52
TOTAL Capital Costs			28,54

Table 45: Capital cost calculation for mining scenario without losses

	Total installed power [hp]	Capacity [t/h]	Capital costs [M$]
Beaver 1	3672	1860	15,75
Beaver 2	1554	787	8,88
Beaver 3	328	166	3,15
TOTAL Capital Costs			27,77

Table 46: Capital cost calculation for mining scenario with losses

Operating costs for BCSD were calculated in chapter 3.6.1. For our further calculations and for the comparability to traditional mining we took an operating cost of 2.5 $/t into account and an operating cost per t ore of 26.4 $/t ore. These are the operating costs which were obtained by using fuel as energy source (Table 47).

OPERATING COSTS FOR DREDGING		Fuel	
BCSD	TOTAL per m³	5.0	$/m³
	TOTAL per ton	2.5	$/t
	TOTAL per ton ore	26.4	$/t

Table 47: Operating costs for BCSD

5.3 Capital and operating costs for treatment scenarios

The in chapter 4 defined treatment scenarios and the definition of the equipment sizes, allowed for the calculation of their needed quantities. Capital costs for these equipments were taken out of the Mine and Mill Equipment Guide 2008 with exception of specifications for screen, sand filter, and crusher, where values were defined by Areva. The costs for the hydro-cyclone are the average of two standard hydro-cyclones with the same maximum flow (Table 48 and 49). Operating costs for all equipments were adapted to conditions in Bakouma as described in chapter 5.

Equipment ore treatment	Capacity/Size		Capital costs		Operating costs Bakouma	
Screen	3*6	m	50.00	k$	3.00	$/h
Crusher	213	t/h	990.00	k$	97.05	$/h
Rod mill	160	t/h	1 628.00	k$	378.57	$/h

Table 48: Capital and operating costs for removed equipment from ore treatment

Additional treatment equipment	Capacity/Size		Capital costs		Operating costs Bakouma	
Screen	3*6	m	50.00	k$	3.00	$/h
Hydrocyclone	9 842	l/min	21.60	k$	0.36	$/h
Thickener	121	m	3 700.00	k$	213.41	$/h
Clarifier	121	m	3 700.00	k$	213.41	$/h
Sand filter	5	m	1 600.00	k$	27.00	$/h
Filter press	200	t/h	683.00	k$	29.68	$/h
Centrifuge	295	t/h	146.50	k$	9.71	$/h

Table 49: Capital and operating costs for additional ore and waste treatment equipments

Dredging allows reducing crushing and sometimes also milling and separating equipments for the ore treatment, compared to a classical treatment arrangement (Table 50). For a scenario with much fine material it demands additional equipment for the ore treatment to clean the water. Furthermore the produced waste slurry demands treatment which results in additional needed equipments (Table 52). The reduction of equipment (Table 51) on one hand and the addition of other equipment (Table 53 and 54) on the other hand were compared for the different treatment scenarios. For the calculations, we took a average flow of 20 to 30% solids.

Number of removed equipment	Ore treatment	
	Few fines	A lot of fines
Crusher	1	1
Rod mill		1
Screen		2

Table 50: Number of removed equipments from ore treatment

Ore treatment removed equipment	Capital Costs [k$]		Operating costs [$/t]
	Few fines	A lot of fines	
Crusher	- 990.00	- 990.00	- 0.5882
Rod mill		- 1 628.00	- 2.3671
Screen		- 100.00	- 0.0364
TOTAL	- 990.00	- 2 718.00	- 2.99

Table 51: Reduced capital and operating cost through removed equipment

Additional equipment [needed number]			Ore treatment		Waste treatment	
			Few fines	A lot of fines	Few fines	A lot of fines
Screen					12	12
Hydrocyclone					10	10
Thickener						5
Option	1	**Clarifier**		1		5
	2	**Sand filter**		2	10	10
	3	**Filter press**		1		8
	4	**Centrifuge**		1		5

Table 52: Number of added equipment to ore and waste treatment

Ore treatment additional equipment	Capital Costs [k$]		Operating costs [$/t]
	Few fines	A lot of fines	
Clarifier		3 700.00	0.674
Sand filter		3 200.00	0.022
Filter press		683.00	0.148
Centrifuge		146.50	0.033

Table 53 : Additional capital and operating costs of ore treatment

Waste treatment additional equipment			Capital costs [k$]		Average operating costs [$/t]
			Few fines	A lot of fines	
Screen			600	600	0.004
Hydrocyclone			216	216	0.002
Thickener			-	18 500	0.674
Option	1	Clarifier	-	18 500	0.674
Option	2	Sand filter	16 000	16 000	0.022
Option	3	Filter press	-	5 464	0.148
Option	4	Centrifuge	-	733	0.033
Total Option 1				37 816	1.354
Total Option 2			16 816.00	35 316	0.702
Total Option 3				24 780	0.828
Total Option 4				20 049	0.713

Table 54: Additional capital and operating costs for waste treatment

Option 4 with centrifuges was eliminated from further calculations because this system does not guarantee efficient water cleaning. The most efficient system would be option 1 with the clarifiers. These clarifiers offer the highest capital and operating costs. That is why we had also to eliminate option 1. For the final comparison option 3 seems to be the most reasonable because of the smaller capital costs compared to option 2.

The removal of equipments from the ore treatment can equal capital and operating costs of additional water cleaning equipments for the ore treatment in the scenario with much fine material. The combination of ore and waste treatment does increase capital and operating costs.

5.4 Comparison of mining scenarios

Dredging has compared to traditional mining more important capital and operating costs.

	Capital cost mining [k$]	Capital cost ore treatment [k$]	Capital cost waste treatment [k$]	Operating cost mining [$/t]	Operating cost mining [$/t ore]	OPEX mining incl. Δ treatment cost [$/t waste]	OPEX mining incl. Δ treatment cost [$/t ore]	Total capital costs [k$]
Traditional mining	27.343,83	-	-	1,15	13,32	1,15	13,32	27.344
Dredging	28.541,80	- 2.035,00	24.780,00	2,50	26,40	3,33	23,56	51.287
Dredging with losses	27.772,18	- 2.035,00	24.780,00	2,50	26,40	3,33	23,56	50.517

Table 55: Comparison of mining scenarios

	Dredging	Traditional	Δ Costs
CAPEX [k$]	51.287	27.344	23.943
OPEX [$/t ore]	23,6	13,3	10,2

Table 56: Difference of capital and operating costs

Dredging itself is not more expensive than a traditional open pit mining. The additional cost is due to waste treatment (Table 55). If it would not be clay but sandy material with a better settle-behaviour, the solution (Table 56) would be totally different. Assumptions about waste processing appear to be essential. If Areva wants to continue to explore the dredging solution, this point has to be studied in details.

6 Conclusions

The hydrological situation in Bakouma would require pumping huge quantities of water for a classical open pit mining. Underwater dredge mining of the uranium deposits in Bakouma was studied because its technique avoids pumping the water. Two principal systems were studied: Beaver Cutter Suction Dredgers (BCSD) and Underwater Mining Vehicles (UMV).

A first economical approach showed that UMVs are very expensive concerning capital and operating costs. One underwater mining vehicle system with two UMVs has only two thirds the capacity of a BCSD 6525 and has capital costs which are 4.3 times higher. Operating costs are 1.4 times higher. That is why UMVs were eliminated at an early stage from a further study.

Underwater dredge mining produces slurry with 20% to 30% of solids. Near the solids one BCSD 6525 C would pump approximately 1133 to 1942 l/s of water. The inflow of water in the pit is assumed to be between 750 and 3000 l/s. Without water-reject dredge mining risks to dry out the pit. It was assumed that BCSDs could manage the water level in the pit for staying in their required working height.

The introduction of BCSDs into a detailed mine planning showed that the principal organisation would be a worst-case-mining level by level. A stabilization of the ore production over the whole lifetime of the mine is hardly possible because of the heterogeneity of ore in the horizontal layers of the pit. Mining of thinner layers allowed a better distribution of the BCSDs over time and resulted in a more regular production output, than mining over more important working heights. Beavers with a length of 47.2 m have reduced mobility. In the pit, they have to mine one strip after another. An irregularly mine organization would hardly be possible. The transport from one pit to the next takes long periods of time which results in production stops. Once reinstalled in a pit, the BCSDs cannot be returned back to an earlier pit. BCSDs were concerning capital and operating costs in a comparable range with traditional mining equipments.

Dredged slurry with 20 to 30% of solids has an impact of ore processing and waste elimination. The treatment arrangement of ore slurry allows eliminating a primary crusher, because the mined material will already be smaller than 200 mm. In case of much fine material it would further be possible to eliminate sizing and milling units, which would have a positive impact on capital and operating costs of the ore treatment. Slurry, which contains a lot of fine material, demands additional treatment to clean the water before recycling it into the processing plant or dumping it into nature. The elimination of ore treatment equipments could balance the additional cost for water cleaning. A dredging scenario demands further treatment of waste slurry, which has to be dewatered before bringing it to a waste dump. Enormous quantities of overflow water have to be cleaned, which demand many large treatment unities. The additional capital costs for waste treatment almost covers the capital costs for traditional mining.

Underwater dredge mining compared to traditional mining is in case of Bakouma no favourable solution, mainly because of huge additional costs for waste treatment. Total capital costs for a dredging scenario (incl. treatment equipments) are 2.3 times higher than capital cost for traditional mining. Operating costs per t mined material are 2.2 times higher and operating costs per t mined ore are 1.8 times higher. In a conclusion dredging presents neither an interest from an economical point of view, nor from a technical point of view.

Dredging could be favourable when the solids in the slurry would have a better settle-behaviour. This would result in less and smaller water cleaning equipments that represent the main cost difference between dredge mining and traditional mining. An adapted geometry of the deposits could further have a favourable impact for an underwater dredge mining site.

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